

NON-FERROUS EXTRACTIVE METALLURGY

SUBJECT CODE: RMM5D006

COURSE: B. TECH

SEMESTER: 5th



LECTURES NOTE BY:
Mr. Rahul Ranjan Mohanta
(Assistant Professor)

**DEPARTMENT OF METALLURGICAL AND MATERIALS ENGINEERING
PARALA MAHARAJA ENGINEERING COLLEGE, BERHAMPUR**

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5th Semester

Non-Ferrous Extractive Metallurgy

Module I:

(8 Hours)

Fundamentals of Unit processes involved in Metal Extraction. Thermodynamic considerations and process selection in Pyro-metallurgical extraction of metals. Kinetics of leaching of ores; effect of various operating variables on leaching process; bioleaching.

Module II:

(8 Hours)

Principles involved in Electro-metallurgical extraction of metals.

Extraction of metals from oxide ores (Sn, Mg).

Extraction of metals from Sulphide ores (Cu, Ni, Pb and Zn)

Module III:

(8 Hours)

Extraction of metals through the halide route (Ti and Zr)

Refining involves oxidation, chemical transport reactions, zone refining, distillation, etc.

Ion exchange and solvent extraction processes and their application in extraction processes (Zr, V, Th, Nb, etc)

Module IV:

(8 Hours)

Electrowinning and Electro refining of metals:

a) From aqueous salts (Cu, Ni, Au, Ag)

b) From fused salts (Al and Mg)

Module V:

(8 Hours)

Sustainable Chemical Metallurgy, Recycling and Recovery from waste.

Environmental pollution and its address related to various metal extraction processes in general.

Books:

- [1] Extraction of Non Ferrous Metals by H.S.Ray, R.Sridhar & K.P.Abraham, Affiliated East – West Press, New Delhi
- [2] Extraction and Refining of Metals, by C. Bodsworth, CRC Press
- [3] Metallurgy of Non Ferrous Metals by W.H.Dennis, Pitman.
- [4] Principles of Extractive Metallurgy, by T. Rosenquist, McGraw hill, 1974
- [5] Rare Metal Extraction by Chemical Engg. Tech. by W.D.Jamrack, Pergamon Press, Oxford.

1. INTRODUCTION TO NON-FERROUS METAL EXTRACTION

1.1 GENERAL PRINCIPLES OF EXTRACTION OF METALS FROM OXIDES & SULPHIDES:

In an extraction process, a specific amount of energy is required to reduce or release a metal ion present in a compound (solid or liquid) in association with other elements and to collect the metal in its elemental form.

In nature, metals generally occur either in free form or in combined form. Generally, low reactive metals show little affinity towards air, moisture, carbon dioxide, or other non-metals present in nature. Such metals may remain in the elemental or native (free) state in nature. Such metals are called *Noble Metals* as they show the least chemical reactivity. For example, gold (Au), silver (Ag), mercury (Hg), and platinum (Pt), etc. Whereas, most metals are active to combine with air, moisture, carbon dioxide, and non-metals like oxygen, sulfur, halogens, etc. to form their compounds, like oxides, sulfides, carbonates, halides, and silicates. i.e., in nature, they occur in a combined state.

A naturally occurring material in which a metal or its compound occurs is called a *mineral*. A mineral from which a metal can be extracted economically is called an *ore*. An ore is a mineral in which a metal is present in appreciable quantities and from which the metal can be extracted economically. The main active substances present in nature, especially in the atmosphere are oxygen and carbon dioxide. In the earth's crust, sulfur (S) and silicon (Si) are found in large quantities. Most active metals are highly electropositive and therefore exist as ions. It is for this reason that most of the important *ores* of these metals occur as (i) oxides (ii) sulfides (iii) carbonates (iv) halides and (v) silicates. Some sulfide ores undergo oxidation by air to form sulfates. This explains the occurrence of sulfate ores. Ores are invariably found in nature in contact with rocky materials. These rocky or earthy impurities accompanying the ores are termed *gangue* or *matrix*.

During metal extraction, the bulk of the metal is separated from the impurities present in the ore by using a system that invariably comprises more than one phase. The metal enters the metallic phase while the impurities enter the non-metallic phase or phases. The driving force for the reactions involved in metal extraction depends on the differences in the chemical potentials of the chemical elements. During a chemical reaction, an element tends to move from a high to a low chemical potential state. In each extraction process, the basic aim is to create conditions conducive to such a transfer, which facilitates the extraction of a metal or its compound.

In metal extraction, a vast majority of metallurgical reactions are made to take place at elevated temperatures because then the ore compounds become relatively unstable, facilitating the release of the metal. This is the basis of pyrometallurgy, in electrometallurgy, the dissociation or decomposition of a compound in solution (i. e. reduction of an iron) is brought about by employing electric forces. In hydrometallurgy, the reduction process depends on the judicious manipulation of the chemical reaction taking place in an aqueous solution.

The process of extracting the metals from their ores and refining them is called *metallurgy*. The choice of the process depends upon the nature of the ore and the type of the metal. The metal content in the ore can vary depending on the impurities present and the chemical composition of the ore. Some common steps involved in the extraction of metals from their ores are:

- i. Crushing and pulverization
- ii. Concentration or dressing of the ore
- iii. Calcination and roasting of the ore
- iv. Reduction of metal oxides to free metal
- v. Purification and refining of metal

1.1.1 Crushing and Pulverization:

The ore is generally obtained as big rock pieces. These big lumps of the ore are crushed into smaller pieces by using *jaw crushers* and *grinders*. It is easier to work with crushed ore. The big lumps of the ore are brought in between the plates of a crusher forming a jaw. One of the plates of the crusher is stationary while the other moves to and fro and the crushed pieces are collected below (Fig. 1.1).

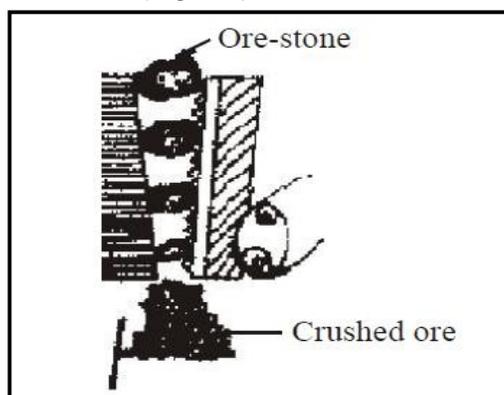


Fig. 1.1: Jaw Crusher

The crushed pieces of the ore are then pulverized (powdered) in a stamp mill shown in (Fig. 1.2). The heavy stamp rises and falls on a hard die to powder the ore. The powdered ore is then taken out through a screen by a stream of water. Pulverization can also be carried out in a ball mill. The crushed ore is taken in a steel cylinder containing iron balls. The cylinder is set into revolving motion. The striking balls pulverize the crushed ore into fine powder.

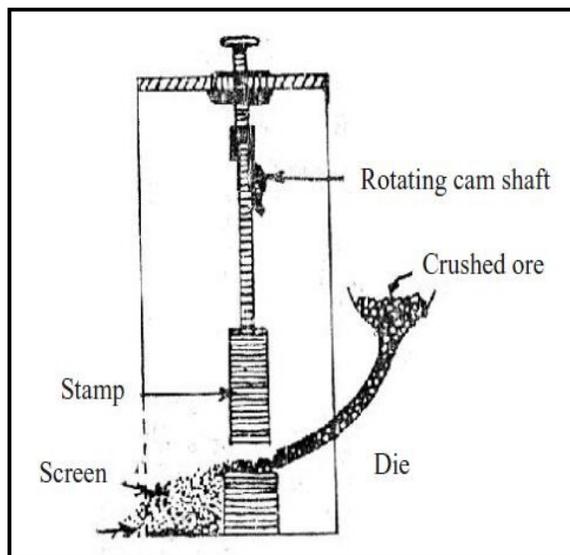


Fig. 1.2: Stamp Mill

1.1.2 Concentration or Dressing of the Ore:

Generally, the ores are found mixed with earthy impurities like sand, clay, limestone, etc. These unwanted impurities in the ore are called *gangue* or *matrix*. The process of removal of gangue from powdered ore is called *concentration* or *ore dressing*. There are several methods for concentrating the ores. The choice of method depends on the nature of the ore. Some important methods are:

1.1.2.1 Gravity separation (Hydraulic washing):

In this method, the light (low specific gravity) earthy impurities are removed from the heavier metallic ore particles by washing them with water. It is, therefore, used for the concentration of heavier oxide ores, like hematite (Fe_2O_3), tinstone (SnO_2), and gold (Au). In this method, as shown in the Fig. 1.3 the powdered ore is agitated with water or washed with a strong current of water. The heavier ore settles down rapidly in the grooves and the lighter sandy and earthy materials (gangue particles) are washed away.

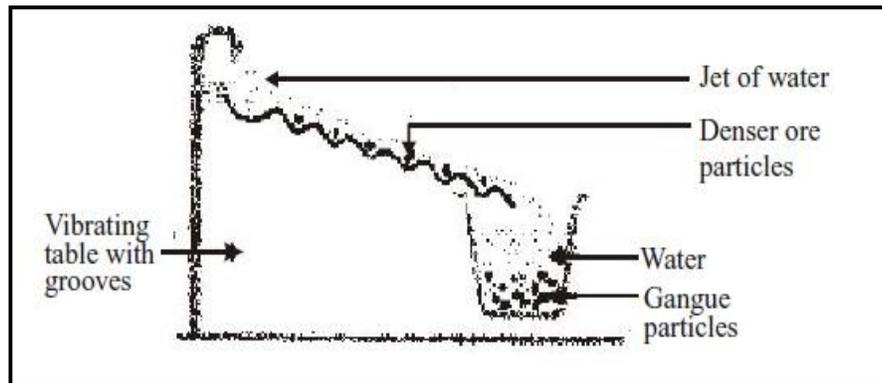


Fig. 1.3: Gravity separation (Hydraulic washing)

1.1.2.2 Magnetic Separation Method:

By this method, those ores can be concentrated which either contain impurities that are magnetic or are themselves magnetic. For example, the tin ore, tin stone (SnO_2) itself is non-magnetic but contains magnetic impurities such as iron tungstate (FeWO_4) and manganese tungstate (MnWO_4). The finely powdered ore is passed over a conveyor belt moving over two rollers, one of which is fitted with an electromagnet (Fig. 1.4). The magnetic material is attracted by the magnet and falls in a separate heap. In this way, magnetic impurities are separated from non-magnetic material.

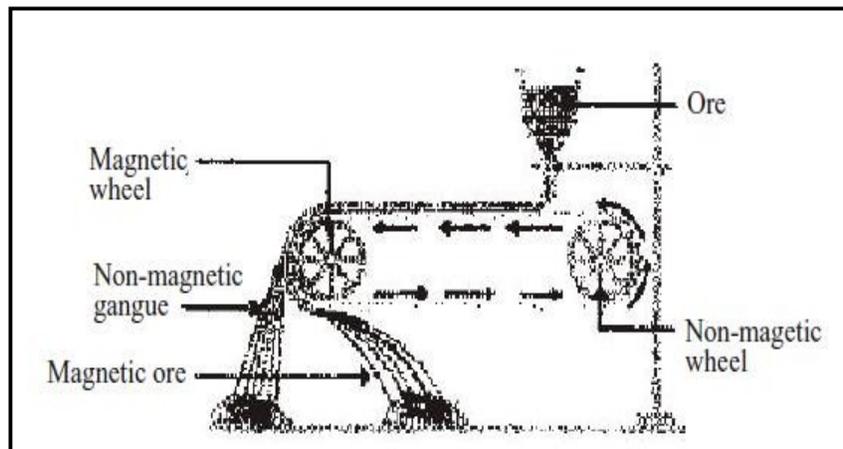


Fig. 1.4: Magnetic Separation

1.1.2.3 Froth Floatation Method:

This method is especially applied to sulfide ores, such as Galena (PbS), Zinc Blende (ZnS), or Copper Pyrites (CuFeS_2). It is based on the different wetting properties of the surface of the ore and gangue particles.

The sulfide ore particles are wetted preferentially by oil and gangue particles by water. In this process, finely powdered ore is mixed with either pine oil or eucalyptus oil. It is then mixed with water. Air is blown through the mixture with great force. Froth is produced in this process which carries the wetted ore upwards with it. Impurities (gangue particles) are left in water and sink to the bottom from which these are drawn off (Fig. 1.5).

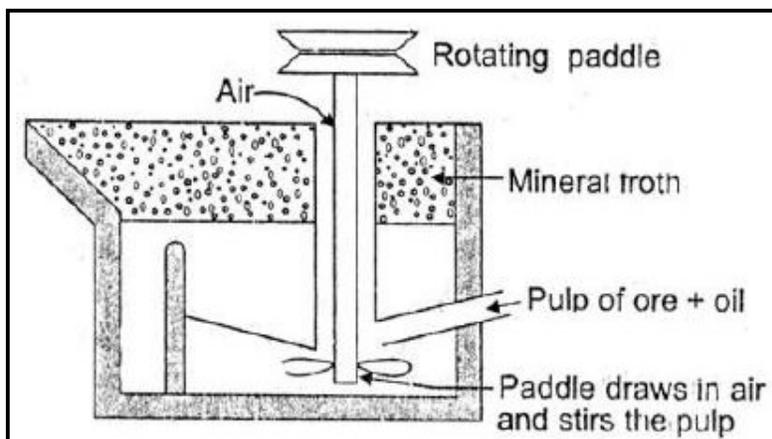
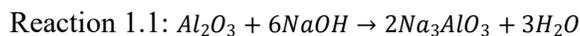


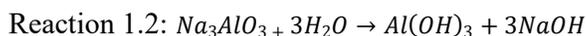
Fig. 1.5: Froth Floatation

1.1.2.4 Chemical method:

In this method, the ore is treated with a suitable chemical reagent which dissolves the ore leaving behind insoluble impurities. The ore is then recovered from the solution by a suitable chemical method. This is applied for the extraction of aluminum from bauxite ($\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$). Bauxite is contaminated with iron (III) oxide (Fe_2O_3), titanium (IV) oxide (TiO_2) and silica (SiO_2). These impurities are removed by digesting the powdered ore with an aqueous solution of sodium hydroxide at 420 K under pressure. Aluminium oxide dissolves in sodium hydroxide, whereas, iron (III) oxide, silica, and titanium (IV) oxide remain insoluble and are removed by filtration.



Sodium aluminate (Na_3AlO_3) is diluted with water to obtain a precipitate of aluminum hydroxide. It is filtered and ignited to obtain pure alumina.



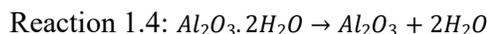
1.1.3 Calcination and Roasting of the Ore:

The concentrated ore is converted into metal oxide by calcination or roasting.

1.1.3.1 Calcination:

Calcination involves heating the concentrated ore in a limited supply of air so that it loses moisture, water of hydration, and gaseous volatile substances. The ore is heated to a temperature so that it does not melt. Two examples of calcination are given below:

- (i) Removal of water of hydration



- (ii) Expulsion of CO₂ from carbonate

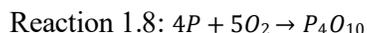
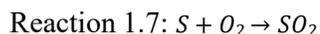


1.1.3.2 Roasting:

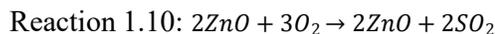
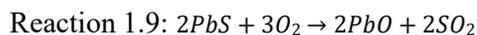
Roasting is a process in which the concentrated ore is heated in a free supply of air at a temperature insufficient to melt it. The following changes take place during roasting:

- (i) Drying of the ore.

(ii) Removal of volatile impurities like arsenic, Sulphur, phosphorus, and organic matter.



- (iii) Conversion of the sulfide ores into oxides



Calcination and roasting are generally carried out in a reverberatory furnace or in a multiple hearth furnaces.

1.1.4 Reduction of Metal Oxides to Free Metal:

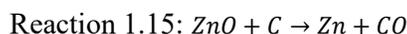
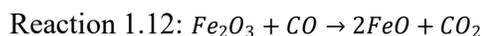
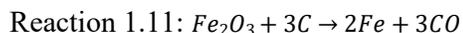
This process is carried out after calcination or roasting of ores. This process called *smelting*, the oxide ores are converted into the metallic state by reduction.

1.1.4.1 Smelting:

Smelting is a process in which the oxide ore in the molten state is reduced by carbon or other reducing agents to free metal.

(i) *By using carbon as a reducing agent:*

This method is used for the isolation of iron, tin, and zinc metals from their respective oxides. The oxide ores are strongly heated with charcoal or coke. Reduction occurs by the action of carbon and/or carbon monoxide which is produced by the partial combustion of coke or charcoal.



Although the ore has been concentrated in an earlier step, it is still contaminated with some gangue material which is finally removed in the reduction process by the addition of *flux* during smelting.

Flux is a chemical substance that combines with gangue at higher temperatures to form an easily fusible material called *slag* which is not soluble in the molten metal. Fluxes are of two types:

Basic Flux:

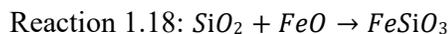
On heating, limestone is converted into calcium oxide used as basic flux which combines with acidic impurities like silica in metallurgy of iron and forms fusible calcium silicate ($CaSiO_3$).



(Basic flux) (Acidic gangue) (Slag)

Acidic Flux:

SiO_2 is used as an acidic flux to remove the basic impurity of FeO in the metallurgy of Cu.



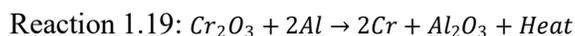
(Acidic flux) (Basic gangue) (Slag)

The fusible slag, such as calcium silicate formed during smelting floats over the molten metal and is thus easily removed. Another advantage is that the slag provides a covering to the molten metal thus preventing it from getting oxidized by air.

(ii) Another reducing agent:

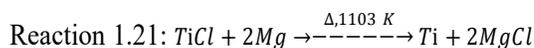
Oxide ores which cannot be reduced by carbon or metals that show affinity to carbon by forming metal carbides are reduced by reducing agents like aluminium, sodium, magnesium, or hydrogen. *Oxides* like chromium oxide (Cr_2O_3) or manganese oxides (Mn_3O_4) are reduced by aluminum powder in a highly exothermic reaction.

This process is known as *Goldschmidt's Alumino-thermite reduction method*.

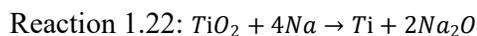


Heat is generated in the process due to the formation of Al_2O_3 which is a highly exothermic reaction.

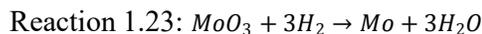
Titanium is obtained by the reduction of TiCl_4 (produced by the action of carbon and chlorine on TiO_2) by Mg in an inert atmosphere of argon (Kroll process).



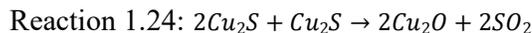
Titanium can also be obtained by the reduction of TiO_2 by sodium



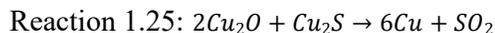
Tungsten and molybdenum can be obtained by the reduction of their oxides by hydrogen,

*(iii) Self-reduction:*

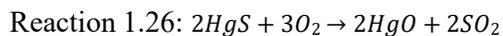
This is applied to the sulfide ores of copper, mercury, and lead. The ores are heated in air, and a part of these sulfide ores is changed into the oxide or sulfate which then reacts with the remaining part of the sulfide ore to give the metal and Sulphur dioxide. The reactions showing their extraction are given below:



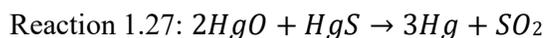
Copper glance

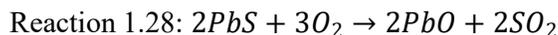


Copper produced at this stage is called *Blister copper*. The evolution of sulfur dioxide produces blisters on the surface of solidified copper metal.

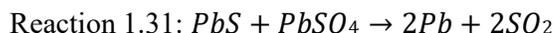
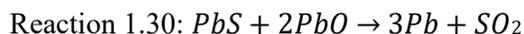


Cinnabar





Galena



1.1.4.2 Reduction of Concentrated Ores by Other Methods:

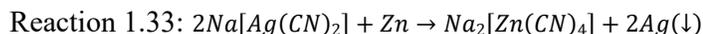
Some metals cannot be obtained from their ores by using common reducing agents such as C, CO, H₂, etc. Other methods of reduction are used for such cases.

(i) *Reduction by precipitation:*

Noble metals like silver and gold are extracted from their concentrated ores by dissolving metal ions in the form of their soluble complexes. The metal ions are then regenerated by adding a suitable reagent. For example, concentrated argentite ore (Ag₂S) is treated with a dilute solution of sodium cyanide (NaCN) to form a soluble complex:



This solution is decanted off and treated with zinc to precipitate silver,



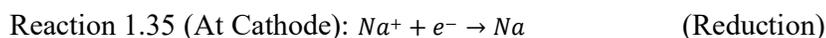
(ii) *Electrolytic Reduction:*

Active metals like sodium, potassium aluminum, etc., are extracted by the electrolysis of their fused (*molten*) salts. For example, sodium is obtained by the electrolysis of fused sodium chloride (Down's process). The reactions taking place in the electrolytic cell are:

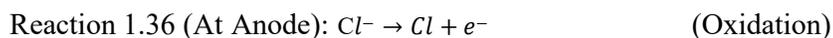


Na⁺ ions move towards the cathode and Cl⁻ ions move towards the anode.

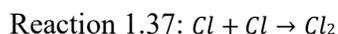
The following reactions take place at the electrodes:



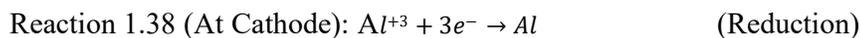
(Negative electrode) (metal)



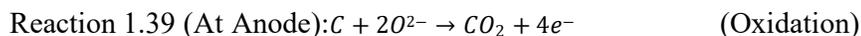
(Positive electrode)



Aluminium is extracted from molten alumina (Al_2O_3) by electrolysis. The melting point of alumina is quite high (2323 K) which is inconvenient for electrolysis. It dissolves in molten cryolite (Na_3AlF_6) at around 1273 K. The reactions which take place in the cell are:



(Negative electrode) (metal)



(Positive electrode)

1.1.5 Purification and Refining of Metal:

Except in the electrolytic reduction method, metals produced by any other method are generally impure. The impurities may be in the form of (i) other metals (ii) unreduced oxide of the metal (iii) non-metals like carbon, silicon, phosphorus, Sulphur etc. and (iv) flux or slag. Crude metal may be refined by using one or more of the following methods:

(i) *Liquation*: Easily fusible metals like tin, lead etc. are refined by this process. In this method, the impure metal is poured on the sloping hearth of a reverberatory furnace (Fig.1.6) and heated slowly to a temperature little above the melting point of the metal. The pure metal drains out leaving behind infusible impurities.

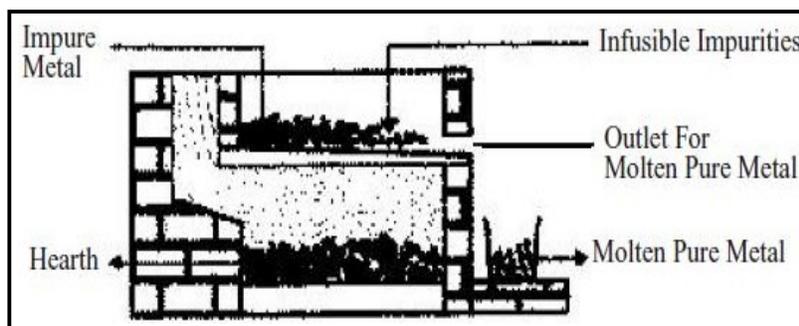


Fig. 1.6: Liquation

(ii) *Poling*: Poling involves stirring the impure molten metal with green logs or bamboo. The hydrocarbons contained in the pole reduce any metal oxide present as impurity. Copper and tin are refined by this method (Fig. 1.7).

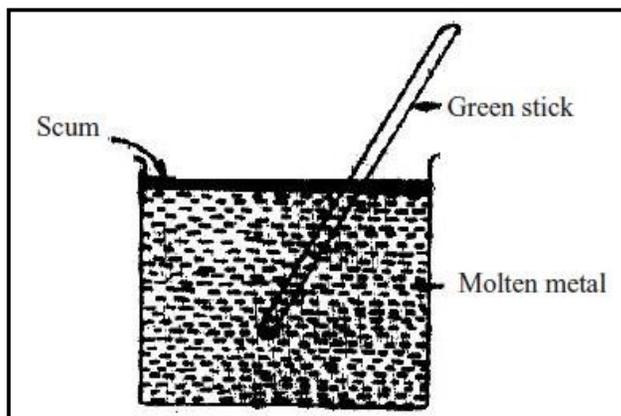


Fig. 1.7: Poling

(iii) *Distillation*: Volatile metals like zinc and mercury are purified by distillation. The pure metal distils over, leaving behind non-volatile impurities.

(iv) *Electrolytic Refining*: A large number of metals like copper, silver, zinc, tin etc. are refined by electrolysis. A block of impure metal is made the anode and a thin sheet of pure metal forms the cathode of the electrolytic cell containing suitable metal salt solution which acts as an electrolyte (Fig. 1.8). On passing current, pure metal deposits at the cathode sheet while more electropositive impurities are left in solution. Less electropositive metals do not dissolve and fall away from the anode to settle below it as *anode mud*.

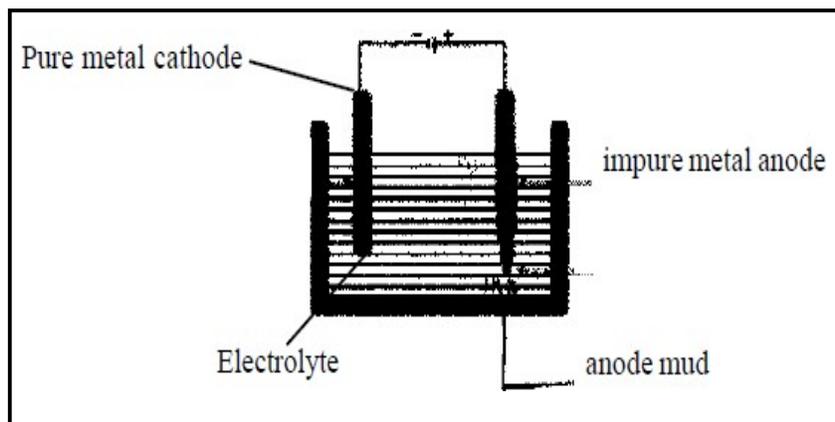
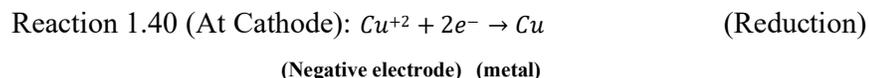


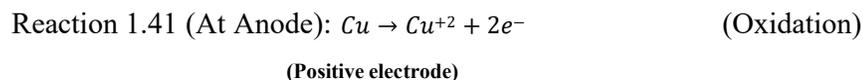
Fig. 1.8: Electrolytic Refining

For example, in the electrolytic refining of crude copper (blister copper), a large piece of impure copper is made anode and a thin piece of pure copper is made the cathode. An acidified solution of copper sulphate is used as an electrolyte. On passing an electric current

of low voltage through the solution copper (II) ions obtained from copper sulphate solution go to the cathode where they are reduced to the free copper metal and get deposited.



An equivalent amount of the metal from the anode dissolves into the electrolyte as Cu^{+2} ions.



As the process goes on, anode becomes thinner while the cathode becomes thicker. The impurities like silver, gold settle down at the bottom of the cell as '*anode mud*'.

1.2 MINERAL RESOURCES OF NON-FERROUS METALS IN INDIA:

The mineral resources of India are broadly classified into three categories as follows

(a) Adequate to Abundant:

Mineral containing Al, Be, Cr, Fe, Mn, Mg, Ti, Zr, Th, and the rare earth metals.

(b) Inadequate:

Minerals containing Cu, Au, Carbon (graphite), Pb, V, Zn, Ni, Cd, U, and Sn etc.

(c) Poor (So far not known):

Minerals containing Bi, B, Co, Hg, Mo, Nb, Tl, P, Ta, Se, S, Sr, Ag, W etc.

The minerals of the 1st category, if properly exploited can not only meet the domestic demand but can also yield a surplus for export. Although the minerals in the 2nd category are insufficient to meet the growing domestic demand, they should be fully exploited. The 3rd category resources are at present inadequate for commercial exploitation.

From the above three categories India is deficient in some of the common metals such as Cu, Zn, Pb, Sn, Co, and Ni, its reserves of Fe, Mn, Mg, Al, and Ti are expensive. Further, India has abundant reserves of strategically important metals namely Th, Zr, and Be. We now discuss the mineral resources of individual elements.

Aluminium (Al): According to reliable sources, the East Coast of India has one of the world's largest deposits of bauxite. Since this deposit is rich in gibbsite and has low content in silica and titania, it is ideal for Bayer process for the extraction of alumina. Currently, India has an installed capacity of 320,000 tons for aluminium products, which is adequate to meet the present demand. Das Gupta (1979) has predicted that after 1990, the whole Europe, including the U.S.S.R., will have to rely largely of Africa and India for obtaining bauxite and alumina.

Antimony (Sb): Commercially exploitable deposits of antimony have been so far located in India, although small deposits of antimony ores have been found in Lahaul and Spiti (Himachal Pradesh). At present, the entire annual requirement, i.e. 500-700 tons, is being met by importing the metal, its ores, and concentrates.

Beryllium (Be): Beryl, the chief ore of Be, occurs in pignatites which is associated with granites and is found in Rajasthan, Tamil Nadu, Jammu and Kashmir, and Bihar. At one time, almost the entire beryl output of India was being exported but the Atomic Energy Establishment (AEE) now handles its production and sale. Although beryllium is not produced on a commercial basis in India, substantial groundwork has been done at Bhabha Atomic Research Centre (BARC), Bombay, in beryllium production technology.

Chromium (Cr): The principal chromite deposits of India are located in Singhbhum district (Bihar), Karnataka, Krishna and Sawantwadi (Maharashtra), and Keonjhar district (Odisha). The chromite reserves of India, although estimated to be limited to about 17 million tons, should be able to meet the domestic requirement for a long period, and a production of 10,000 tons per annum can be sustained for years.

Nickel (Ni) and Cobalt (Co): The Cu ore belt passing through Singhbhum (Bihar) contains also a Ni ore body which may provide 1000 tons of nickel annually. In fact, India's nickel reserves are about 100 million tons, of which 6.5 million tons, averaging 0.85 % nickel, are located at Sukinada (Odisha). Ni has also been found associated, in very minute quantities, with the gold-quartz reefs at Kolar (Karnataka) and uranium ores at Jaduguda (Bihar). The traces of Co have been found along with Cu ores of Khetri and Mn ores of Kalahandi (Odisha).

Cadmium (Cd): Cadmium is recovered as a byproduct from the zinc smelters at Debari (Rajasthan), Vishakhapatnam (Andhra Pradesh), and Alwaye (Kerala). Indigenous sources meet most of the domestic cadmium requirements.

Copper (Cu): About 90% of reserves are spread over Bihar, Rajasthan, and Madhya Pradesh, the most important deposits being located in the Singhbhum copper belt in Bihar. At present, Hindustan Copper Limited (HCL) is the sole producer of Cu in India is able to meet about 42% of country's requirements, i.e. about 23,000 tons in 1977-78.

Gold (Au): India's gold reserves are estimated to be 3.15 million tons (1977); a majority of them located in Karnataka. The gold content of these reserves ranges from 5.65 to 19.98 gm/ton of ore. Minor deposits of gold have also been found in the Hutti gold field (Karnataka). The entire production of gold in India is by the govt. of India's Bharat Gold Mines Limited (BGML) & the Karnataka state-owned Hutti Gold Mines Company. In 1977, the Au production in India was 2854 kg compared with 3132 kg in 1976. This decrease can be attributed to the shrinkage reserves, because of which, the gold production is unlikely to go up in the future, unless, of course, new reserves are found.

Silver (Ag): India has no reserves of silver ores. However, small quantities of silver as a by-product during the indigenous refining of gold, copper, and lead; the major part coming from the lead smelters at Tundoo (Bihar) and the gold mines in Karnataka.

Lead (Pb) and Zinc (Zn): India has about 140 million tons of lead-zinc ores, of which nearly 118 million tons are located in two areas in Rajasthan, namely, the Zawar mines and the Rajapura-Dariba lead-zinc belt. The lead-zinc ores from Zawar mines are beneficiated at the site to produce both zinc and lead concentrate. These concentrates are then processed; the lead concentrate at Tundoo by the Hindustan Zinc Limited (HZL)'s smelter and the zinc concentrate by another HZL smelter at Dabari. In 1977, HZL produced 46,500 tons of zinc concentrate, which could meet about 36% of India's total demand for 1977-78. In 1978, the smelters at Tundoo and Visakhapatnam together produced 10,060 tons of lead.

Manganese (Mn): India is the 3rd largest producer of manganese in the world. The ore deposits presently worked in India are located in Singhbhum, Panchmahals (Gujarat), Balaghat (Madhya Pradesh), Bhandara and Nagpur (Maharashtra), Vishakhapatnam (Andhra Pradesh), Sandur and Shimoga (Karnataka), Bonai, Keonjhar, and Koraput (Odisha), and Banswara (Rajasthan). The total Mn reserves according to the Indian Bureau of Mines (1977), are estimated to be 108 million tons. In 1977, the total production of Mn ore in India was 1,774,000 tons.

Magnesium (Mg): India has extensive deposits of Mg in the form of magnesite (MgCO_3), chiefly found in Chalk Hills of Salem district (Tamil Nadu). On the whole, India's in situ magnesite reserves are estimated to be 524 million tons, of which the existence of 33 million tons has been conclusively established. Apart from Salem district, the other important deposits are found in Idar (Maharashtra), Kurnool (Andhra Pradesh), Tiruchirapalli (Tamil Nadu), Coorg (Karnataka), Dungarpur (Rajasthan), and Singhbhum. It should be noted that in some of these deposits, Mg is present as dolomite ($\text{CaCO}_3 \cdot \text{MgCO}_3$). Widespread reserves of dolomite and magnesium limestones are found in Tamil Nadu, Andhra Pradesh, Bihar, Madhya Pradesh, Rajasthan, and Odisha.

Molybdenum (Mo): Exploration surveys in Andhra Pradesh and Tamil Nadu have shown that primary molybdenite (MoS_2) occurs in the quartz veins that traverse porphyritic granites and, at places, in the granite itself. Mo in a base metal sulphide that is associated with Cu ores is found at Rekha and Bhatin in the Singhbhum district. Preliminary beneficiation of the Mo ores has indicated the recovery during concentration to be 95%.

Tin (Sn): Tin is not extracted on a commercial scale in India although tin deposits have been detected in Ranchi and Haziribagh (Bihar). The entire present requirement of almost 10,000 tons per year is being met by imports.

Titanium (Ti): Abundant quantities of ilmenite ($\text{FeO} \cdot \text{TiO}_2$) are found in the beach sands along a 160 km stretch of the Kerala coast. In fact, a particular 23 km coastal strip contains 35 million tons of ilmenite and 3 million tons of rutile (TiO_2). The ilmenite from Kerala contains a high % (55 %) of titanium. Smaller deposits of ilmenite occur near Tuticorin (Tamil Nadu), Waltair (Andhra Pradesh), and Ganjam district (Odisha). The total reserves of ilmenite in the form of beach sands are estimated to be 133 million tons. Apart from the beach sand deposits, small quantities of massive and crystalline ilmenite are found in Bhagalpur district (Bihar) and Kisengarh and Jodhpur (Rajasthan). The beach sands of Kerala contain a mixture of mineral sands whose specific gravities vary from 2.3 (silica) to 4.9 (monazite).

Tungsten (W): The presence of tungsten minerals has been detected in certain parts of the country, namely, Chandapathan in Bankura district (West Bengal), Agargaon in Nagpur district (Maharashtra), and Degana (Rajasthan); the present production of tungsten comes from the first two sources. The volume of tungsten concentrate produced in 1977 was 43,046 kg, of which Rajasthan accounted for 90% and West Bengal for 10%.

Vanadium (V):

India depends entirely on imports to meet its vanadium requirements although vanadium-bearing titaniferrous magnetite ($\text{FeO} \cdot \text{Fe}_2\text{O}_3$) has been located near Dublabera in Singhbhum district and in Mayurbhanj district (Odisha). Also, a small deposit near Nausahi in Keonjhar district (Odisha) is known to contain vanadiferrous iron ores. The total volume of vanadiferrous magnetite ores in India is estimated to be approximately 26 million tons.

2. EXTRACTION OF NON-FERROUS METALS

2.1 Extraction of Cu in Pyrometallurgical Process:

2.1.1 Details about Cu:

Atomic Number: 29

Atomic Weight: 63.54 amu

Density: 8.94 gm/cm³

Melting Point: 1083⁰ C

Boiling Point: 2595⁰ C

2.1.2 Common Minerals of Cu:

Copper Oxides		Copper Sulphides	
Name	Chemical Formula	Name	Chemical Formula
Cuprite	Cu ₂ O	Chalcocite	Cu ₂ S
Chrysocolla	CuSiO ₃ .2H ₂ O	Chalcopyrite	CuFeS ₂
Malachite	CuCO ₃ .Cu(OH) ₂	Bornite	Cu ₅ FeS ₄
Azurite	2CuCO ₃ .Cu(OH) ₂	Covellite	CuS

2.1.3 Extraction of Cu from Sulphide Ores:

The steps involved in extraction of Cu by conventional routes as concentration, roasting, smelting, converting, and refining at the left side of the flow sheet drawn in fig 1.1 and the newer route at the right side of the flow sheet.

(i) Conventional Route: -

Concentration:

A naturally occurring Cu sulphide contains 0.5-2 % of Cu. To recover this, first, we go for crushing and grinding to liberate sulphide grains from the gangue of average 40 µm sizes.

Then go for froth floatation, where addition of CaO as P^H controls and Xanthate as collector, where we separate Cu sulphide from a mixture of Cu Sulphide, Lead Sulphide and Zinc Sulphide. The Cu Sulphide contains 15-35% Cu, 15-35% Fe, 25-35% S, 3-15% gangue.

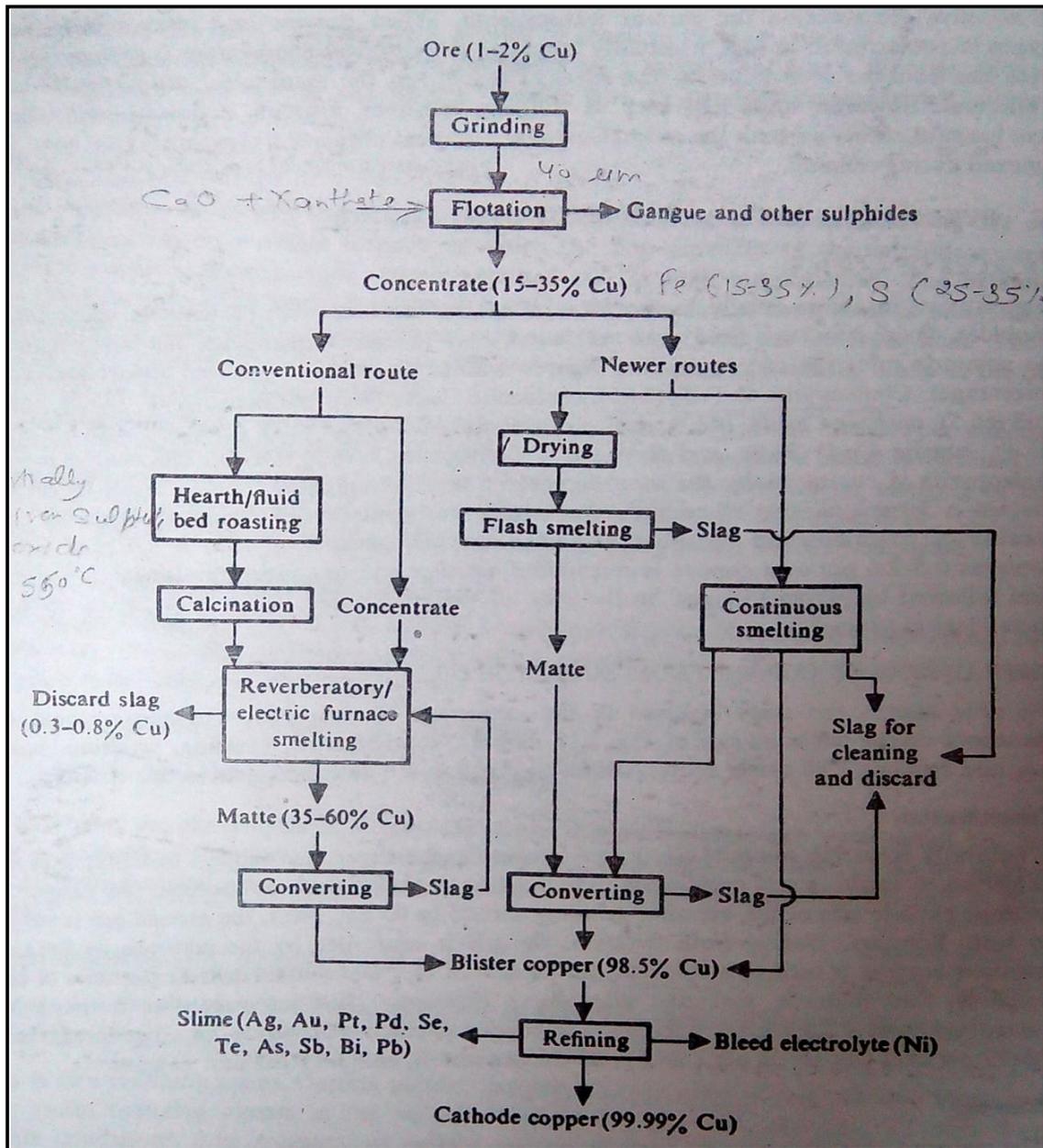


Fig. 2.1: Pyrometallurgical Extraction of Cu from Sulphide Ores

Roasting: The purpose of roasting is to partially oxidize the iron sulphide present in Cu Sulphide from slag in the next i.e. smelting. Here the concentrate contains high iron sulphide and low Cu Sulphide content (<25%).

In conventional plants, roasting is carried out in multiple hearth roasters with the gradual downward movement of feed having a gradual increase in temperature of each hearth by fuel firing as 550°C . Whereas in newer route, roasting is carried out in fluidized bed roaster at about 550°C . The product is SO_2 gas which is finally used for manufacturing for H_2SO_4 .

The principal reactions take place in roasted calcine containing Cu Sulphide and Fe oxides of Fe and mixed sulfates of Fe and Cu given as

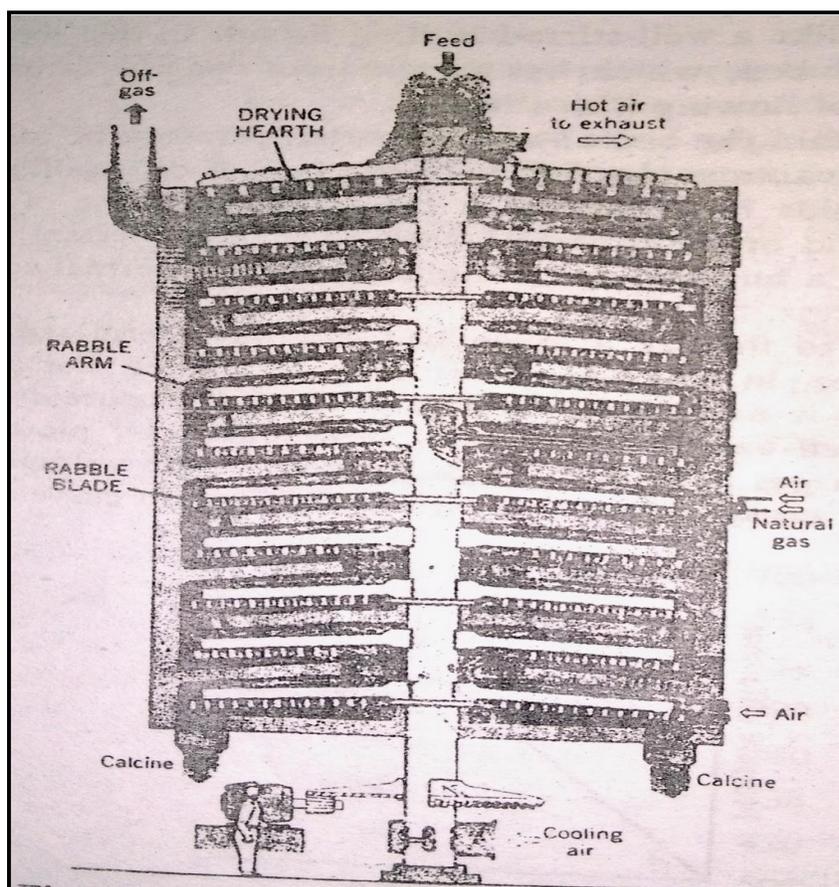
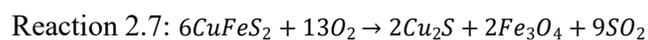
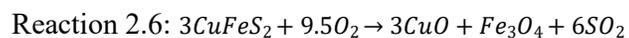
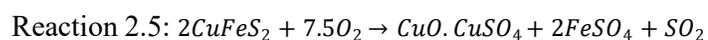
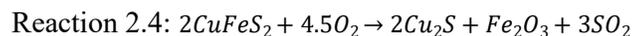
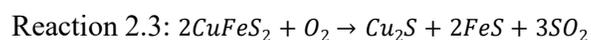
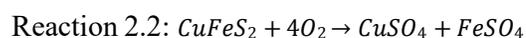


Fig. 2.2: Multiple Hearth Roaster

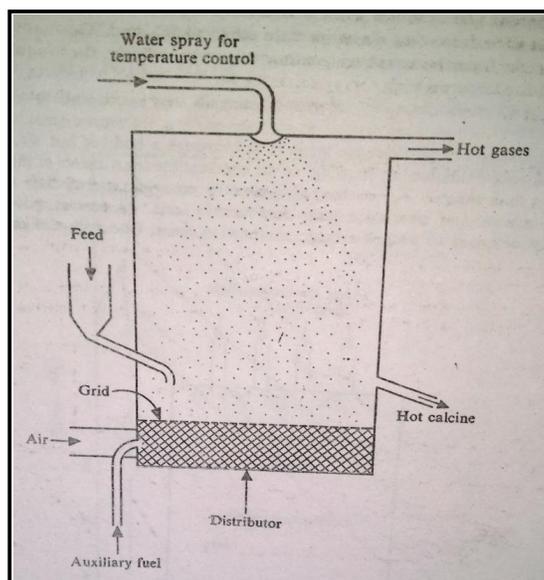


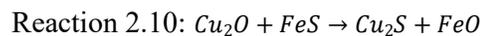
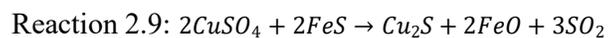
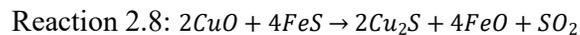
Fig. 2.3: Fluidized Bed Roaster

Smelting:

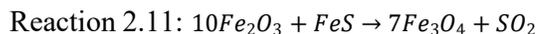
It seeks to separate the metal sulphides in an ore concentrate or calcine from the gangue by charging the furnace with a suitable charge along with flux at about 1250⁰C. During smelting, two separate layers are formed, as slag and matte layers having specific gravity 2.8-

3.8 gm/cm³ and 5.0-5.5 gm/cm³ respectively. If gangue and iron oxides are present in the concentrate/calcine to be smelted combine during smelting to form iron silicate slag. If iron oxide is insufficient the lime is added as flux to form an iron calcium silicate slag. The melting point of slag is approximately 1150⁰C and for fluid slag with a low Cu content, the temperature at about 1250⁰C.

An exchange reaction takes place between oxides and sulphides of Cu and FeS present in the furnace charge.



These reactions occur because O_2 has a greater affinity for Fe than Cu in the Ellingham diagram. The unoxidized iron sulphide reduces the higher oxides of iron to ferrous oxide as



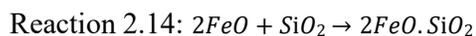
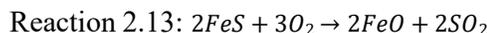
Smelting determines the extent of oxidation of FeS in charge to slag object to produce a matte containing 35-45% Cu, 20-22% S, and 25-35% Fe. Smelting not only minimizes the loss of Cu to slag, but also provides a matte with a sufficient quantity of iron sulphide for use in the next stage as converting.

Smelting conventional route occur in reverberatory furnace and electric furnace, where in newer route flash smelting and continuous smelting takes place. The conventional route, electric furnace because a large volume of combustion gas generation avoided and facilities both recovery of SO_2 and cleaning of furnace gases.

Converting: The purpose of converting is to remove Fe, S, and other impurities from matte. Generally side blown converter used.

The capacity of converter is 100-200 tons, 4m in diameter and 9 m in length. Lined with a layer of Chrome-magnesite refractory about 40 cm thick. The oxidizing atmosphere inside the converter by a supply of O_2 enriched air through the tuyere of 5 c.m. with 40 in number. The total volume of gas is $600 \text{ m}^3/\text{min}$. The product is blister Cu and slag. So, the Cu converting namely the slaging stage and blister formation stage.

(i) Slaging Stage: The iron sulphide present in matte is oxidized and oxide is slagged out by the addition of siliceous flux. The reactions are



Mainly three processes are done as

- Adding freshly obtained matte.
- Blowing of air.
- Slag gets skimmed off by tilting.

The slag contains 2-9% Cu, 40-50% Fe, 20-30% SiO₂, and 1-5% (CaO+MgO). Conventionally, Cu is recovered from the slag produced in the converter by transferring the slag in a molten state to a smelting furnace, where in the case of a newer process, metallic values of the slag are recovered by employing either an electric furnace with provision for cleaning the slag or by slow cooling of slag followed by grinding and floatation.

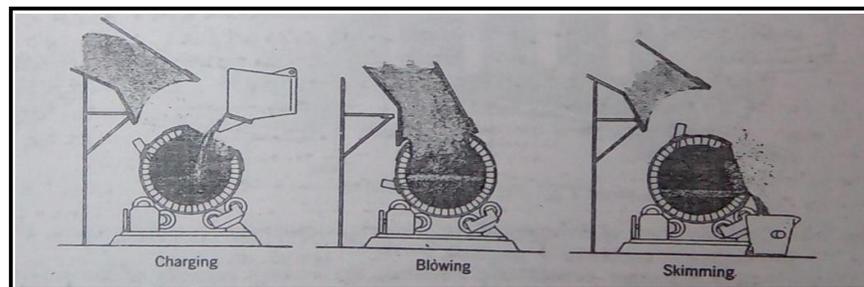


Fig. 2.4: Converter Operation

(ii) **Blister Formation Stage:** After slagging converter contains Cu₂S known as white metal because of its appearance. In the blister Cu formation stage, Cu₂S is oxidized to form Cu by a combination of reactions.

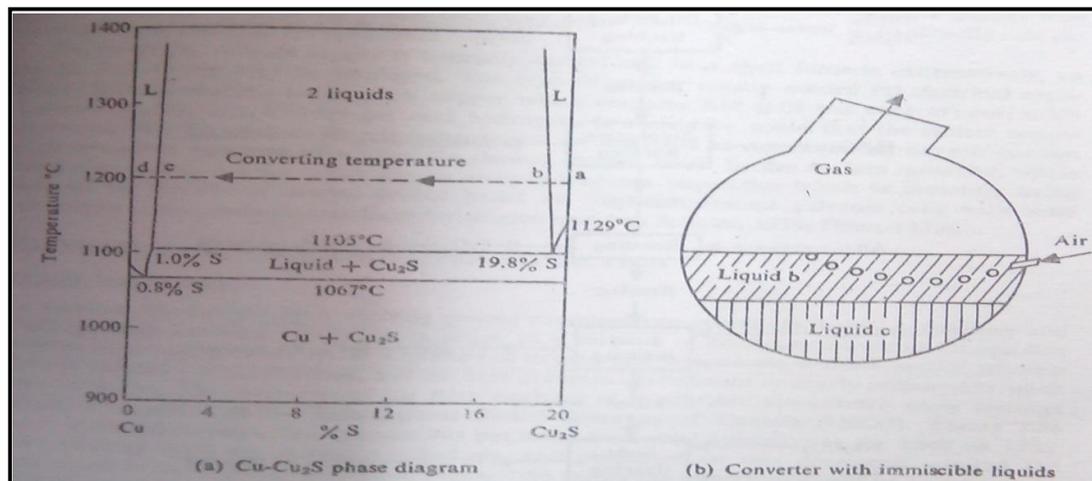
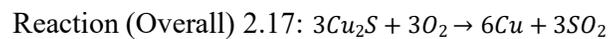
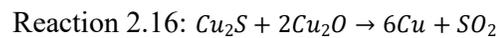
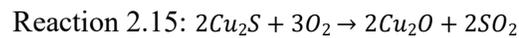


Fig. 2.5: Operating Principles of Side-blown Converter

When white metal (Cu_2S) is oxidized the S content in the bath decrease by about 19.5% and hence, the bath split into two layers as

- Top sulphide layer of less dense.
- The bottom Cu layer of denser contains about 1.2% S.

As oxidation continues the sulphide layer (liquid b) decrease and Cu layer (liquid c) increase. The relative volume of the two phases is determined by the Lever rule.

The blister Cu contains 0.02-0.05% S along with 0.2-0.5% dissolved O_2 .

Refining:

The purpose of refining to get Cu extraction is twofold as

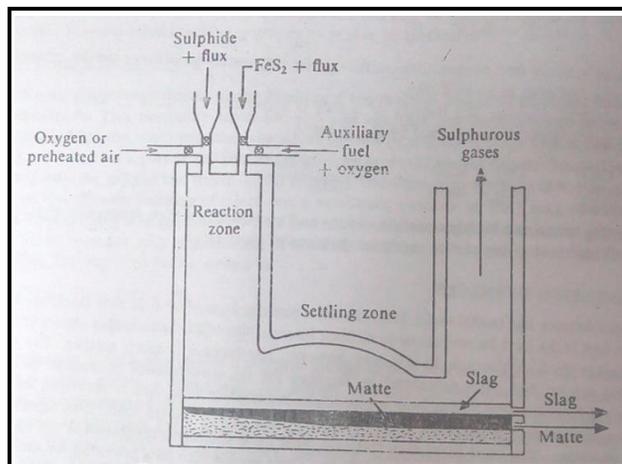
- First, to obtain metal in pure form.
- Second, to recover precious metals containing in blister Cu produced.

(i) *Fire Refining*: The purpose is to remove S from liquid blister Cu as SO_2 by oxidation with air and eliminate O_2 by introducing hydrocarbons. In this case, the refining is done in the reverberatory furnace of 400 tons of capacity containing blister Cu get oxidized to recover Cu removing the impurities such as S, Fe, Se, and Zn by converting its corresponding oxides and then skimmed off. But some Cu is also in the form of oxides.

To prevent Cu loss poling with green branches is used to reduce Cu_2O using hydrocarbon or some other reducing gases. In this case, the purity of Cu was obtained at 99.97%. Fire refining is also done in a rotary type of refining furnace, where blister Cu is directly treated by blowing air.

(ii) *Electrolytic Refining*: The purpose is to further refine the fire-refined Cu by electrolysis. The electrolysis is done in an electrolytic refining tank made of concrete or wood of 3-5 m deep and utilizes minimum space with maximum cathode and anode area. The electrolyte is CuSO_4 , H_2SO_4 , some glue, and alcohol at temperatures $50\text{-}60^\circ\text{C}$.

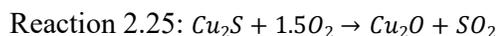
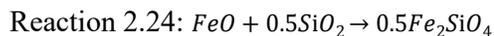
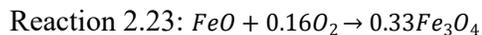
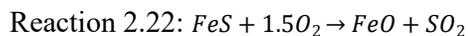
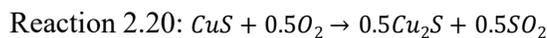
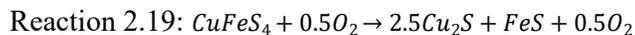
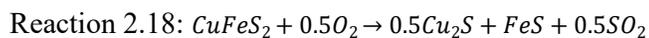
Cu transferred from crude anode to pure cathode. Impurities in blister Cu such as Fe, Co, Ni, Se, Te, etc. go into the solution and precious metals get collected below the anode known as anode slime.

(ii) Newer Process:**Flash Smelting:****Fig. 2.6: Flash Smelting**

Process: Enriched preheated air or pure O_2 is used to increase the combustion rate and autogenous smelting. The gases come out rich in SO_2 due to the high combustion rate and are used for H_2SO_4 manufacturing.

The process is autogenous and provides exothermic heat. Air is used as an oxidant to preheat. The composition of concentrate used in flash smelting has Chalcopyrite ($CuFeS_2$) 66%, Pyrite (FeS_2) 24%, and gangue (SiO_2) 10%. Whereas the Cu matte contains 70% Cu, 8% Fe, and 22% S, slag contains Fe 40% at a furnace temperature of $1300^{\circ}C$.

Main reactions of flash smelting of Cu concentrate are mentioned below



Continuous Smelting:

It encompasses smelting and converting in a single vessel i.e. Cu concentrate charged at one end and Cu metal withdrawn continuously at the other end.

Mainly three are three main processes, given below

(a) WORCRA. (b) Noranda. (c) Mitsubishi.

(a) WORCRA Process:

This name is divided as the first 3 alphabets stand for the developers and the last 3 alphabets for the place.

Features:

- Counter current movement of gas and concentrate. So, continuous production of blister Cu.
- Directly blister Cu i.e. metal instead of matte form.
- Combine smelting and converting.
- The heat required for the reaction is directly obtained as the reaction is exothermic.
- Counter movement causes continuous production of H_2SO_4 due to continuous extraction of gas.
- Cu% continuously obtains from slag using cleaning operation.

Process:

The process combines 3 different operations in a single furnace as

- Continuously smelting
- Continuously converting
- Continuously slag cleaning by conditioning and settling

Efficiency:

It increases using counter-current movement to increase the reaction surface area in the smelting and converting zone. Hence, effective removal of impurity occurs i.e. mainly Fe due to counter movement of slag and matte. As a result, Cu gets reverted to matte and obtained.

Advantage:

- Continuous process
- Capital cost low
- Concentrate passes large surface area. Hence accelerate the reaction.

Disadvantage:

- Not durable
- Operating cost is high

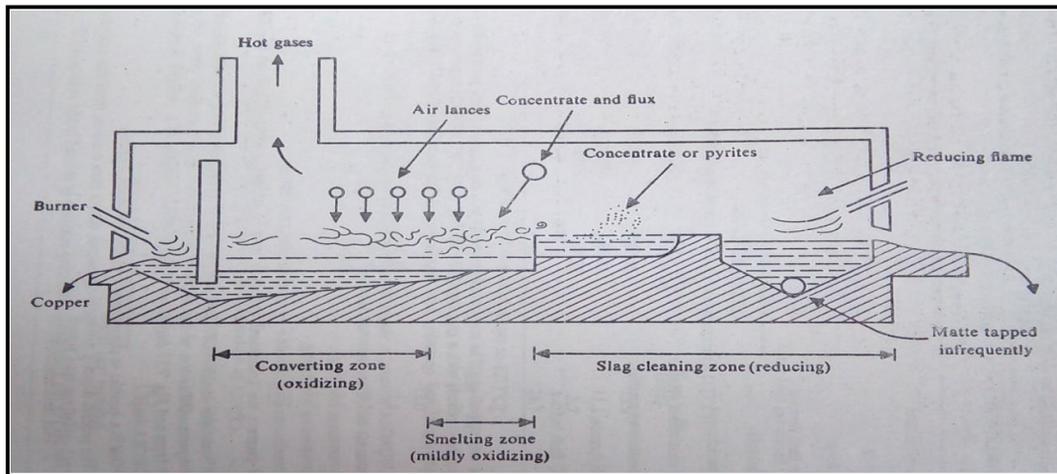


Fig. 2.7: Vertical Sectional Diagram of Straight form of WORCRA reactor

(b) Noranda Process:

Principle: In this process, high-grade Cu matte directly forms from the sulphide using air blown through the tuyeres to oxidize. The Cu or matte collected at the tap hole where the slag collected at the other tap hole at the slag end.

In this process, the slag contains high Cu compared to the WORCRA process. Mainly 3 layers present in the product as

- Cu – First layer.
- Matte – Second layer.
- Slag – Third layer.

Condition:

- If the given air is more than the stoichiometric amount of air required for oxidation, then the matte level decreases, and the Cu level increases.
- If insufficient air is required for the stoichiometric amount, then unoxidized iron and S tends to combine with Cu to form matte. So matte level increases and the Cu level decreases.
- If the air supply is equal to the stoichiometric amount, then both matte and Cu levels get increases.

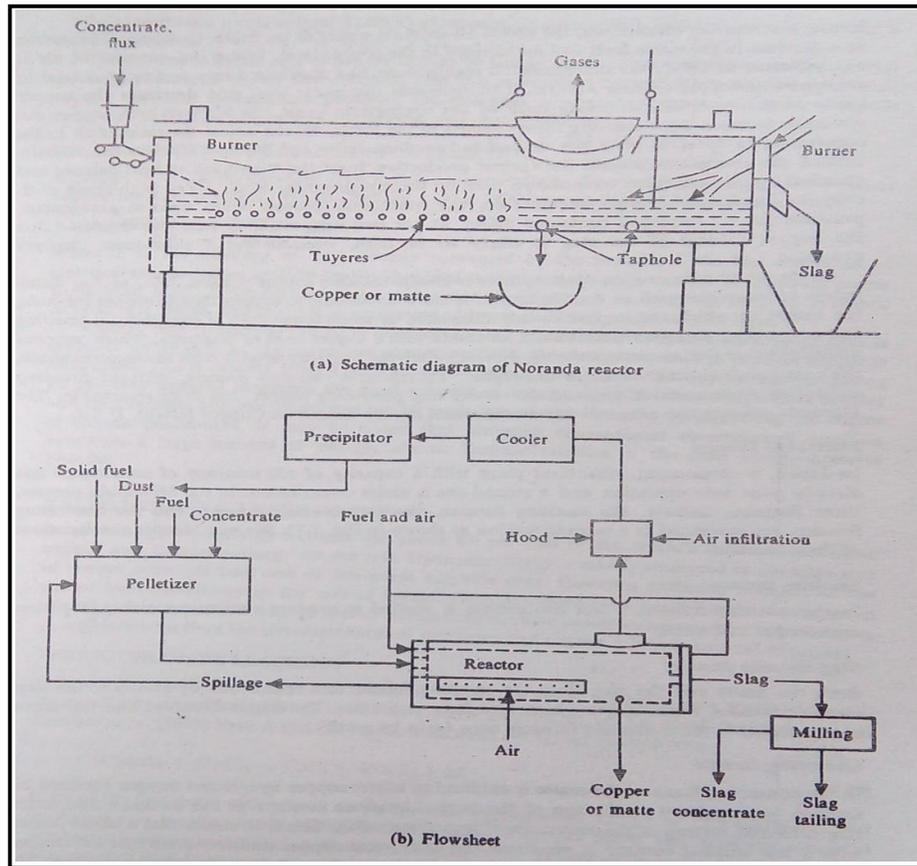


Fig. 2.8: Schematic diagram of Noranda Process

(c) Mitsubishi Process:

Principle: There are three furnaces such as smelting, slag cleaning, and converting furnaces are connected in a cascade manner. The product of one furnace goes to the next furnace for the next operation using gravity force.

Process:

First, in a smelting furnace (wet concentrate + flux + air + O_2) is smelted to produce a matte of 60-65% Cu, and the rest is slag.

Secondly, both matte and slag go into a slag cleaning furnace where slag gets discarded and matte goes to the next furnace operation.

Thirdly, in the converting furnace matte oxidized to blister Cu by blow of O_2 enriched air and limestone added as slag. So, slag is discarded as lime ferrite. Blister Cu produced of low % S and hence, obstruct the transfer of Cu to lime ferrite slag.

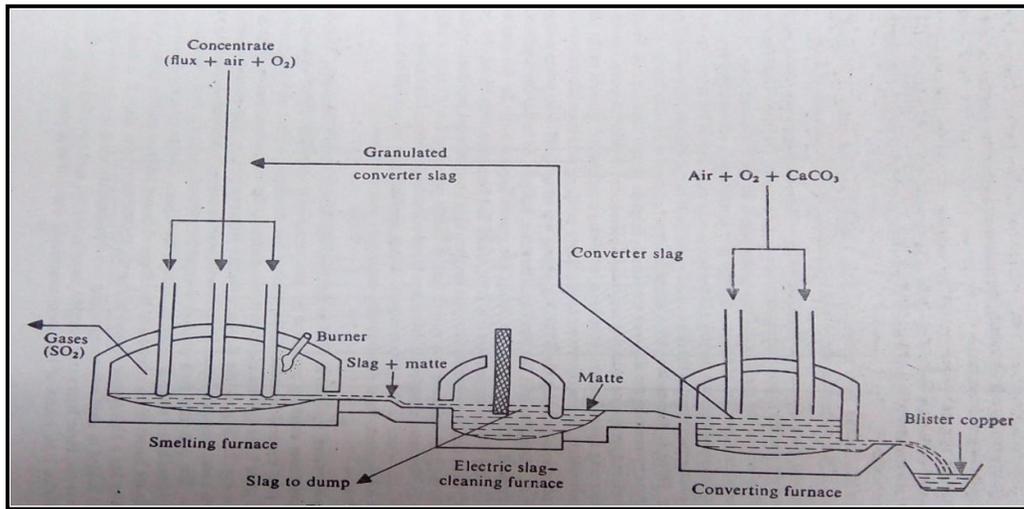


Fig. 2.9: Mitsubishi Continuous Smelting Process

2.1.4 Area of Extraction:

- HCL (Hindustan Copper Ltd.)
- Khetri mines
- Kholihana mines

2.1.5 Uses:

- Conducting element
- Alloying addition

Question 2.1: What do you mean by Zone refining?

Answer:

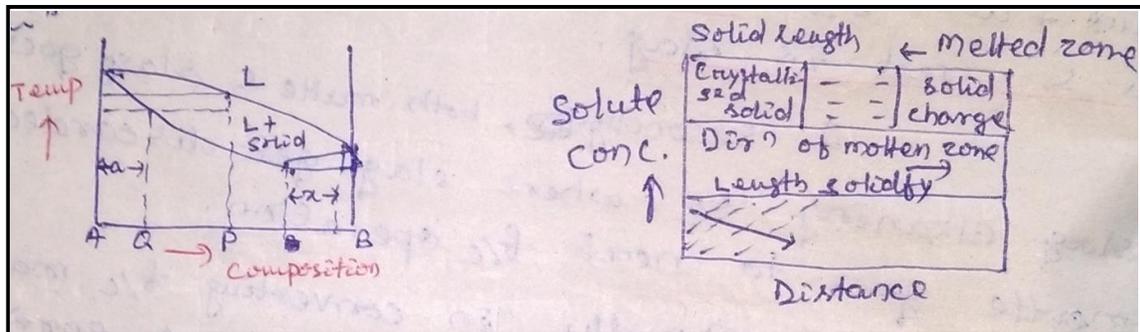


Fig. 2.10: Zone Refining

This refining technique is applied only for ultra-pure material i.e. low concentration of impurity. If 'A' is solvent and 'B' is solute then addition of A with B caused increase in melting point of B as due to bond formation as A-A, A-B, B-B.

When A at the composition of 'P', it gets melted and when it is at 'Q' gets solidified, whereas the same composition for B at equilibrium condition is less than A.

The free energy difference is

$$\Delta G_B(s) - \Delta G_B(l) = RT \ln \left(\frac{a}{b} \right) = RT \ln K_0$$

[Where $K_0 = a/b =$ segregation coefficient

$a =$ concentration of impurity in liquid

$b =$ concentration of the liquid]

It mainly applicable to the semiconductor industry during coal refining of Ge and Si

2.2 Extraction of Zn by Pyrometallurgical Process:

2.2.1 Details about Zn:

Atomic Number: 30

Atomic Weight: 65.38 amu

Density: 7.13 gm/cm³

Melting Point: 419.5⁰ C

Boiling Point: 910⁰ C

2.2.2 Common Minerals of Zn:

Zinc Oxides		Zinc Sulphides	
Name	Chemical Formula	Name	Chemical Formula
Zincite	ZnO	Sphalerite	ZnS
Franklinite	[ZnO(Fe, Mn) ₂ O ₃]		
Willemite	Zn ₂ SiO ₄		
Smith Sonite	ZnCO ₃		

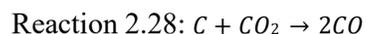
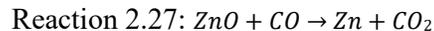
2.2.3 Extraction of Zn from Sulphide Ores:

Mainly five important processes are there in the extraction of Zn. Given as follows

1. Horizontal Retort
2. Vertical Retort
3. Electrothermic
4. Electrolytic
5. Imperial Smelting

The first 3 processes are the retort process. Horizontal retort made a major contribution to Zn production. The electrolytic process gives a high degree of purity at about 99.95%. In this case, Si rectifiers are incorporated in electrolytic cells to reduce power consumption and higher current density achieve.

Recently imperial smelting process (ISP) was used, in which ZnO gets reduces by the use of molten lead to give Zn vapour which on condensation gives Zn.



Flow sheet Explanation

The technique of smelting and converting is not applicable for Zn extraction because ZnS does not melt even at 1500⁰ C. Zn% 55 in concentrate treated under roasting, then it goes for sintering where roasted oxide ground and agglomerates then go under C reduction to give Zn in vapor form. For vitalization hearth roaster is used instead of another.

Initial stage roasting i.e. vitalization a hearth roaster employs. Then remaining PbS removed during sintering rather than high temperature. To prevent distillation of Pb present in Zn concentrate to obtain high-grade Zn with low Pb in retort distillation.

In most cases, a fluid bed roaster is allotted instead of a hearth roaster because of the following advantages

- Autogenous
- Good control over high throughput of Zn calcine
- Generate SO_2 suitable for acid production

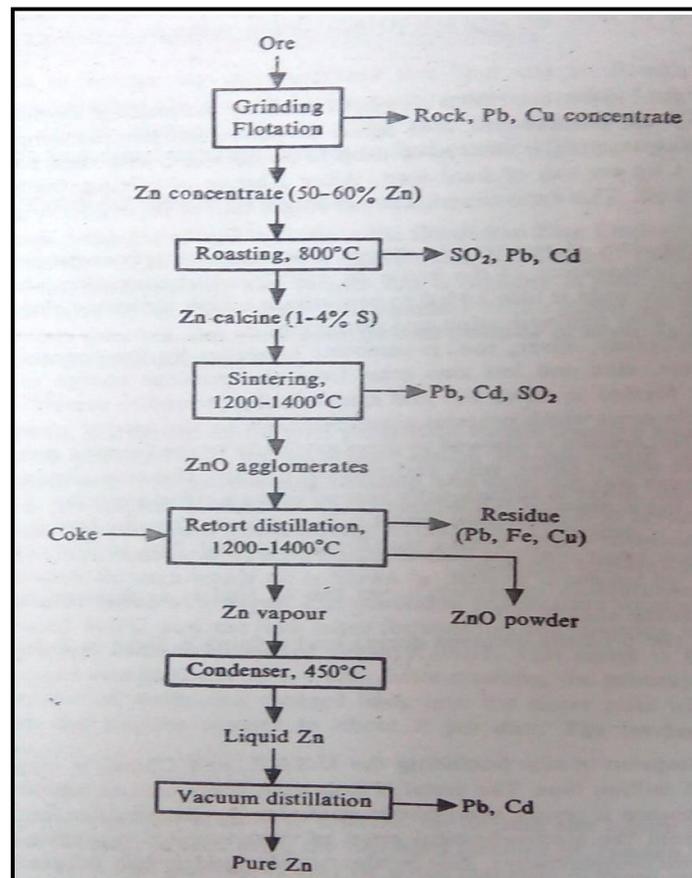


Fig. 2.11: Extraction of Zinc by Reduction of ZnO by Carbon

1. Horizontal Retort Reduction:

It is a 2m long, 0.25 m dia., 5-6 cm thick gas-fired retort furnace made of clay used instead of metal retort because most metals give alloy when heated with Zn. Here, a first loose mixture of ZnO presence of coke breeze charged at 1400°C to volatile Zn which on condensation gives Zn.

2. Vertical Retort Reduction:

It is 10 m in height, 0.7 m dia., and 30 cm thick of capacity 7-10 tons Zn per day made of SiC because it has a high conductivity of approximately 5 times compared to horizontal retort.

3. Electrothermic Reduction:

It occurs in a vertical retort 15 m high, internal dia. of 24 m about 100 tons Zn per day. Electrodes of graphite are introduced through SiC walls at two places near the bottom and at a zone of 9 m from the bottom. The gas liberated due to the reduction of charge contains 40-45% Zn, 45% CO, 5-8% H₂, and 5-10% N₂. Avoid the formation of blue oxide of Zinc (ZnO+Zn) quick condensation takes place. Residue contains 3-4% Zn. Zn recovery over 95%.

The Zn product obtained in foregoing retort is known as Spelter.

Three distinct layers as

- The bottom layer is molten lead and contains some Zn of 1.5%.
- The top layer is Zn contains some lead of 0.8%.
- Intermediate layer of Zn and Fe (15-20) % called hard metal.

Zn layer is further refined by fractional distillation because the wide difference in boiling point of Zn (907⁰ C), Cd (780⁰ C), and Pb (1620⁰ C) facilitates the separation of one metal from the other.

4. Electrolytic Reduction:

In this case, when Zn immersed in their salt solutions, rapidly establish a reversible potential. Since the activation overpotential is low, the displacement behavior exhibits a good correlation with the electrochemical series of standard potentials.

5. Imperial Smelting Process (ISP):

The process inside the furnace based on reaction i.e. reduction of ZnO by C in the imperial blast furnace gives rise to Zn in vapor form which gets condensed by using molten lead.

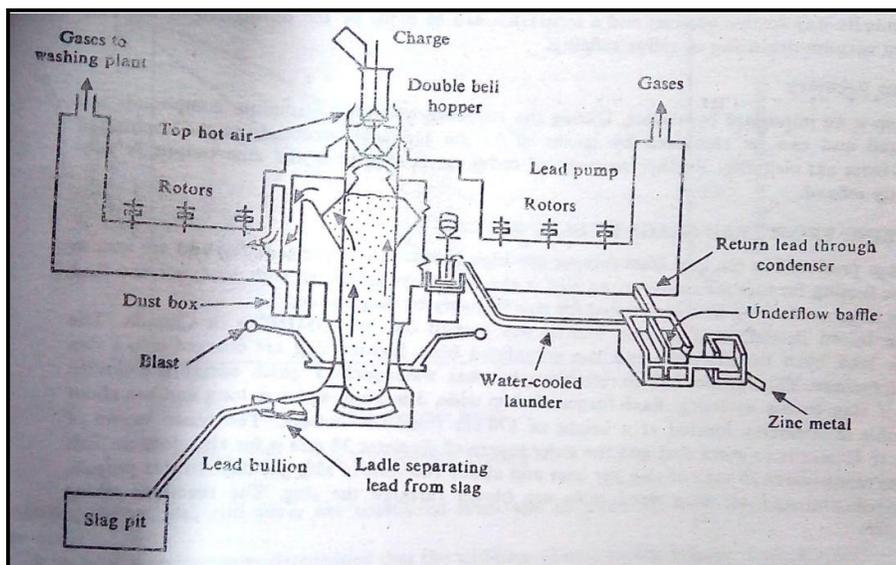


Fig. 2.12: Imperial Smelting Blast Furnace

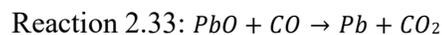
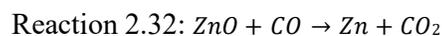
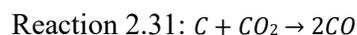
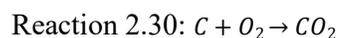
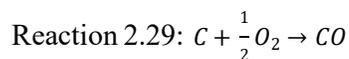
Principle: The process is carried out by using counter current principle, where the blast and preheated air given to the furnace through the tuyere and the preheated coke get charged from the top.

- ISP furnace is of square cross section consisting of water jacket brick-lined shaft.
- Feed is given from the top of the furnace at positive pressure where we introduced preheated coke and sinter through a double bell charging system.
- ISP consists of two condensers.

The function of Condenser: These are used mainly for two purposes as

- Cooling of furnace gases resulting due to condensation of Zn
- Dissolving condensed Zn in molten Pb

The smelting reactions that take place in ISP are



Carbon does not reduce ZnO until 1120°C , since at this temperature Zn gets vaporized. After reduction complete the molten slag and Pb are taken out from the furnace hearth at the melting point of the slag. Hence, Zn gets collected. So, the reaction (2.31) may backward as due to the difference.

Lead Bullion (Rich 70-80% Pb):

It is the mixture of Pb and precious metal, where preheated coke 800°C and sinter 21% Pb are charged from the top of ISP furnace. So, at the bottom collect precious metals such as Au, Cu, etc. and this liquid is known as Lead Bullion.

Advantages of ISP:

- Economical production of Zn.
- Efficiency overall high but Zn recovery is expensive.
- Entire amount of Au, Sb, etc. recover.
- Capacity large.
- Complete mixed charge of Zn and Pb simultaneously charged and recovered.
- Operational cost is low.
- No additional C is required.

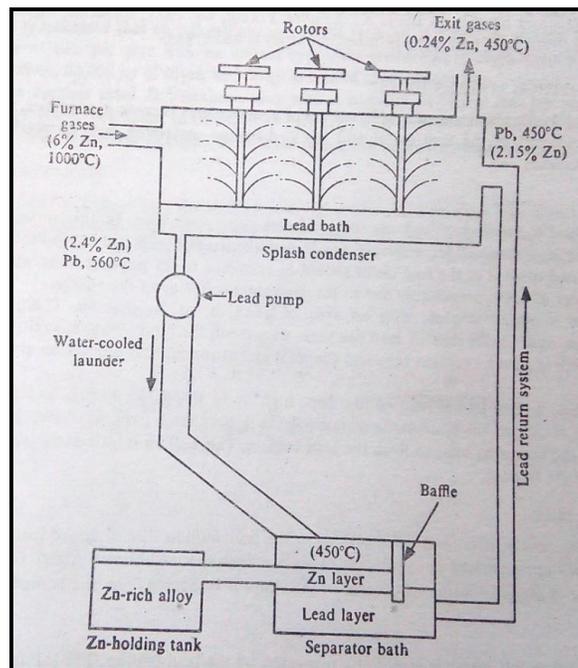


Fig. 2.13: Zinc Recovery System

Zn from Lead Slag by Slag Fuming Process:

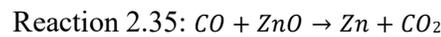
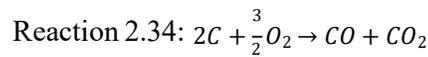
The process initiated by Consolidated Mining Smelting Corporation (COMINCO) of Trail, Canada.

Process:

The slag produced in lead baths or furnaces is 15-18% Zn. Molten slag from lead blast furnace together with cold slag from dump get transformed by ladle and charged to the furnace. Then the treatment growing of molten slag by treated with a mixture of pulverized fuel and air through the tuyere. Hence, Zn gets volatilized and slag gets solidified using a water jacket on all sides of the furnace except the roof as the roof is covered with fire bricks.

So, due to a mixture of pulverized fuel and air through tuyere, the reactions occur as

In both,



Above both,

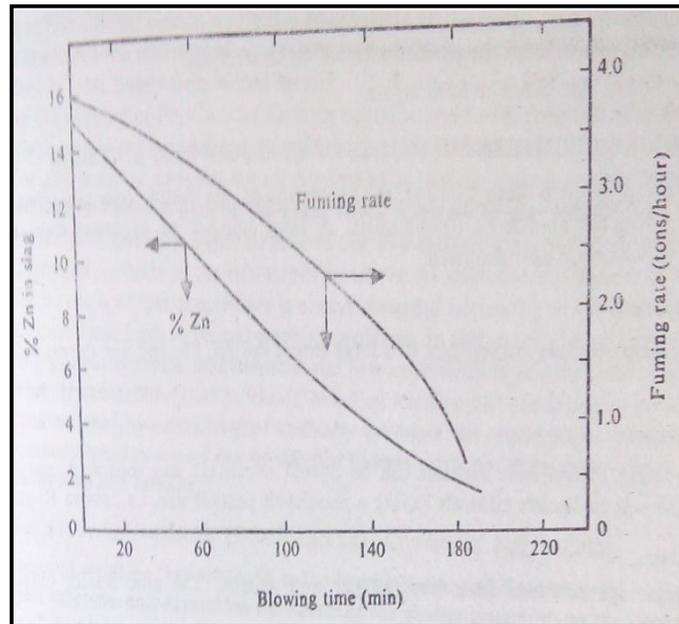
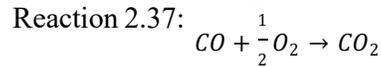
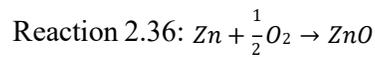


Fig. 2.14: Zinc Elimination during Slag Fuming

The excess heat gets collected by boiler located in gas collection system and the control of coal to air ratio determine the extent of heat released and maintained the bath temperature at about 1200⁰ C.

First, the elimination of Zn is slow but reaching 1000⁰ C get accelerate so Zn% 2.5%. Whereas Zn 1% in slag product contain 93-94% Zn. Fume leaves furnace at 1100⁰ C by combustion which effected by water cooling tower at 200⁰ C.

Then fume enters to big house where Zn collected and dispatched to Zn smelter on an electrolytic plant.

2.2.4 Uses or Applications:

- Zn extensively used as a protective coating for steel (Galvanization).
 - Restrict atmosphere corrosion by impervious basic ZnCO₃ layer.
 - Zn is more electropositive.
- Fabrication of Cu-Zn alloys i. e. brasses.
- Spraying – Zn used as for spraying in comparison to other metal on that metal which has low melting point.
- Rolled Zn: Usual method of cold working. Zn is rolled to sheet, plate, and strip. Where, sheet plates are rolled from 98.5% Zn.
- Pigment: Zn in the form of its oxides used in manufactured of paints.
- Alloys: Mainly Cu-Zn alloy produce of 30-37% Zn which is much less plastic when cold and worked about 500⁰ C which mainly used as die casting alloy.

2.2.5 Area of Extraction:

- Zawar Mines (Rajastan).
- HZL (Hindustan Zinc Ltd.).
- COMINCO-BINANI at Kerela.

2.3 Extraction of Pb by Pyrometallurgical Process:

2.3.1 Details of Pb:

Atomic Number: 82

Atomic Weight: 207.21 amu

Density: 11.34 gm/cm³

Melting Point: 327.40⁰ C

Boiling Point: 1737⁰ C

2.3.2 Common Minerals of Pb:

Pb Oxides		Pb Sulphides	
Name	Chemical Formula	Name	Chemical Formula
Cerrusite	PbCO ₃	Galena	PbS
Anglesite	PbSO ₄		

2.3.3 Extraction of Pb:

2.3.3.1 Production of Pb bullion:

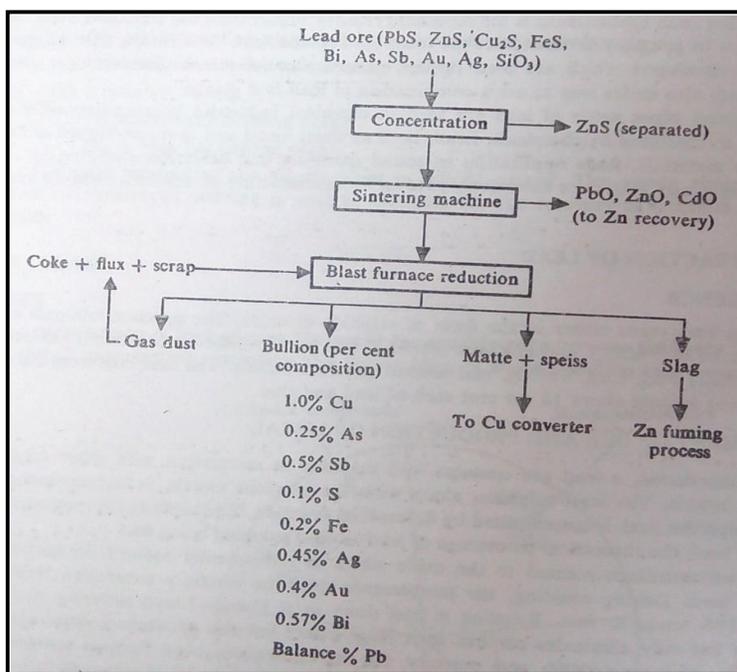


Fig. 2.15: Flow Sheet of Production of Lead Bullion

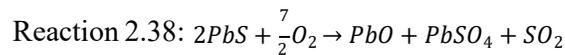
Complete Description of Flow Sheet (Extraction of Lead Bullion):

The concentrate is roasted to oxides which get reduced by C at 800⁰ C above to fuse PbS, which occurs in Dwight-Lloyd sintering machine in which not only S eliminate but also produces sinter with liquidized properties. In this S% reduce from 16-18% to 1-2%.

Now, up draught is preferred compared to down draught because of

- Higher capacity
- Elimination of precipitate wind box Pb
- Production of high-grade SO₂ gas

In, up draught air is admitted from the perforated bottom gate. Heat liberated due to roasting of Galena ore out of temperature below fusion temperature.



Hence, free Pb gravitated to the wall of furnace, and was trapped out at a working temperature of 1000⁰ C. Thick crust was allowed to accumulate slowly down even stopped reaction by the action of the air blast.

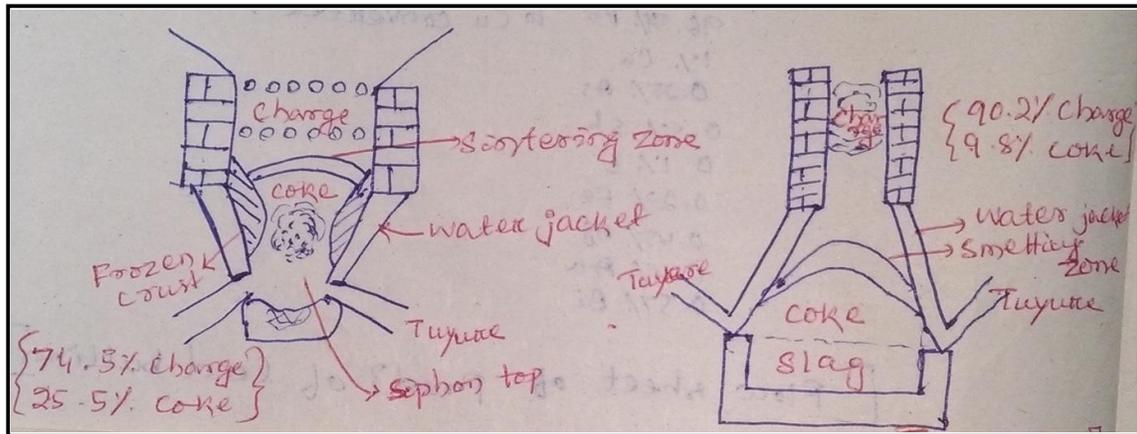


Fig. 2.16: Molten Blast Furnace

Older Blast Furnace Reduction: Furnace decreases in area from throat downward a minimum being attended end at tuyere of area 4-5 m². Blast pressure increases resulting in the vitalization of lead. As increase volume of blast lead vitalization decreases.

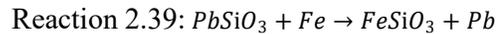
Siphon top: A device connects molten Pb in the crucible with a bowl shape which keeps the level of Pb in the crucible constant by withdrawing from the top.

Bosh: Purpose is to retard the descent of charge in front of tuyere as material melts away.

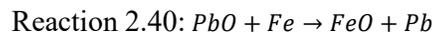
Newer Blast Furnace Reduction: Furnace area increases from the throat to downward at the end of tuyere. In this case following advantages occur

- Gases more evenly through charge
- Decreasing their velocity
- Decrease the amount of dust
- Here scrap is used for the reduction of unreduced PbO

PbO also helps in reducing PbS left unroasted in the sinter machine and hence, combines with Silica (SiO_2) in charge to form $2\text{FeO} \cdot \text{SiO}_2$. Hence, lower the melting point of slag and increase the fluidity of metal layer.



Finally helps to reduce the Pb as



The disadvantage of the Newer Process:

- A thick mushy layer forms in between matte and slag which interferes with theseparation
- A wide variation in temperature from 200°C at the top to 1200°C at tuyere

In blast furnace, reduction produces 4 distinct layers based on their specific gravity as

- First layer – Slag Layer – Specific gravity 3.6
- Second layer – Matte Layer – Specific gravity 5.2
- Third layer – Speiss Layer – Specific gravity 6
- Forth layer – Lead Bullion/Base Bullion

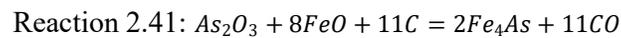
Question 2.2: What is Speiss?

Answer: When Unvolatized As reacts with Fe and C to form Speiss

OR

A molten phase consists primarily of Fe, as which is commonly encountered in lead smelting operations.

Mainly Speiss closely relate in the presence of As and Fe in the blast furnace.



Speiss have high content of Cu which get recovered by treated it in converter to get Cu in blister form. Finally, the Pb bullion or base bullion gets from the blast furnace which on refining produces pure lead.

2.3.3.2 Refining of Pb Bullion to extract Pb:

Liquation: Refining technique for removing impurities from metal based on the factor such as the difference in melting point of alloy components. So as a number of layers formed on the basis of their difference in density.

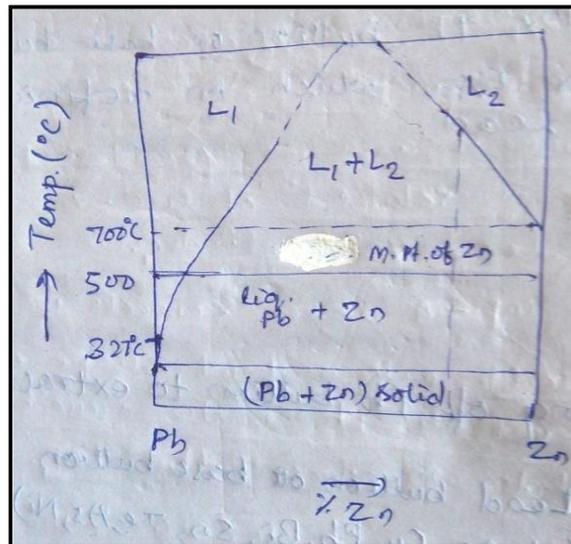


Fig. 2.17: Lead Zinc Phase Diagram

The crude Zn containing 5% Pb was refined in a reverberatory furnace by slowly cooling of molten matte.

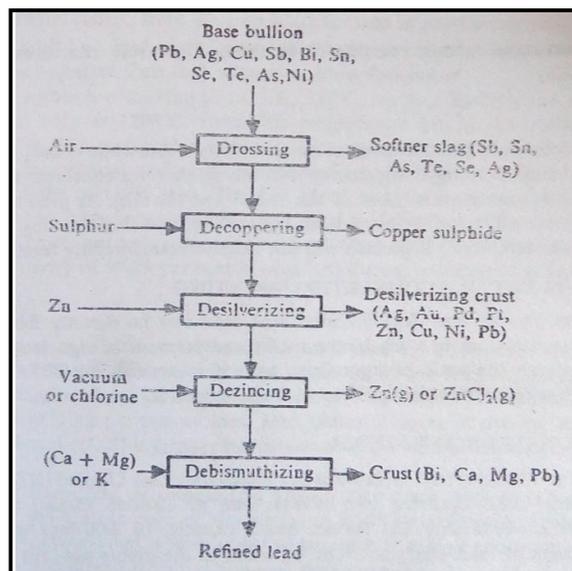


Fig. 2.18: Refining of Lead Bullion

Drossing: It is the combined process of liquation and oxidation in reverberatory furnace at around temperature 350°C at which many metals oxidized and skimmed off subsequently increase to $(500-550)^{\circ}\text{C}$. At that temperature, Cu get eliminated using S in the form of Cu_2S and at last softener slag (Sb, Sn, As, Te, Se) is removed.

Question 2.3: What do you mean by softening and why it is required?

Answer: Drossed bullion containing 1-1.5% of impurities as Sn, As and Sb treated in a softening furnace in oxidized condition. It required for refining of soft malleable Pb and hence, increases corrosion resistance. As the presence of impurities Sn, As, Sb causes hardened of alloy. So softening is required.

Desilverising of Pb (PARKE'S Process):

The process of desilverising of Pb by Zn addition followed by cooling for Ag recovery is known as desilverising of Pb or PARKE's process.

In this process, Zn is added to molten Pb because of the high affinity of Zn for Ag and Au than Pb. Then two liquid layers such as

- Liquid Zn at the upper layer having Zn% of 0.6
- Liquid Pb at the bottom layer having Pb% of 1.5

As Zn layer (first layer) has a higher melting point. So, solidify first.

Question 2.4: What is Cupellation?

Answer: Melting Ag containing Pb in a reverberatory furnace by exposing it to a blast in a layer. So Pb or other metals get oxidized and slag off except Ag as it has no attraction towards O_2 . Hence Ag remains behind i. e. get separated in the hearth.

Dezincing:

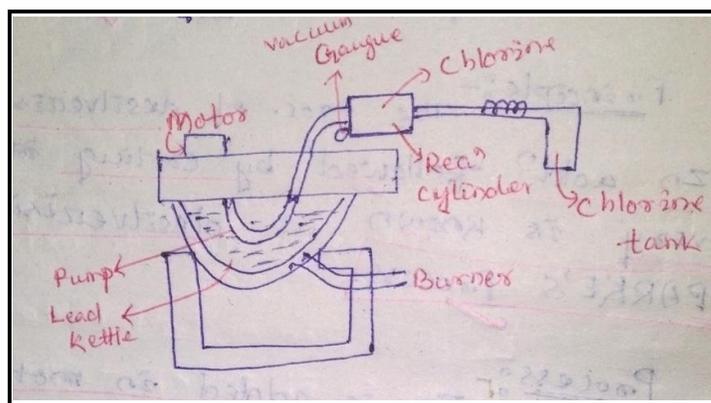


Fig. 2.19: Dezincing Process

As Cl_2 reacts with Zn to give $ZnCl_2$ in an enclosed chamber cause the left of Pb and falls on the bath to look grayish scum surface (layer of bubble formation) which gets skimmed off.

At the endpoint, all Zn chlorinated and white crystals of $PbCl_2$ appeared on the surface. Vacuum dezincing desilverised lead pump to kettle i. e. bath enclosed by steel ball shaped with a stirrer and vacuum pump attached. As the temperature rose to $540^{\circ}C$ vacuum distillation was carried out in reduction of Zn content from 0.6 to 0.002% in lead.

Debismuthizing:

The base bullion consists of high Bi% which separated by addition of Ca and Mg to forms an insoluble intermetallic compound of Bi.

2.3.3.3 Refining of Lead bullion to get Pb using electrolytic reefing:

In the electrolytic cell, the pure Pb is obtained by taking base or Pb bullion at the anode where the electrolyte consists of a solution of $PbSiF_6$ (Lead Fluoro Silicate) in H_2SiF_6 (Hydrogen Silicic acid). In this case, H_2SiF_6 gets acted on white Pb. So, granulated Pb was obtained to give $PbSiF_6$ in the next step.

2.3.4 Use of Pb:

- Manufacture of battery, cable, pigment, flexible sheet, and pipe. Basic Pb carbonate such as $2\text{PbCO}_3 \cdot \text{Pb}(\text{OH})_2$ is used in Pb pigment form based on grade point. Also, litharge (PbO) is used in reverberatory furnaces for oxidation for pigmented varnish and glass production.
- Red Lead (Pb_3O_4): Higher oxide of Pb made by heating Litharge under control conditions at about 450°C and same use as Litharge.
- Lead Chromate: Normal yellow chromate made by the precipitate from the solution of Lead acetate and nitride to which Sodium Bicarbonate will be added.

2.3.5 Area of Extraction:

- HZL – Major production of Pb in Bihar (Tundoo)
- Visakhapatnam (Andhra Pradesh) – 15,000 tons/year

Question 2.5: Discuss the important mechanical properties of Lead (Pb).

Answer: Important mechanical properties of Lead (Pb) are

- Heaviest metal
- Most malleable, easy to shape and working
- Low melting point

2.4 Extraction of Ni by Pyrometallurgical Process:**2.4.1 Details about Ni:**

Atomic Number: 28

Atomic Weight: 58.69 amu

Density: 8.98 gm/cm^3

Melting Point: 1452°C

Boiling Point: 2900°C

2.4.2 Common Minerals of Ni:

Pentalandite – $[(NiFe)_9S_8]$

Violarate – $[Ni_2FeS_4]$

2.4.3 Extraction of Ni from Sulphide Ore:

2.4.3.1 Explanation of Flow Sheet:

Initial Treatment: The ore is a mixed Cu-Ni ore with a nearly equal amount of Cu and Ni. The ore undergoes into grinding and froth floatation to produce a bulk concentrate which sent to copper cliff mill for separation of Cu concentrate, Ni concentrate and Pyrrhotite concentrate with iron sulphide with about 0.8% Ni.

Then the Cu concentrate was subjected to O_2 flash smelting for Cu extraction. From Pyrrhotite after roasting iron oxide form and Ni separated by leaching. The Ni concentrate with about 10% Ni, 2% Cu, 40% Fe, and 30% S goes for extraction of Ni in the next stage.

Roasting: In the conventional process, the concentrate is partially roasted to oxidize the iron sulfide either multiple hearth roaster or fluidized bed roaster. But a fluidized bed roaster is preferable because of

- High output and rich SO_2 gas stream generate
- Process is autogenous
- Temperature range $(550-600)^{\circ}C$ about 40% S oxidized

Smelting: The roasted calcine contains the desired amount of siliceous flux is smelted in a reverberatory furnace to produce a matte containing Cu, Ni as 20%, 7% respectively where the slag discarded contains gangue and oxidized iron. There is also converter slag of both Ni and Cu converters are returned to the reverberatory furnace.

Converting: Furnace matte is converted to Ni-enriched matte with 50% Ni, 25% Cu, 0.7% Fe, and 21.5% S at $1150^{\circ}C$ in Pierce-Smith converters. The slag discarded contains 2% Ni, 1.5% Cu, 40% Fe, and 25% SiO_2 and is returned to the reverberatory furnace for recovery of Ni and Cu.

Slow Cooling: Converter matte subjected to slow cooling process from melting point to $400^{\circ}C$ for 3 days to form three layers precipitate out as

- First layer, Cu_2S precipitate and grows
- Second layer, metallic Cu-Ni alloy at $700^{\circ}C$
- Third layer, solid Ni_3S_2 phase precipitate at $575^{\circ}C$

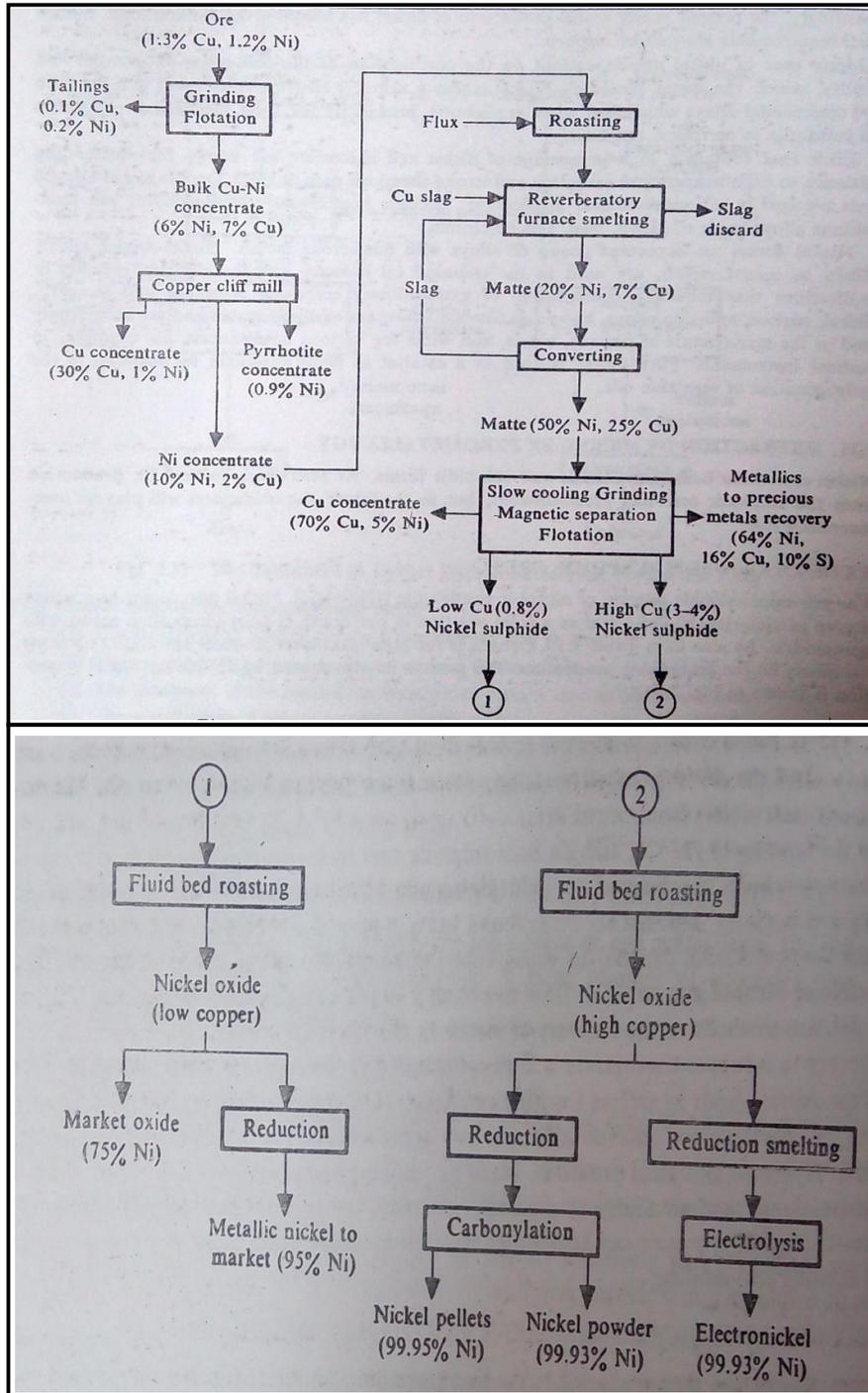


Fig. 2.20: Nickel Extraction (INCO Process)

Question 2.5: Why slow cooling required after converting process?

Answer: Slow cooling required obtaining necessary grain growth, which then go for subsequent processing of sulphides to recover it in the froth floatation.

Magnetic Separation and Floating: Diphenyl guanidine used as collector as well as frother rather than Xanthate. In floatation, Ni-Cu alloy contains 95% precious metals is undergo for magnetic separation for recovery of it. Cu_2S produced by floatation contains 70% Cu, 5% Ni, 20% S. Ni sulphide recovered as a low Cu-Ni sulphide with 74% Ni, 0.8% Fe, 0.8% Cu, and 22% S. High Cu-Ni sulphide with 72% Ni, (3-4) % Cu, 0.8% Fe, and 21% S at a temperature (1100-1250)⁰ C roasting produce granular nickel oxide.

Final Treatment:

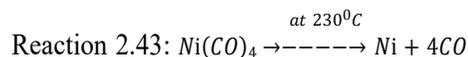
- Low Cu-Ni oxide is marketed directly as Nickel oxide or reduced to metal.
- High Cu-Ni oxide sends for refining by carbonyl process and other half by electrolytic refining.

2.4.3.2 Refining Process:

1. Carbonyl Process for Refining Ni:

(i) Mond's Process:

In 1889, this refining process of Ni recovered by Carl Langer and Ludwig Mond. In this process, at temperature (40-90)⁰ C metallic Ni combine with CO to give gaseous nickel carbonyl $[\text{Ni}(\text{CO})_4]$. At higher temperature (150-300)⁰ C $\text{Ni}(\text{CO})_4$ decomposes to give Ni and CO gas. The reactions are



Other forms of Carbonyls are volatile carbonyl $[\text{Fe}(\text{CO})_5]$, Co carbonyl in tetracarbonyl $[\text{Co}_2(\text{CO})_8]$ tricarbonyl $[\text{Co}_4(\text{CO})_{12}]$ form. Cu and other major elements are not form carbonyls.

(ii) INCO Process:*INCO Atmospheric Carbonylation Process:*

The oxide first reduces to active Ni in the presence of H_2 at about $400^{\circ}C$. Then active Ni undergoes for carbonylation at $50^{\circ}C$ to form $Ni(CO)_4$ then at $230^{\circ}C$ goes for decomposed to Ni either in pellets about 1 cm dia or powder form about $3.5 \mu m$ size.

INCO Pressure Carbonylation Process:

The carbonylation reaction has 4 to 1 volume change permits at about $180^{\circ}C$ and 70 atm pressure carbonyls of Ni, Fe, and Co formed. From which $Ni(CO)_4$ recovered by fractiona distillation and converted to metallic Ni in pellet decomposer or a powder decomposer.

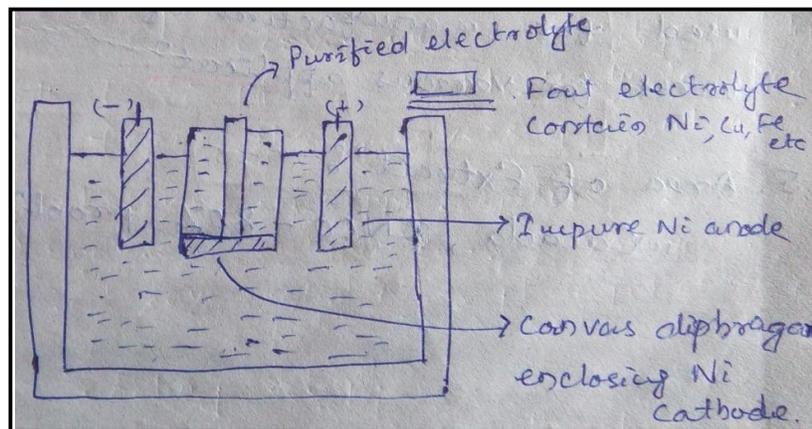
2. Electrolytic Refining of Ni:

Fig. 2.21: Electrolytic Refining of Ni

The Ni oxide reduce by coke in fuel fired furnace or electric furnace, and then cast into Ni metal anode. These anodes are electrolytically refined in a bath contains 60 gm/lit Ni^{+2} , 95 gm/lit SO_4^{2-} , 35 gm/lit Na^+ , 55 gm/lit Cl^- , 16 gm/lit H_3BO_3 . This electrolysis carried out at $60^{\circ}C$. Cu remove by cementation with active Ni powder, Fe and other impurities remove by aeration of electrolyte, Co remove by Cobaltic Hydroxide for further Chlorine oxidation. Electrolyzed Ni analyzes about 99.93% Ni.

But in INCO process, electro refining Ni sulphide to metallic Ni to produce anode contains 76% Ni, 0.5% Co, 2.6% Cu, and 20% S. The process is same as the above one, only difference is that the anode enclosed in a bag to collect anode slime contains 95% S and is processed to recover elemental Sulphur and precious metals. The electrolyte contains 99.95% Ni.

2.4.4 Uses of Ni:

- Important alloying elements used in chemical processing, space research and nuclear reactor.
- Ni-Cu alloys known as Monel metals are used in chemical oil industry and electric industry.
- Ni, Cu, and Zn alloys are known as German silver. It is used for manufacture of ribbons, bands, and wires for various applications.

2.4.5 Area of Extraction:

Canada, INCO-80% production

2.5 Extraction of Magnesium by Pyrometallurgical Process:

2.5.1 Details about of Mg:

Atomic Number: 12

Atomic Weight: 24 amu

Density: 1.738 gm/cm³

Melting Point: 650⁰ C

Boiling Point: 1107⁰ C

2.5.2 Common Minerals of Mg:

Dolomite – (MgCO₃.CaCO₃)

Magnesite – MgCO₃

MgCl₂ and MgSO₄ present in sea water.

Brucite – Mg (OH)₂

Carnallite – MgCl₂.KCl.6H₂O

Olivine – Mg₂SiO₄

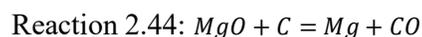
Serpentine – Mg₃Si₂O₇

Asbestos – $\text{CaSiO}_3 \cdot 2\text{MgSiO}_3$

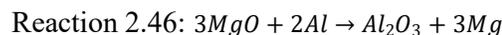
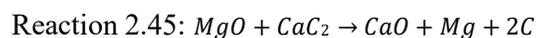
Kainite – $\text{MgSO}_4 \cdot \text{KCl} \cdot 3\text{H}_2\text{O}$

2.5.3 Extraction of Mg:

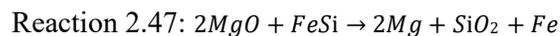
The extraction of Mg occurs by the reaction of MgO with C but not accepted commercially because it is difficult to prevent the backward reaction of Mg vapour with CO.



Again, MgO reacts with CaC_2 and Al to give Mg as



Again, in Pidgeon process MgO reacts with Ferro-Si to give Mg vapour in vacuum as

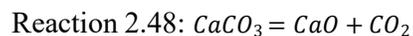


Dow Process:

It is used for production of Mg from sea water containing MgCl_2 . Sea water used because of

- Economically Mg extract.
- Uniformity of Mg content.

The sea water contains $\text{MgCl}_2 \cdot 1.25 \text{H}_2\text{O}$ as feed material. In this process, the following reactions occur as given below



The first step involves collection of Mg as insoluble hydroxide by precipitation with lime according to the above reaction. The suspension is distributed to thickeners, in which hydroxide settles, clear overflow pumped back to sea. The milk of Magnesia underflow is picked up by slurry pumps and delivered to filter tanks. Subsequently; $Mg(OH)_2$ is agitated with 10% HCl. The dilute solution is concentrated and filtered to recover crystallized solids. The filtrate is subjected to evaporation to recover the remaining chloride. During evaporation because hydrous chloride is to lose HCl and result final product is contaminated with oxide. The reaction is

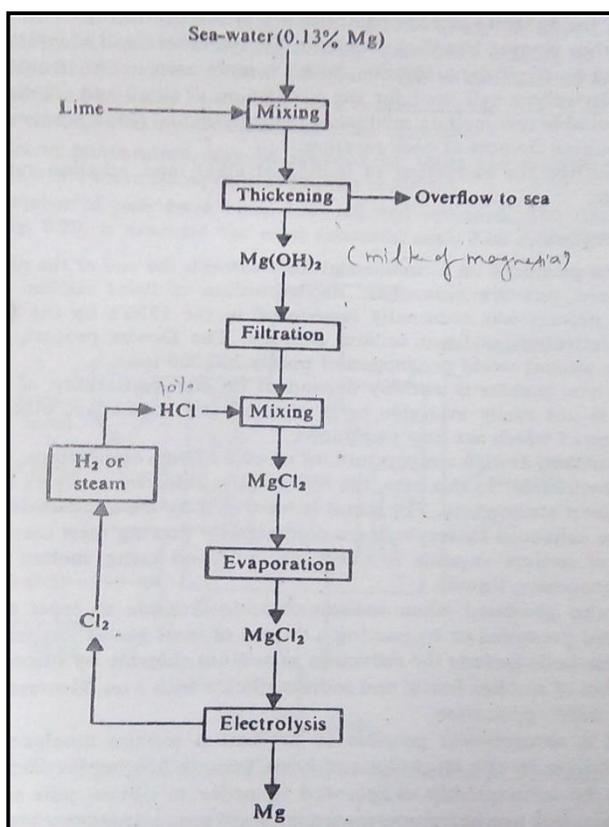
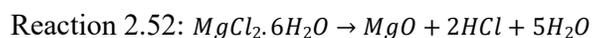


Fig. 2.22: Production of Mg from Sea Water

Electrolysis of MgCl₂:

Mg is reactive metal has high decomposition potential. Therefore, aqueous electrolysis H₂ evolution occurs before decomposition of MgCl₂. In electrolysis, MgCl₂ get fluidized by use of flux and to increase the density of bath so that metal liberated at cathode floats on the surface of electrolyte and not sink.

The electrolytic cells are two types lined and unlined. First one provides separate cathodes made of steel where the later one is steel wall of itself used as cathode. In both cases, graphite used as anode which is inert i. e. not consumed to the metal divert away from anode zone and Chlorine gas collect in a hood and metal to a pool. Chlorine gas from the anode used to dry incoming MgCl₂ and chlorinated Mg (OH)₂.

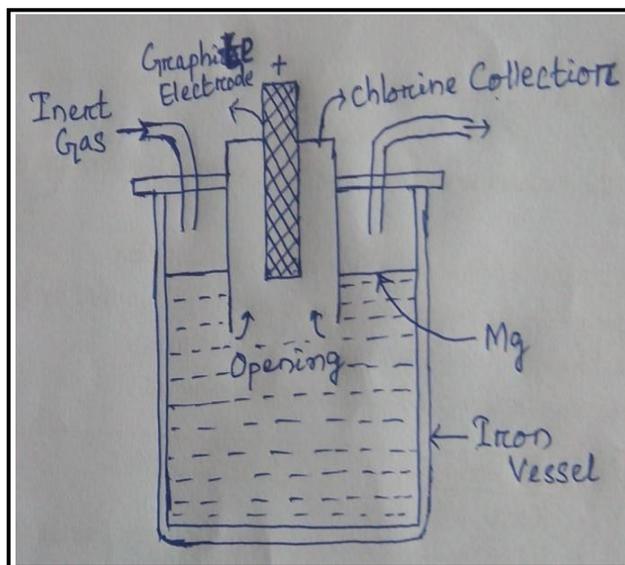
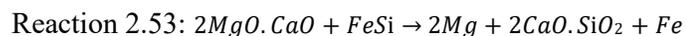


Fig. 2.23: Electrolytic Cell for Mg Production

Pidgeon Process:

Pure Si reduces MgO when both reactants are in standard state. To overcome these difficulties there we will be taken Ferro-Si powder to make in briquette form (75% Si) with calcine dolomite to give Mg in vapour form at (1100-1200)⁰ C under vacuum of 0.1 mm Hg in external of retort as



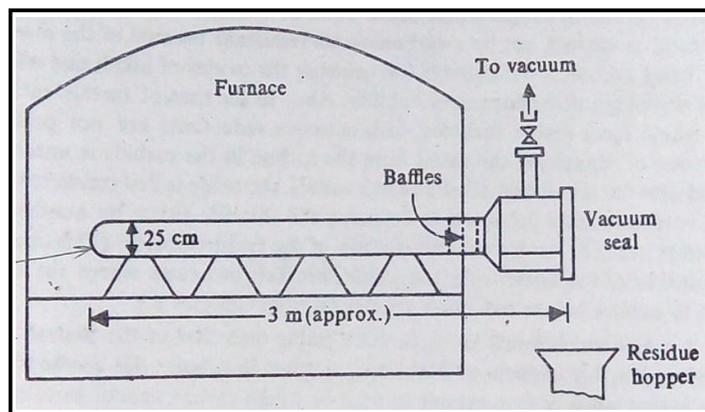
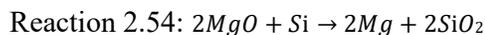


Fig. 2.24: Retort for Pidgeon Process

The product Mg is gaseous form removed from the retort to avoid possibility of back reaction during cooling and condensation.

The reaction occurs in two ways at 1200⁰ C

- Absence of lime



- Presence of lime



The process is highly endothermic reaction, so large heat input required. There is small addition of CaF₂ or MgF₂ added to speed of the evolution of Mg vapour. Since, vapour pressure of Mg is not altered the effect of fluoride addition is considered catalytic.

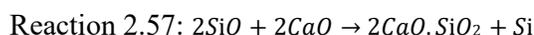
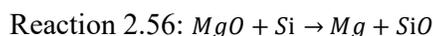
Equipment and Operation:

The retort is of 25 cm dia. and 3m length inserted horizontally in the furnace made of Nichrome steel with 15% Ni, 28% Cr, 1.25% Mn, 1.5% Si, 0.3% C and Fe balanced in air cooled continuously batch process. The furnace heated either by gas or by electricity.

The charge is of 1:6 ratio of Fe-Si to calcine dolomite of 100 kg weight. Heating carried out to vaporize structurally combined water. The temperature raised about 1150⁰ C and in vacuum. At this stage, Mg vaporizes and condensed by baffles. Application of vacuum stopped and seal broken condensed Mg removed out. Approximately, 120-ton dolomite with 12 tons Ferro silicon gives rise to 10 tons of Mg.

Reaction Mechanism: The reaction occurs in following stages

- Reaction between CaO and Fe-Si produce liquid Ca-Si-Fe. The reaction occur at 1000⁰ C and mildly exothermic. The ternary alloy Ca-Si-Fe serves as a reducing agent.
- Mg vapors produce by reduction of MgO by Ca-Si-Fe alloy. Here, pressure increases and slow down the rate of reaction. Kinetics of the reaction control by the presence of SiO vapour in Pidgeon process as



Magnotherm Process:

This process is same as the of Pidgeon process. But the only difference is temperature maintained about 1500⁰ C and the bath maintained in the molten state by addition of alumina to form molten slag.

2.5.4 Uses of Mg:

Used of Mg fall under two main heads as

- Non – Structural: Use in alloying, deoxidation and desulphurization of molten metal.
- Structural: Use in automotive industry.

2.5.5 Area of Extraction:

- National Metallurgical Lab. (NML), Jamshedpur
- Central Electrochemical Research Institute (ECRI), Karoikudi, Tamilnadu.

2.6 Extraction of Aluminium by Pyrometallurgical Process:

2.6.1 Details about of Al:

Atomic Number: 13

Atomic Weight: 27 amu

Density: 2.702 gm/cm³

Melting Point: 660.37⁰ C

Boiling Point: 2467⁰ C

2.6.2 Common Minerals of Al:

Diaspore – $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$ (Al % - 65.4)

Gibbsite - $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ (Al % - 65.4)

Bauxite – Mixture of Diaspore and Gibbsite

2.6.3 Extraction of Al:

Extraction of Al by carbothermic reduction of Al_2O_3 in commercially not adopted because of

- Required high temperature
- Undesirable Al carbide form
- Refractories required for this process are expensive and also, not readily available

Extraction of Al salt in electrolysis not adopted because of decomposition of aqueous solution results evolution of H_2 gas at lower voltage. So, as Al^{+3} not occur. The H_2 over voltage insufficient to overcome 1.67 volt that the difference in potential between Al deposition and H_2 evolution.

Extraction of Al in Hall – Heroult process: In this process, dissolved Alumina (Al_2O_3) in cryolite ($3\text{NaF} \cdot \text{AlF}_3$) is electrolyzed, and Al_2O_3 obtain in pure form.

Extraction of Alumina in Bayer's process: In this process, AlCl_3 electrolyzed in a fused salt medium contains NaCl and LiCl to obtain alumina (Al_2O_3).

2.6.3.1 Bayer's Process for Alumina Production:

Bauxite initially leached by NaOH under high pressure (25 atm), and temperature (220°C) to form soluble NaAlO_2 (Sodium Aluminate) from which $\text{Al}(\text{OH})_3$ is precipitate out. When $\text{Al}(\text{OH})_3$ calcined, Al_2O_3 or alumina obtained.

Bauxite obtains in mines get crushed and ground to very fine size in jaw crusher and hammer mills, result in high SiO_2 content in finer fractions. Then SiO_2 remove/render by screening in 100 mesh sizes, SiO_2 content is about 10% after comminution. Thus, leads to enrichment of Al_2O_3 in coarser fraction.

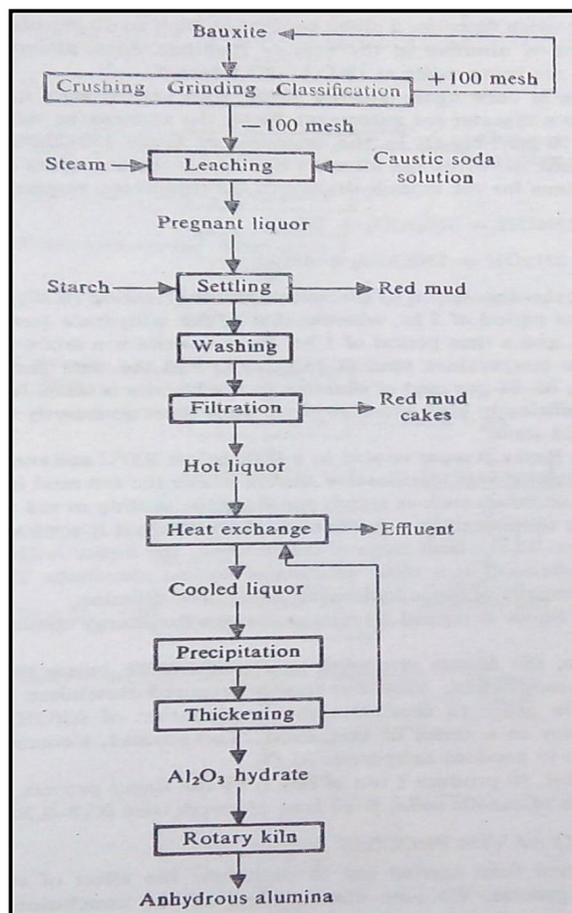
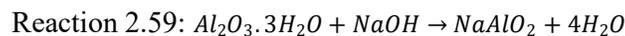
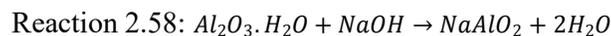


Fig. 2.25: Flow Sheet of Bayer's Process

Crushed bauxite once again ground with caustic soda (NaOH) in ball mill to yield slurry in which alumina dissolves in bauxite and dissolves in NaOH in a temperature (150-220)⁰ C and a pressure of (5-25) atm. As a result, impurities leave a suspended solid. The reactions for monohydrate and trihydrate are given below



Since bauxite is a mixture of mono and trihydrate contains (86-88) % alumina.

Then the liquor cooled little below 100⁰ C and completely pressurized is to settling and clarification section red mud get deposited in presence of starch which speed of the settling. Any remaining red mud gets separated by a series of washers by encountering flow of hot water. After washing the solution, the remaining residue consists of ferric hydroxide, silica and alumina.

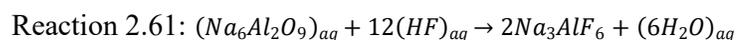
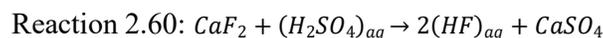
Hot liquor tapped by heat exchanges to obtain cool liquor. Then cool liquor under precipitation carried out a temperature below critical point. Then $\text{Al}(\text{OH})_3$ used for separation precipitate from liquor in a series of thickeners. The product, a coarse hydroxide is finally calcined in a rotary kiln to produce an anhydrous Al_2O_3 .

Factors affecting Bayer Process:

- Finer bauxite gives better digestion of alumina in liquid liquor.
- Wet grinding is more efficient than dry grinding.
- Digestion accelerates at high temperature above 100°C .
- At low temperature below a limit cause loss of alumina. So, maintain the temperature at about boiling point of water.
- Sensible heat of recovered from hot liquor by heat exchangers.
- In precipitation stage, entire precipitates of alumina not allow because it may cause precipitation of silica along with alumina.
- For efficient calcinations of alumina in rotary kiln at 1400°C to give anhydrous alumina.

2.6.3.2 Hall – Heroult Process:

This process based on electrolytic decomposition of alumina dissolved in a liquid bath of cryolite. This process requires high purity alumina, cryolite and ash less C electrode. Since alumina is not an ionic. So, it is capable to dissolve in cryolite up to 15% at 1000°C . The synthesise of cryolite occur as reaction of HF with sodium aluminate were, HF form due to the reaction of CaF_2 in H_2SO_4 as



Cryolite bath get enhanced conductivity by addition of CaF_2 and NaF . At 1000°C density of molten cryolite is 2.1 gm/cm^3 and Al_2O_3 is 3.96 gm/cm^3 . Al liberated heavier than bath and sinks at bottom. Higher the amount of Al_2O_3 in bath, the higher its density and consequently the sinking of Al is rendered more difficult.

Decomposition Potential of Al_2O_3 in dissolved in Cryolite:

- When anode is oxygen, decomposition potential of Al_2O_3 is about 2.1-2.15 v.
- Decomposition potential of graphite at anode is 1.12 v.
- Graphite aids the decomposition of Al_2O_3 because it reacts with O_2 to form CO and CO_2 .

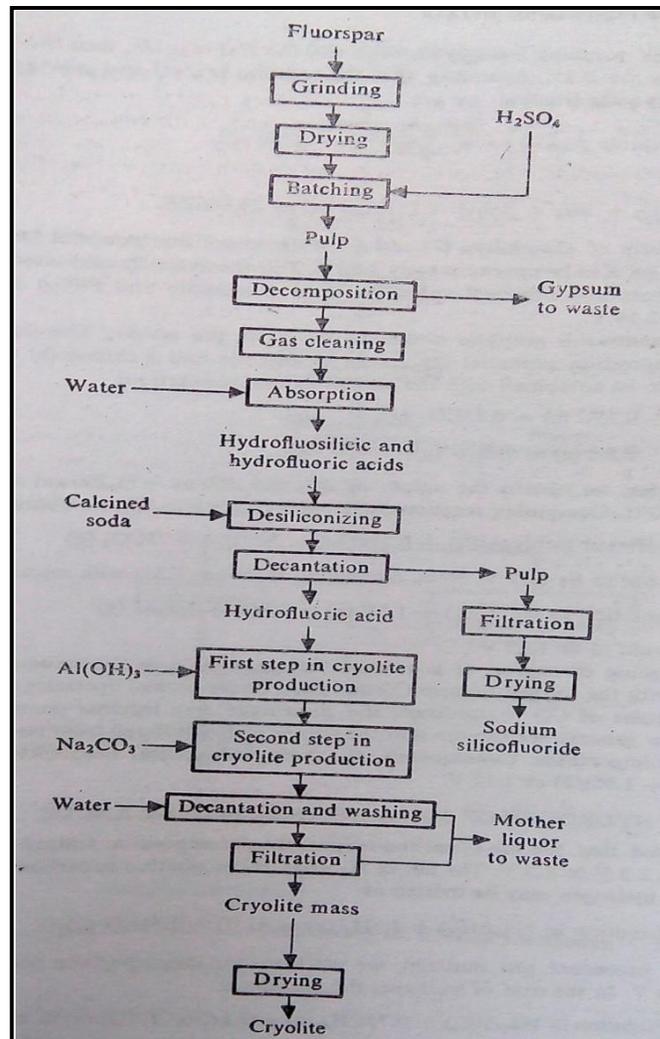
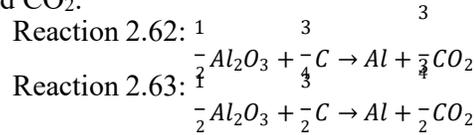
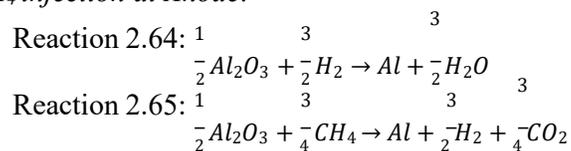


Fig. 2.26: Synthesis of Cryolite

Influence of H₂ or CH₄ injection at Anode:



Injection of H₂ or CH₄ would not be effective in appreciably reducing the decomposition potential of Al₂O₃. Injection may be effectively cut down graphite consumption and prove economical if H₂ or CH₄ available is cheaper. Actual decomposition potential is 5-7 volt, whereas

- Voltage needs for electrolytic reduction: 1.7 volt
- Voltage drops across C lining: 0.6 volt
- Voltage drops due to anode resistance: 0.5 volt
- Voltage drops due to resistance of electrolyte: 0.8 volt
- Voltage drops due to contact resistance: 0.3 volt

2.6.3.3 Electrolytic Reduction Cell:

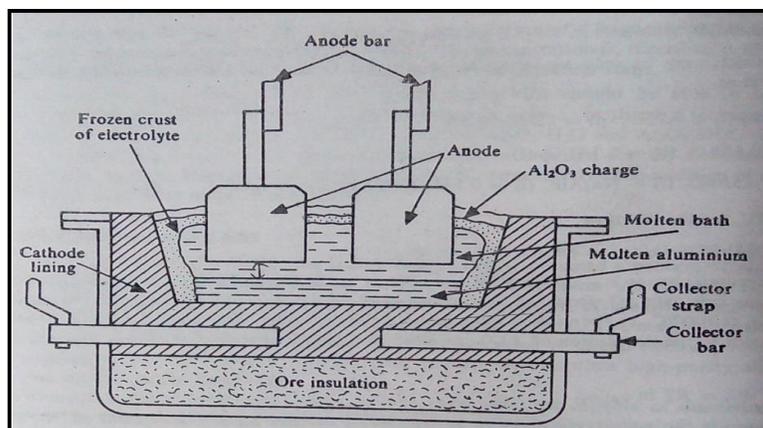


Fig. 2.27: Electrolytic Reduction Cell

The cell consists of rectangular refractory lining steel box with l=5 m, w=2 m, depth=1 m. Cathode lining consists of refractory bricks faced with C mixed tar binder. A consumable graphite electrode serves as anode. A Fe plate and Fe bar form the cathode are embedded in the C mixture. A tap hole is there, through which metal get collected at the bottom of the hole.

C gets backed in steel mould to produce anode. Fe pins are inserted by electric contact. The baked C electrodes and cathode lining must be sufficiently strong and dense.

Cell Operation:

During operation, the cell bath is filled with cryolite and anode lowered into it. A current is passed through it until cryolite melts (M. Pt. – 990⁰ C).

When the bath attains a molten state addition of alumina commences. The alumina decomposes to yield Al and O₂. The necessary heat is supplied by the resistance offered by both the electrodes and the electrolytes.

2.6.3.4 Anode Effect:

If the alumina content of the bath (normally 5-10%) falls below 2% then normal contact between the anode and bath is interrupted by the gas film in which abruptly increases positiveresistance. As a consequence, the normal operation ceases. This effect is called anode effect.

Under certain conditions, the electrolytes of fused salt are associated with the periodically occurring phenomena in which the terminal voltage increases to a high value and current density decreases to a lower value from their normal value. This effect is known as anode effect.

If power consumption, electrolyte, and anode consumption increase during the process. Then, the current density exceeds above critical current density.

Factors Affecting Anode Effect:

- Nature of electrolyte: The anode effect occurs most readily with fluoride and least with iodides, Bromides and Chlorides require large critical current density than iodide, fluoride both.
- Purity of electrolyte: Greater purity is lower in current density and vice versa.
- Type of electrolytic Anode: The cryolite-alumina melt anode effect at 4-5 amp/sq. cm. when C anode is used and 7-8 amp/sq. cm. for graphite electrodes.
- Temperature: An increase in temperature reduces the current density.

Cause of Anode Effect:

- Thermal Effect: Local overheating can lead formation of gaseous film which deposits an anode from the electrolyte.
- Electrostatic Causes: During electrolysis, gas bubbles form which is electrically charged. The charge may originate from the anions discharged at the boundary of newly formed gas bubbles at high current density. So, gas is partly ionized when gas bubbles acquire an opposite charge to that electrode i. e. negative charge there will be formation of adherent gaseous film which could separate the molten electrolyte from anode.

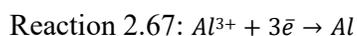
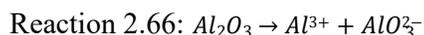
Metallic Clouds or Fogs:

In the electrolysis, fused salt electrolyte, fine globules at the cathode metal get appear in electrolyte is known as metallic clouds.

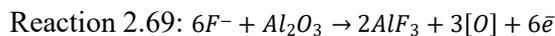
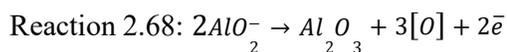
Role of Cryolites in Electrolysis:

Al_2O_3 dissolve in cryolite and form Al^{3+} , O^{2-} (in solute) and Na^+ , AlF_3^- , F^- (in solvent).

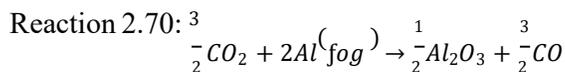
Cathodic Reaction:



Anodic Reaction:



Oxygen combines at carbon electrode to form CO_2 , according to the reaction as given below

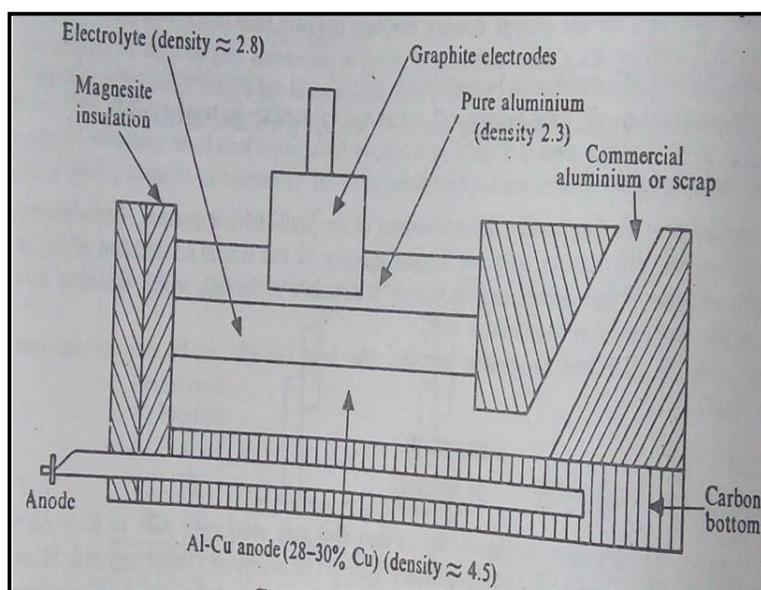


Simple molecular theories of electrolysis are based on the hypothetical dissociation reactions



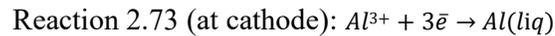
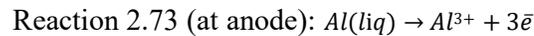
Factors Affecting Electrolysis:

- Temperature: Increase in temperature, current efficiency increase. For every 4⁰C rise of 1% current efficiency increases. More number of side reactions and hence dissociation of metal in the bath.
- Current Density: Current efficiency increases with an increase in current density.
- Inter polar Distance: As the distance increases along the bath from bottom to top, current efficiency also increases.
- Addition of Al₂O₃: At Al₂O₃% is 4%, there no effect on current efficiency. But, a decrease or increase in Al₂O₃ from 4% causes an increase or decrease in current efficiency.
- Bath Density: With the increase in bath density by AlF₃/NaF ratio. But, for large AlF₃% cause bath conductivity and fluidity decrease and volume increases
i.e. current efficiency decreases.

2.6.3.5 Electro refining of Al:**Hoopes Process:****Fig. 2.28: Three-Layer Process**

To achieve a high degree of purity three-layer processes are employed as

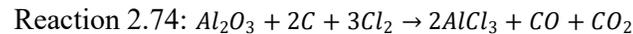
- Bottom or Anode Layer: The specific gravity of this layer is maximum and consists of impure Al alloyed with Cu. This layer serves as an anode.
- Middle or Electrolyte Layer: It consists of anhydrous fluorides and chlorides as AlF_3 36%, Barium Fluoride 18%, and CaF_2 16%.
- Top or Cathode Layer: Pure Al layer acts as a cathode.



The purity of metal is 99.99%.

2.6.3.6 Alternative Methods of Alumina and Al Production:

Alcoa Process:



Al_2O_3 from the Bayer process is chlorinated under reducing conditions in the presence of C at $(700-900)^\circ\text{C}$ to produce AlCl_3 , CO, and CO_2 . AlCl_3 vapor from the reaction is condensed at 70°C in a fluid bed containing AlCl_3 particles. The solid AlCl_3 particles feed into an electrolytic cell containing a fused chloride electrolyte ($\text{AlCl}_3 + \text{NaCl} + \text{LiCl}$) at 700°C . On electrolysis, Al (liquid) formed at the cathode, and gaseous Cl_2 liberated at the anode.

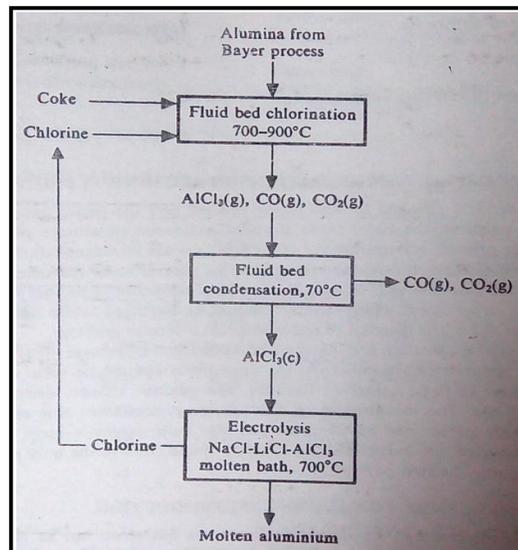


Fig. 2.29: Flow Sheet of ALCOA Process

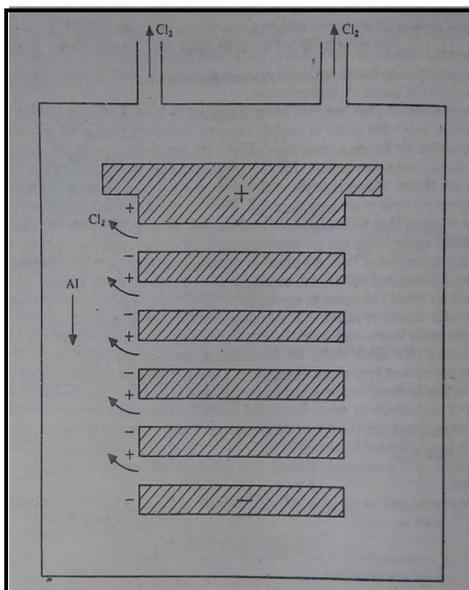


Fig. 2.30: ALCOA Bipolar Cell

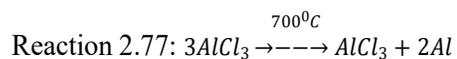
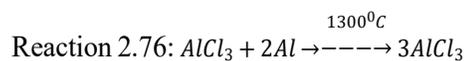
Toth Process:

A novel process has been proposed by Toth for the extraction of Aluminium. In this process, too, as in the ALCOA process, the alumina from the Bayer process is converted to $AlCl_3$ by chlorination in the presence of carbon. The $AlCl_3$ thus formed is reduced to Al by treating it with Mn at a temperature of 300^0C and a pressure of 15 atm. The reaction is



ALCAN Process:

- Bauxite reduced to Al alloy.
- Reduction carried out in EAF at 200^0C presence of coke.
- The alloy reacts with preheated $AlCl_3$ vapor in the reactor.



2.6.4 Uses of Al:

- Use as a good conductor
- Electrical application

2.6.5 Area of Extraction of Al:

- ALIND – Aluminium India
- HINDALCO – Hindustan Aluminium Corporation
- MALCO – Madras Aluminium Company

2.7 Extraction of Zirconium:

2.7.1 Details about of Zr:

Atomic Number: 40

Atomic Weight: 91.22 amu

Density: 6.45 gm/cm³

Melting Point: 1852⁰ C

Boiling Point: 3580⁰ C

2.7.2 Common Minerals of Zr:

- Zircon: ZrSiO₄
- Mineral associated with beach sands along with other minerals (Rutile, Ilmenite, and Monazite) and oxides of metals such as Fe and Mg.

2.7.3 Extraction of Zr:

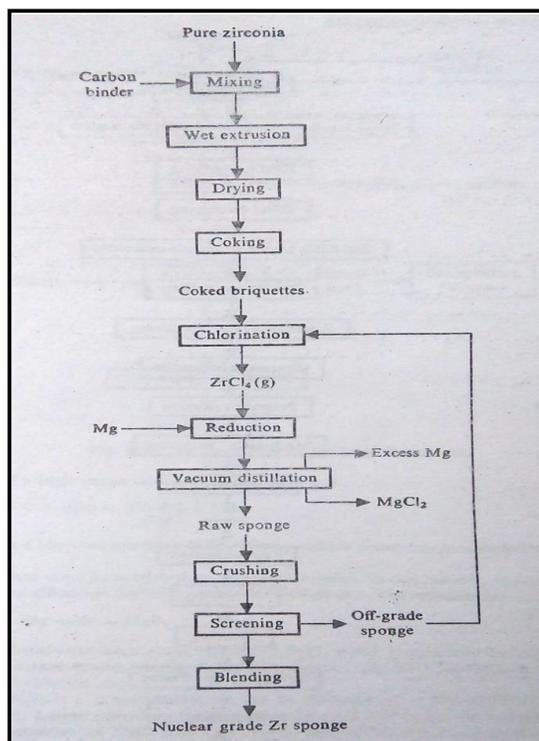


Fig. 2.31: Production of Zirconium Sponge

As mainly Zr constitute with HF. So our first aim is to separate HF from Zr.

2.7.3.1 Pyrometallurgical Methods:

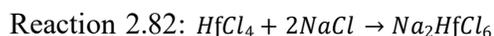
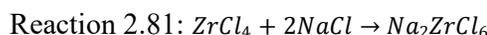
- a. Differential Oxidation of Tetrachloride reaction



- b. Formation of Lower Chloride of Zr reaction as



- c. Differential reaction of double chlorides with alkali chloride. The reactions are



2.7.3.2 Reduction of Zr Compound to Metal:

Metallic Zr is conventionally produced by metallothermic reduction of Zr tetrachloride. Kroll's process of Mg reduction is the most popular method. Recently Zr was successfully produced by the fused salt electrolytes of double salts such as Cs_2ZrCl_6 dissolved in alkali chlorides.

2.7.3.3 Production of Mill Products from Sponge Metal:

Conventional induction melting Zr alloys in graphite crucibles is unsuitable due to the prohibitive C pick-up. At present, consumable electrodes are melting with water-cooled Cu molds employed for tonnage melting of alloys. Procedure evolved for

- Compaction of sponge briquettes with alloying additions
- Fabrication of consumable electrode
- Controlled vacuum arc melting

For homogenizing the alloy composition and for removing the volatile impurities, double vacuum arc melting is used. The shaping of alloy and extrusion of tubes is achieved through a series of steps, each of which needs special precautions.

2.7.4 Uses of Zr:

- Nonnuclear applications: Flash bulbs in the form of thin foils and ceramics as an oxide.
- Nuclear reactor used as a cladding element.

2.7.5 Area of Extraction:

Tarapur (Maharashtra)

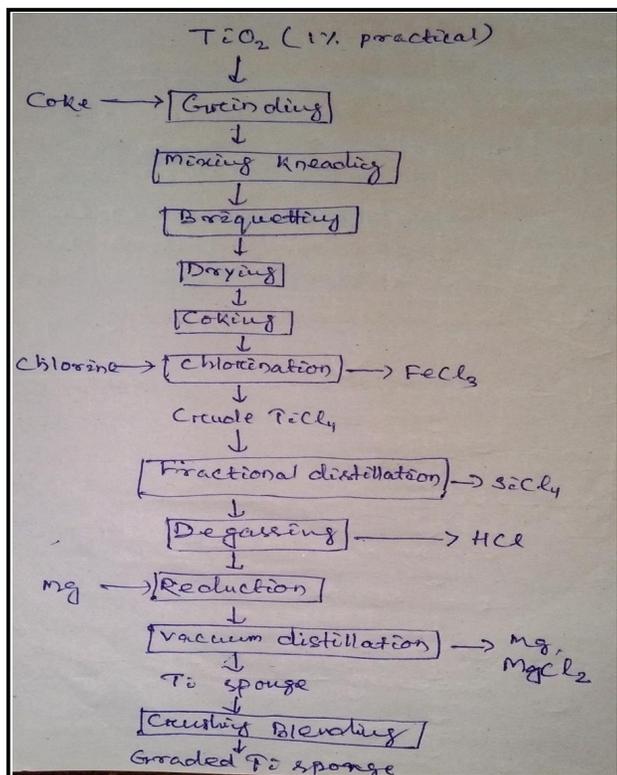
Kota (Rajasthan)

Kalpakkam (Tamil Nadu)

2.8 Extraction of Titanium:**2.8.1 Details about of Ti:**

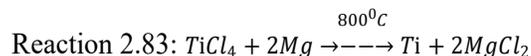
Atomic Number: 22

Atomic Weight: 47.90 amu

Density: 4.54 gm/cm³Melting Point: 1668⁰ CBoiling Point: 3260⁰ C**2.8.2 Common Minerals of Ti:**Rutile: TiO₂Ilmenite: FeO. TiO₂**2.8.3 Extraction of Ti:****Fig. 2.32: Production of Graded Titanium Sponge**

2.8.3.1 Kroll's Process:

The reduction reaction in Kroll's process can be written as

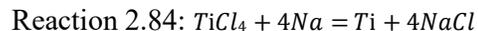


The reaction is carried out in a stainless-steel container for external heating by gas or electricity. An exothermic reaction occurs. After the reaction is over the Ti sponge is recovered either by dissolving MgCl₂ and excess Mg by leaching with cold dilute hydrochloric acid or by distilling off MgCl₂ and excess Mg by vacuum heating. The residue is pure Ti sponge. The maximum yield is 96% because of

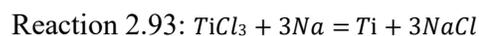
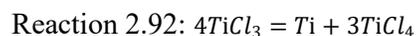
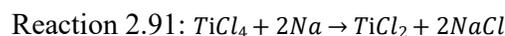
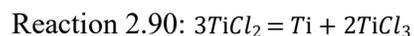
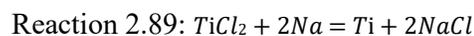
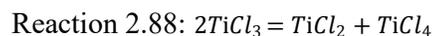
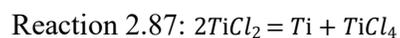
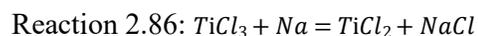
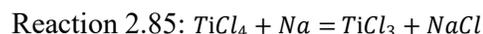
- Efficient Scavenger w.r.t impurity chlorides such as FeCl₃ and AlCl₃.
- Better conductivity of NaCl compared to MgCl₂ and hence, reaction is exothermic.
- NaCl-MgCl₂ leads to a better separation of metal from slag.

2.8.3.2 Hunter Process:

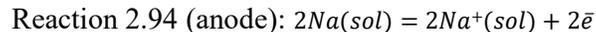
The reduction of TiCl₄ by Na is to give Ti known as the Hunter process and written as



The reaction is highly exothermic, there is a series of reactions occur as



The reaction of sodium with titanium subchlorides in fused sodium chloride is of an electrochemical nature and occurs the same as like galvanic corrosion of iron.



The anode reaction takes place at a metal surface where sodium metal, fused NaCl, and Cl^- ions are available for the salvation of Na^+ ions. The Cathodic reaction takes place at a metal site where soluble Ti in fused salt is available and Cl^- ions are released completely in the anodic reaction.

2.8.4 Uses of Ti:

The sponge Ti is produced in Kroll's process and ductile Ti is produced in Hunter's process.

The main uses of Ti overall are

- Jet engine components
- Airframes
- Missiles and spacecraft

2.8.5 Area of Extraction:

- Tuticorin (Tamil Nadu)
- Kerala

2.9 Extraction of Tin (Sn):

2.9.1 Details about Tin:

Atomic Number: 50

Atomic Weight: 118.71 amu

Density: 7.31 gm/cm³

Melting Point: 231.9⁰ C

Boiling Point: 2270⁰ C

2.9.2 Common Minerals of Tin:

Cassiterite: SnO₂

2.9.3 Extraction of Tin:

The concentration containing FeS is oxidized to oxide during smelting. The carbothermic reduction of SnO₂ is feasible at moderate temperatures. At higher temperatures (1200-1300)⁰ C FeO is more stable. The difference in stability between FeO and SnO₂ is used as the basis of selectivity of reducing SnO₂ to Sn. The difference in stability is further reduced because slag has a higher affinity for Tin oxide compared with iron oxide and the tendency to attain separation of Tin from Fe becomes difficult. So, the smelting of Tin is done in 3 ways.

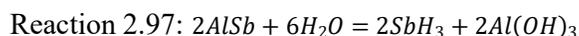
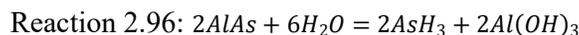
The first stage is a 99% partial reduction of the tin concentration of almost all Fe in slag. The slag was further reduced in 2 stages. The first stage produces tin containing 5% and in the second stage, tin contains 20% Fe. The third stage has very low % of tin. So, 2nd and 3rd stages are subjected to liquation to eliminate Fe.

When Sn-Fe alloys are heated to a temperature higher than the melting point of Sn. The metal known as first-run metal contains 99.9% Sn. Temperature progressively increases Fe content in Sn increase known as second-run metal. Where, for taking smelting in rotary instead of the reverberatory furnace of higher efficiency Sn recovery. First smelting slag is reduced and Sn is volatile either in the form of SnO or SnS. Clean slag discarded after reduction vapor pressure of SnS greater than SnO. So, SO₂ presence is beneficial.

Refining of Tin:

There is mainly two methods of refining of Sn as Pyrometallurgical refining (or fire refining), and electro refining.

Pyrometallurgical Refining: Ultimate recovery of Tin is more than 85%. The principle is to separate several common impurities from intermetallic compounds with each other or with Sn. Thus, if impure liquid tin is cooled to a temperature just above the melting point of tin (232⁰ C), these intermetallic compounds as Cu₂Sb, Cu₃Sn, Cu₃As, FeSb₂, FeSn₂, FeAl, etc. have higher melting points than tin. So, separate and get floats. S adds to remove Cu in CuS form. AlAs and AlSb by addition of Al, As, and Sb eliminated.



NH₄Cl is added with tin after Al treatment to remove surplus Al as



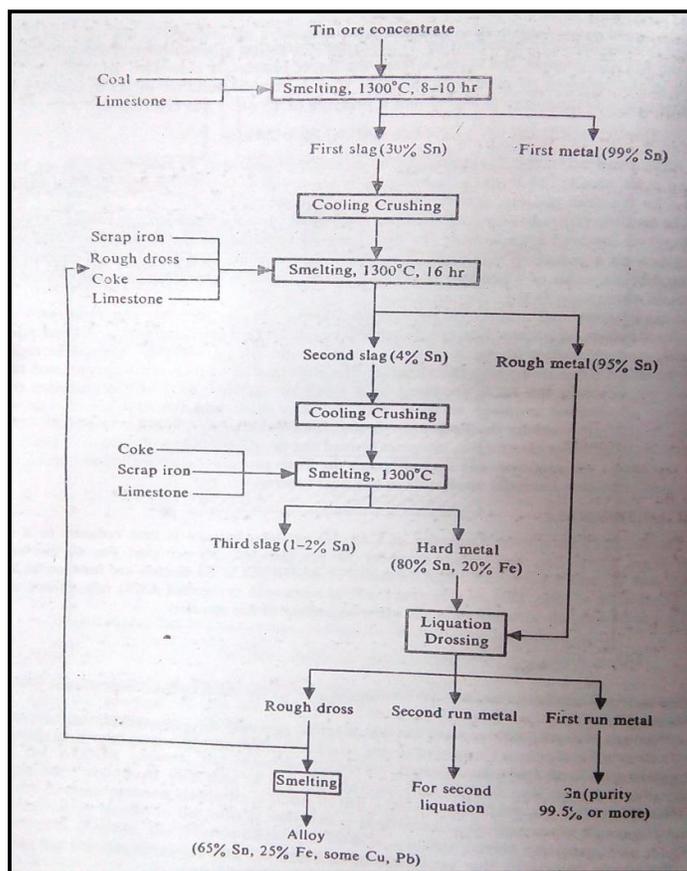
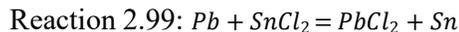


Fig. 2.33: Smelting of Tin Concentrate

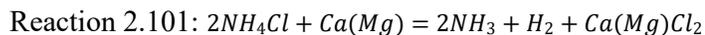
Pulverize coal separate and precipitate out AlCl_3 . Deleading of Sn by SnCl_2 based on



A Sn-Pb alloy is removed from the PbCl_2 - SnCl_2 mixture by melting in contact with Zn.



Any excess Ca or Mg is removed by adding NH_4Cl according to the reaction



Electro Refining of Sn:

Using a mixture of H_2SO_4 and phenol sulphonic acid as electrolytes is feasible. If, Sn contains a large number of impurities such as Pb, Bi, Sb, As, etc. A mixture of Cresol sulphonic acid and phenol sulphonic acid is now used as an electrolyte.

It is analyzed 95% Sn dissolve in pure metal to obtain a cathode sheet of 99.95% Sn.

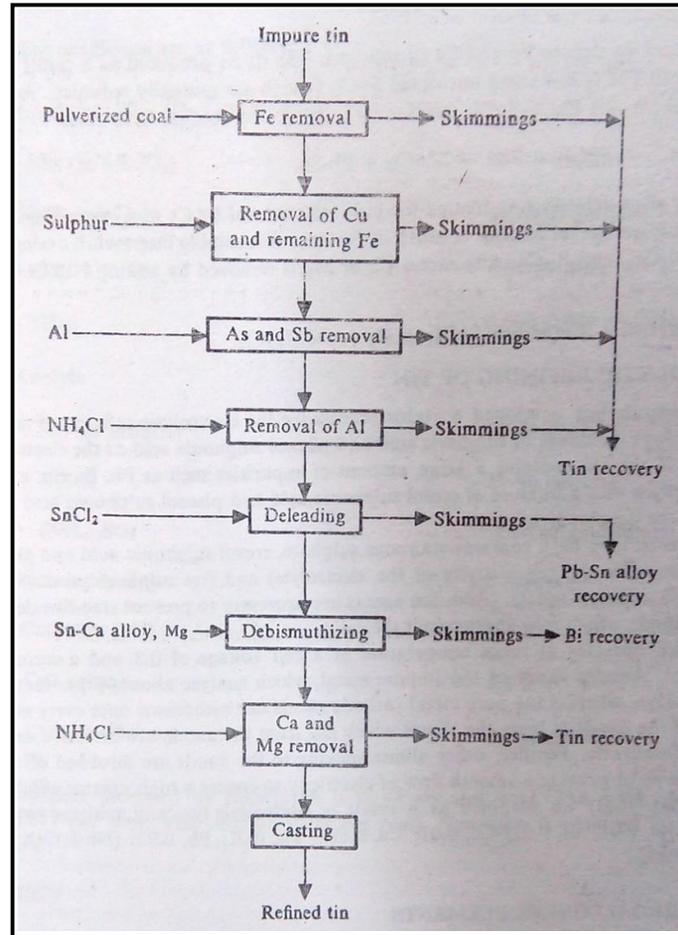


Fig. 2.34: Flow Sheet of Production of Refined Tin

2.9.4 Use of Sn:

- Resistance to both corrosion and toxicity. So, for plating cans, general plating other materials.

2.9.5 Area of Extraction:

- Bolivia
- Malaysia
- Indonesia

2.10 : Extraction of Nb (Colombium):

2.10.1 Details of Nb:

Atomic Number: 41

Atomic Weight: 92.91 amu

Density: 8.57 gm/cm³

Melting Point: 2750 K

Boiling Point: 5017 K

2.10.2 Common Minerals of Nb:

Columbite – (Fe, Mn) (Nb, Ta)₂O₆

Pyrochlore – (Ca, Na)₂(Nb, Ta, Ti)₂O₆(OH,F)

2.10.3 Extraction of Nb:

The extraction of Nb follows the same extraction route as Ta. Fig. 2.35 represents the extraction route of Nb and Ta.

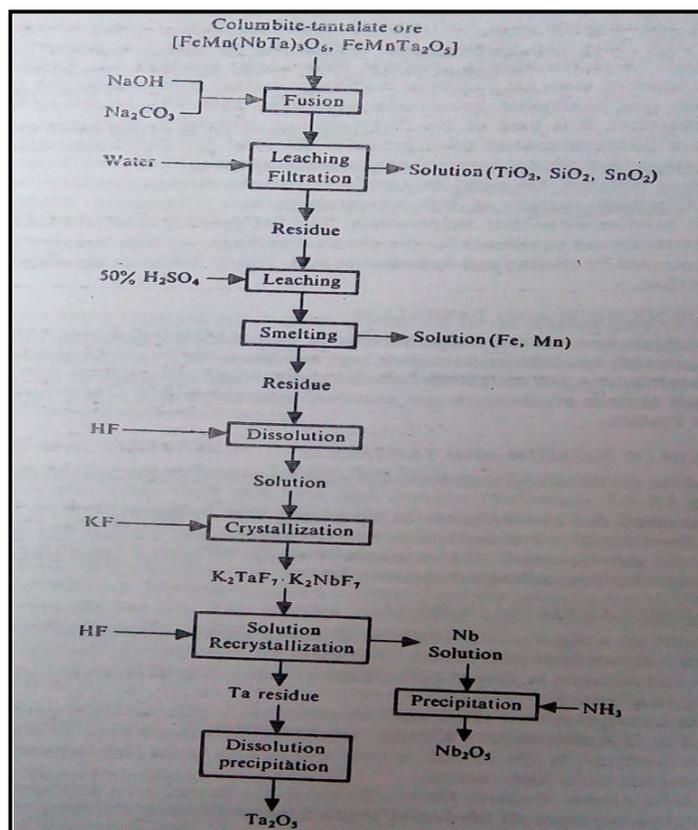


Fig. 2.35: Extraction of Niobium and Tantalum.

2.10.4 Use of Nb:

- Niobium is used in alloys including stainless steel. It improves the strength of the alloys, particularly at low temperatures. Alloys containing niobium are used in jet engines and rockets, beams, and girders for buildings and oil rigs, and oil and gas pipelines.
- This element also has superconducting properties. It is used in superconducting magnets for particle accelerators, MRI scanners, and NMR equipment.
- Niobium oxide compounds are added to glass to increase the refractive index, which allows corrective glasses to be made with thinner lenses.

2.10.5 Area of Extraction:

- Canada
- Brazil
- Australia
- Nigeria

2.11 Extraction of U (Uranium):**2.11.1 Details of Uranium (U):**

Atomic Number: 92

Atomic Weight: 238.03 amu

Density: 19.1 gm/cm³

Melting Point: 1405.3 K

Boiling Point: 4404 K

2.11.2 Common Minerals of Uranium (U):

Uraninite – UO₂

2.11.3 Extraction of Uranium (U):

The extraction route of Uranium (U) follows the flow sheet given in fig 2.36.

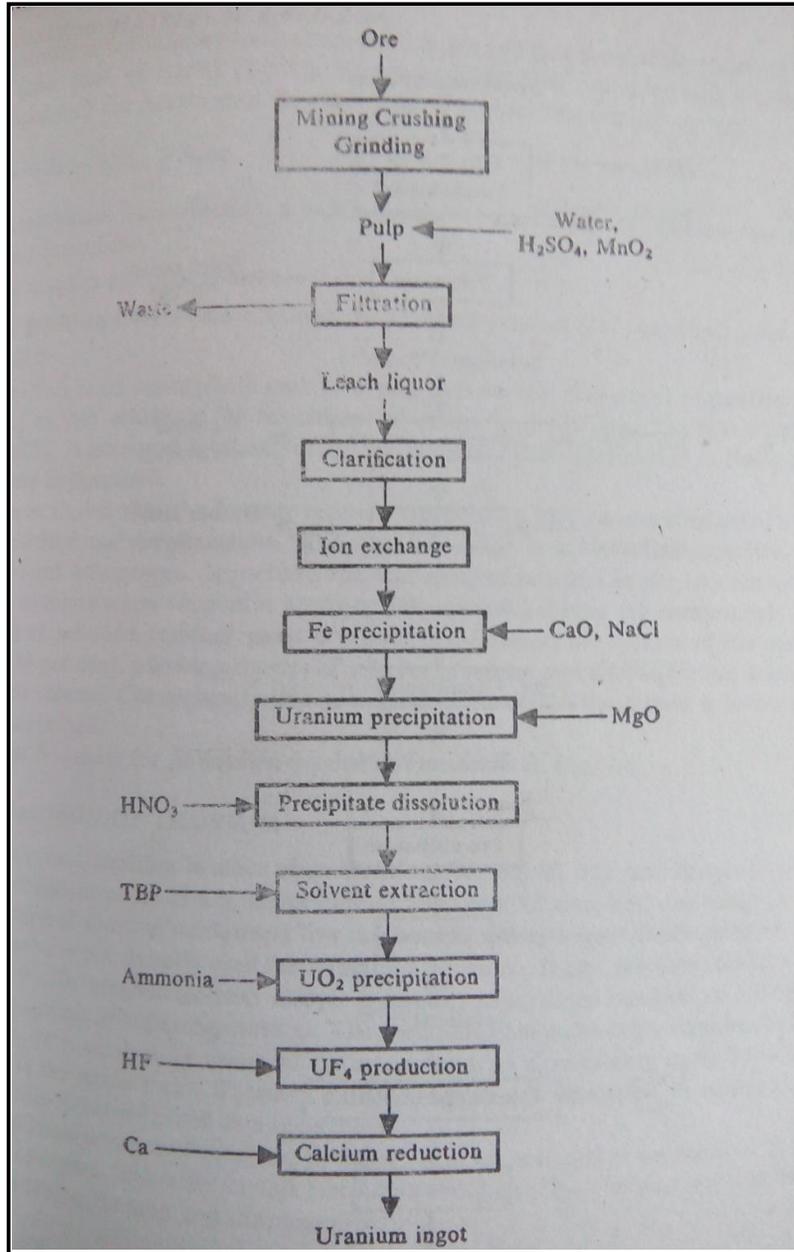


Fig. 2.36: Extraction of Uranium Ingot

2.11.4 Use of Uranium (U):

- Uranium is a very important element because it provides us with nuclear fuel used to generate electricity in nuclear power stations. It is also the major material from which other synthetic trans-uranium elements are made.
- Naturally occurring uranium consists of 99% uranium-238 and 1% uranium-235. Uranium-235 is the only naturally occurring fissionable fuel (a fuel that can sustain a chain reaction). Uranium fuel used in nuclear reactors is enriched with uranium-235. The chain reaction is carefully controlled using neutron-absorbing materials. The heat generated by the fuel is used to create steam to turn turbines and generate electrical power.
- In a breeder reactor uranium-238 captures neutrons and undergoes negative beta decay to become plutonium-239. This synthetic, fissionable element can also sustain a chain reaction.
- Uranium is also used by the military to power nuclear submarines and in nuclear weapons.
- Depleted uranium is uranium that has much less uranium-235 than natural uranium. It is considerably less radioactive than natural uranium. It is a dense metal that can be used as ballast for ships and counterweights for aircraft. It is also used in ammunition and armour.

2.11.5 Area of Extraction:

- Kazakhstan
- Canada
- Australia

2.12 Environmental Pollution and its address related to various Metal Extraction**Processes in General:****Environmental Pollution:**

There are various ways through which environmental pollution occurs such as dust emission, noise pollution, wastewater disposal, fugitive emissions, and poisonous gas emissions etc.

Prevention and Controlling Methods of Environmental Pollution:

- Use of primary controls for the flue gas of the furnace, including venturi scrubbers with complete combustion techniques; with suppressed combustion systems, furnace gas is cleaned and stored to be used as fuel.
- Installation of secondary controls to capture off-gas escaping from the furnace.
- In the casting area (ingots and continuous casting), particulate matter and metals arise from the transfer of molten steel to the mold and from the cutting to the length of the product by oxy-fuel torches during continuous casting. Exhausts should be fitted to filters and other relevant abatement equipment, especially in the casting rolling, and finishing shops, where relevant.
- Bag house filters and ESP have higher particulate collection efficiency, whereas wet scrubbers also allow capturing water-soluble compounds (e.g. sulfur dioxide [SO₂] and chlorides) but need additional effluent treatment facilities.
- Bag filters are typically installed to control melting shop emissions. They are often preceded by cyclones, which are installed to act as spark separators
- Mechanical Actions: Scarfing and grinding activities may generate particulate matter emissions. Exhausts should be fitted to filters chosen based on the specified activity
- Raw Material Handling: To reduce fugitive emissions of particulate matter during handling of materials, the following prevention and control techniques are recommended
- Use indoor or covered stockpiles.
 - water spray system, dust suppressants, windbreaks, and other stockpile management techniques.
 - Design a simple, linear layout for material handling operations to reduce the need for multiple transfer points.
 - Maximize the use of enclosed silos to store bulk powder.
 - Enclose conveyor transfer points with dust controls.
 - Clean return belts in the conveyor belt systems to remove loose dust.