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BUREAU OF MINES AND GEOLOGY

E. G. Koch, *Director*

March 1966

**THE RECOVERY OF ELEMENTAL SULFUR
FROM SULFIDE ORES**

by

Fathi Habashi

Department of Metallurgy

Montana College of Mineral Science and Technology

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T H E R E C O V E R Y O F E L E M E N T A L
S U L F U R
F R O M S U L F I D E O R E S

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ABSTRACT

Pyrometallurgical, hydrometallurgical, and electrolytic processes aimed to recover the metals from their sulfide ores and at the same time the by-product sulfur in the elemental form, are reviewed. Pyrometallurgical methods involve thermal dissociation, reaction of sulfide ores with SO_2 gas, production of sulfur via SO_2 , reduction smelting, and chlorination. Hydrometallurgical methods involve the production of sulfur via H_2S , and the aqueous oxidation of sulfide ores. Electrolytic methods involve electrolysis of molten sulfides, and the electrolysis of sulfide anodes in aqueous phase.

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INTRODUCTION

During the smelting of copper, nickel, lead, and zinc sulfide ores, large amounts of sulfur dioxide are produced. Because the release of this gas in the atmosphere causes pollution problems, many attempts were made in the past for its recovery. In one example, pure SO_2 was obtained from flue gases by an absorption-desorption process, and in another example, the smelting operation was conducted with oxygen instead of air, to get a relatively concentrated SO_2 . In both cases SO_2 recovered was either liquified or processed to sulfuric acid. Although these two forms of sulfur are the usual forms that are consumed by the chemical and metallurgical industries, yet their storage and transport usually raise economic problems. Unless they are used in the smelter itself, or a market exists near the smelter, the process will be uneconomical.

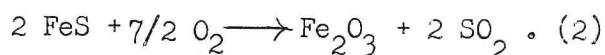
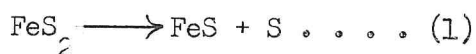
It was realized, long ago, that a process by which sulfur can be recovered directly in the elemental form would be most attractive. This applies not only to the processing of nonferrous metals but also for pyrite or pyrrhotite ores whose shipping costs to sulfuric acid manufacturers may be, in some cases, prohibitive. Elemental sulfur is easily stored and transported, weighs only one half an equivalent quantity of SO_2 and one third as much as the corresponding quantity of sulfuric acid, and is readily converted to any of these forms when needed.

In this paper, an attempt will be made to review all the processes that are aimed to recover the metals from their sulfide ores, and at the same time the by-product sulfur in the elemental form.

PYROMETALLURGICAL METHODS

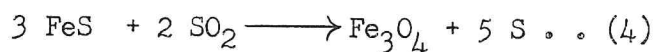
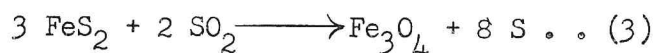
1. Thermal Dissociation

To reduce the shipping costs of pyrite, the Noranda Company in Canada has designed a plant in which pyrite is first heated in absence of air to distill off one atom of sulfur in the elemental form, then roasting the residue to SO_2 for sulfuric acid manufacture (i.e.)



2. Reaction of Sulfide Ores with SO_2

An improvement in the thermal dissociation method is to utilize sulfur dioxide in distilling off elemental sulfur from pyrite or pyrrhotite at 800-900°C according to:



Sulfur dioxide for the reaction can be produced by burning some of the pyrite with air. The process was developed by the Duisburger Kupferhütte in Germany (Schackmann, 1954). Under these conditions, non-ferrous metals present in the pyrite are transformed into sulfates, which can be recovered from the cinder by leaching (Fig. 1). Thermodynamic and kinetic studies of the above reactions were conducted by Wöhler et al. (1923), Rosenqvist and Hyne (1953), Morawietz (1954), Schmahl (1954), Iriarte et al. (1963), and Pannetier and Davignon (1963).

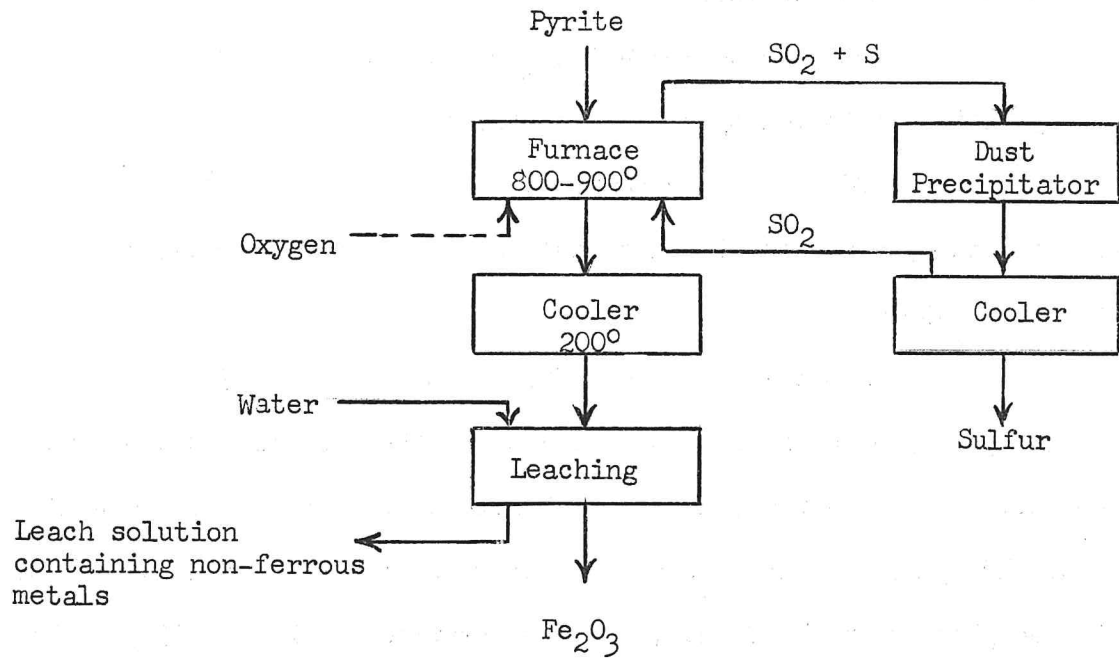


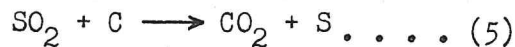
Fig. 1 - Recovery of Elemental Sulfur from Pyrite by Reaction with SO_2 Gas. The Duisburger Kupferhütte Process.

3. Production of Sulfur via SO_2

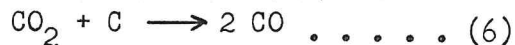
Sulfur dioxide can be reduced to elemental sulfur by coke, methane, or natural gas at about 1200°C .

a) Reduction by coke

The reaction



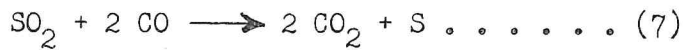
is exothermic. However, the conditions are favorable for the reaction:



to take place, which is endothermic. To achieve the required temperature in the reduction, external heating must be supplied, and the simplest

method is by introducing oxygen to utilize the exothermic combustion of carbon.

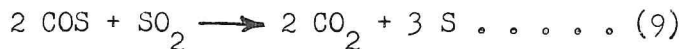
Carbon monoxide formed according to equation (6) reacts with SO₂ at 600°C to form elemental sulfur:



Carbon oxysulfide is formed by a side reaction:



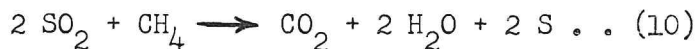
However, it can be converted to elemental sulfur easily when passed over bauxite at 425-450°C:



Coke consumption, which is the major factor in the process, is about 0.4 tons/ton sulfur. The thermodynamics and kinetics of this process were discussed by Lepsoe (1938, 1940). The process was operated by the Consolidated Mining and Smelting Company in Trail, Canada, and was described by King (1950). The plant was shut down in 1943 because SO₂ was needed for making H₂SO₄ locally to meet the fertilizer demand. Before the shut down, the production was 150 tons/day of sulfur of 99.99% purity.

b) Reduction by methane and natural gas:

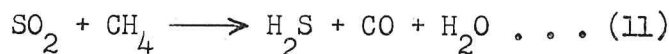
Sulfur dioxide is reduced by methane at 1250°C to elemental sulfur according to:



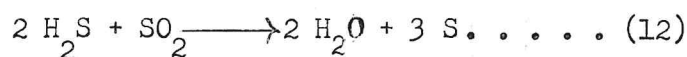
Reduction with methane was tried by Young (1915), and later natural gas was used by Fleming and Fitt (1942) on a pilot scale (5 tons/day sulfur) in the 1940's for 4 ½ years at the American Smelting and Refining Company in Garfield, Utah. The gas contained 5.5-7.25% SO₂ and the sulfur produced was 99.9% pure. A plant description was given by Fleming and Fitt (1950).

The combustion takes place in a large vertical steel chamber lined with refractory and insulating bricks. In order to obtain surface for complete combustion, the chamber is packed with a checker work of refractory brick.

Small amounts of hydrogen sulfide and carbon oxysulfide are formed by side reactions:



Both can be converted to elemental sulfur by passing the gas mixture over a suitable catalyst at the right temperature. Carbon oxysulfide reacts according to equation (9), and hydrogen sulfide reacts at 210-235°C in contact with activated alumina, according to:



Sulfur is formed in the gases as a mist and is collected in electrostatic precipitators. A flowsheet is shown in Fig. 2. The process is simple and efficient; it was shut down only because of shortage in the supply of natural gas.

4. Reduction Smelting

Elemental sulfur was first successfully recovered as a by-product from the blast-furnace smelting of pyritic copper ore by Orkla Grube Aktiebolag at a small plant at Lokken, Norway in 1928. The success of this pilot plant led to the construction in 1932 of a large modern smelter with four blast furnaces. Similar operations are in Ronnskar (Sweden), Mina de S. Domingos (Portugal), Rio Tinto (Spain), and USSR.

This method is only economical for pyrites containing copper. The ore is mixed with coke, quartz, and limestone and heated in a blast furnace (Fig. 3). Copper is recovered as a molten matte, and iron is eliminated

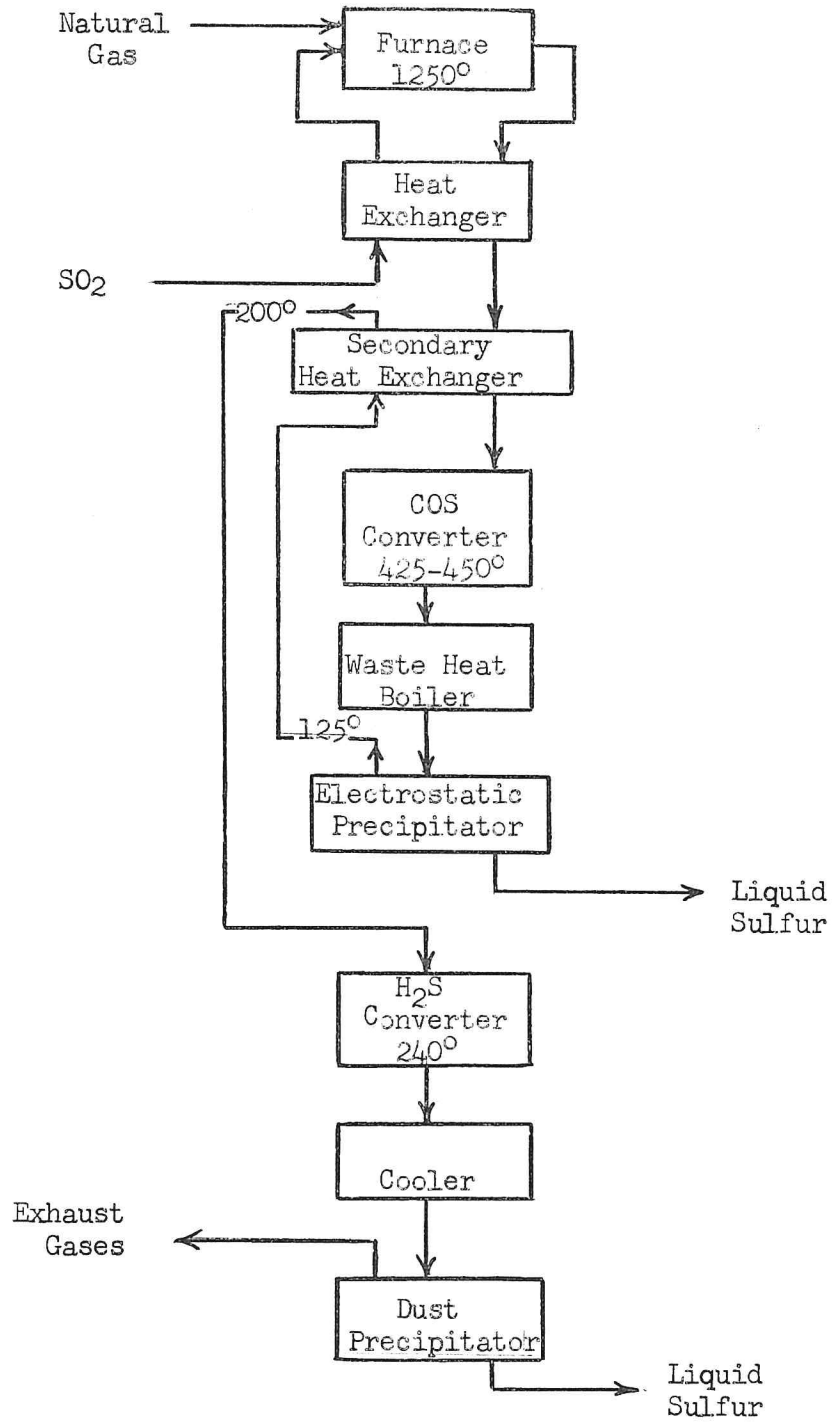


Fig. 2 - Reduction of SO₂ to Elemental Sulfur by Natural Gas.

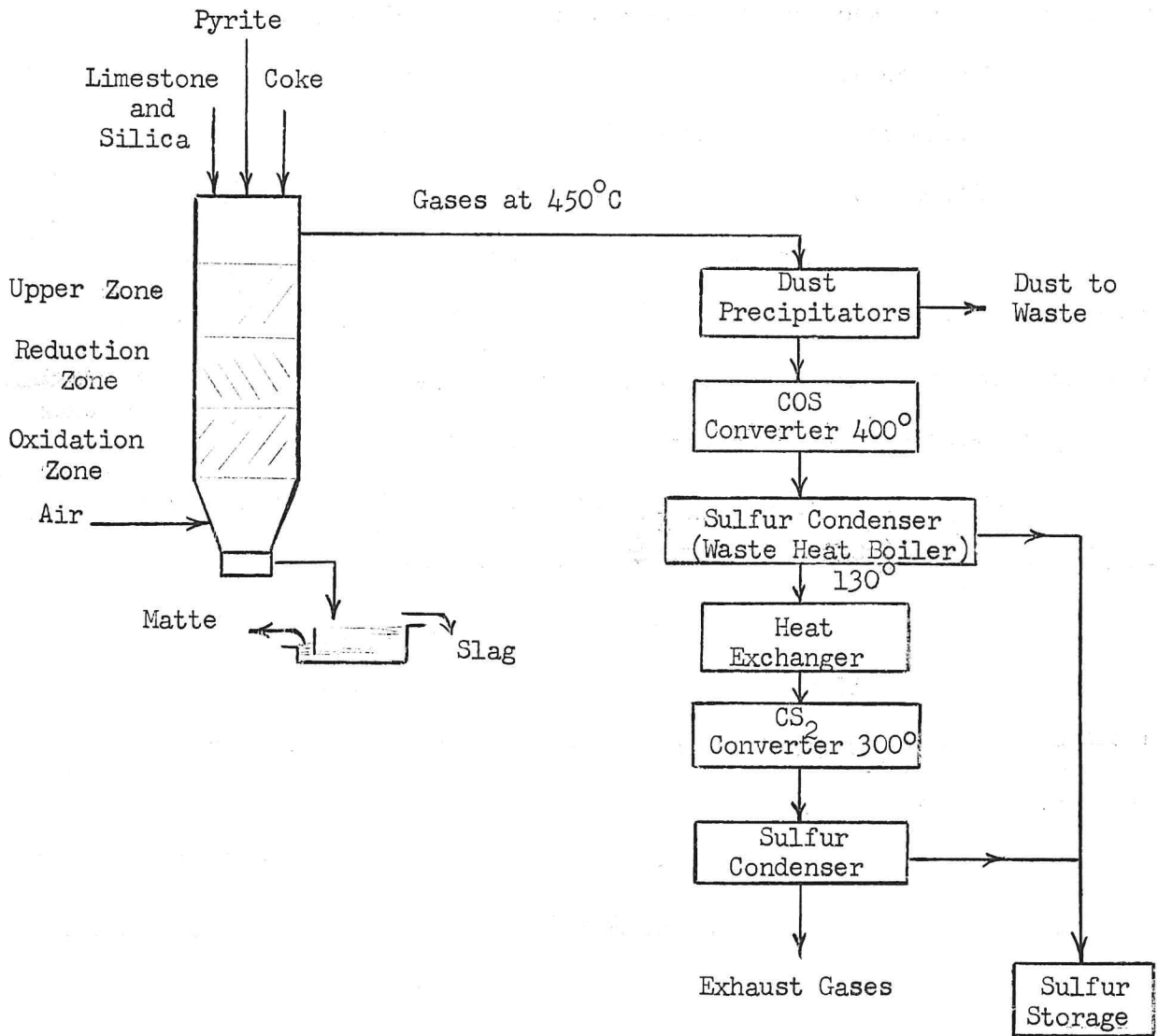
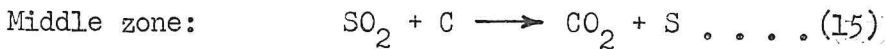
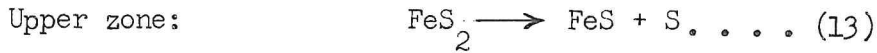


Fig. 3 - Recovery of Elemental Sulfur and Copper from Pyrite by Reduction Smelting (The Orkla Process).

in the slag. In the upper part of the furnace, one atom of sulfur in pyrite is distilled as elemental sulfur. In the oxidizing zone, FeS formed is oxidized to ferrous oxide and SO₂. In the middle part of the furnace, the reduction zone, SO₂ is reduced by coke to elemental sulfur, which is volatilised as vapor. The reactions taking place can be represented by the following equations:



The method has the disadvantage that only copper is recovered; iron and other metals present in pyrite, e.g. cobalt and zinc, are not recovered.

The matte produced contains 6-8% Cu, is unsuitable for direct converting, and is usually re-smelted with coke, silica, and limestone to 40% Cu.

Carbon disulfide and carbon oxysulfide are formed in the furnace; they are, however, converted to elemental sulfur on catalytic beds;

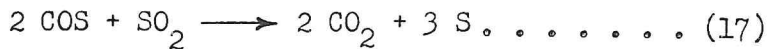
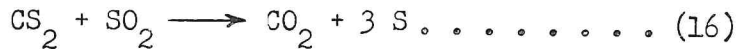


Table 1 gives average analysis of the ore treated in Norway, and the composition of slag, matte, and the exit gases from the furnace.

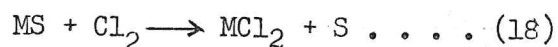
Table 1 - The Orkla Process (after Kiaer, 1954)

<u>Analysis of ore</u>		<u>Slag and matte</u>			<u>Exit gases from furnace</u>		
	%		Slag %	Matte %		% Vol.	g/M ³
S	41.0	Cu	0.3	38.50	S	---	300
Cu	1.8-2.5	Fe	42.0	32.46	CO	13.2	
Fe	38.0	S	2.3	25.55	CO ²	0.2	
Zn	1.9	SiO ₂	33.5	0.14	O ₂	0.5	
Co	0.1	Zn	1.6	---	SO ₂		75.0
SiO ₂	12-16	CaO	4.5	---	H ₂ S		12.3
Al ₂ O ₃	0.3	Al ₂ O ₃	1.4	---	CS ₂		28.8
CaO	0.8	MgO	0.13	---	CO ₂		22.3
MgO	0.3	Co	---	0.11			
As	0.05						
Se	0.005						
Ag	20.0 g/ton						
Au	0.2 g/ton						

Ores containing arsenic, as is the case in Rio Tinto, are difficult to treat. Sulfur vapor containing a large percentage of arsenic starts to condense at 350°C, but arsenic-free vapor condenses at a much lower temperature. Further, liquid sulfur containing arsenic is very viscous and therefore is difficult to handle. Owing to these facts, passing the gases over catalyst beds for converting CS₂ and COS to S is not successful.

5. Chlorination of Sulfide Ores

Chlorine reacts with metal sulfides readily to form metal chlorides and elemental sulfur:



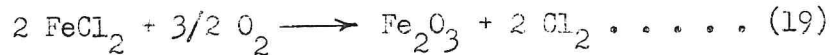
The use of chlorine for the treatment of sulfide ores is an old technique, and much work was published in this connection. However, no plants are at present operating, although the process is very promising. The reaction can be conducted in two ways:

- a) Low-temperature chlorination at 60-100°C, i.e., at temperature below the melting point of sulfur so that the liberated sulfur does not melt and agglomerate the charge. In the past, seemingly no attempts were made to recover the sulfur.
- b) High-temperature chlorination at 400-700°C. In this process sulfur is volatilized, condensed, and recovered.

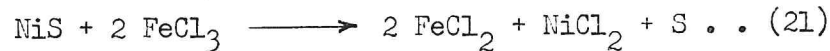
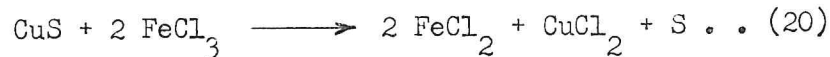
The chlorination reaction is strongly exothermic, and therefore, once started, it would require no external heat. The required reaction temperature can be controlled simply by adjusting the flow of chlorine to the reactor. In both processes, the non-ferrous metals are converted to chlorides and are recovered usually by leaching. The leach solution is first purified

from iron by adding zinc oxide to precipitate ferric hydroxide. Each metal is then separated by successive cementation, e.g., gold and silver by copper, copper by lead, and lead by zinc. The final solution, containing only zinc, is evaporated under vacuum to anhydrous zinc chloride, which is then electrolyzed in the fused state to metal and chlorine. Molten zinc cathode and graphite anode are used. The chlorine is collected and reused for the process.

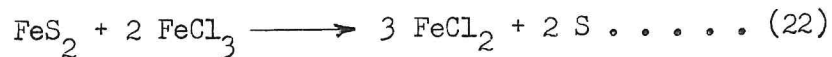
Practically all sulfide ores contain pyrite, which is also easily chlorinated, but there is no chlorine lost in the process, as FeCl_2 formed is readily decomposed by oxygen liberating chlorine as follows:



Also, if any FeCl_3 is formed, it itself acts as a chlorinating agent for the other sulfides in the ore:



It also reacts with pyrite according to the equation:



Historically, the high-temperature chlorination process was developed first. Thus, as early as 1897 Ashcroft patented this process, and a plant was operated at Broken Hill, Australia, for the treatment of a lead-zinc ore.

In USA, John L. Malm of The Western Metal Company developed the low-temperature chlorination process, and a 10 t/day plant was in operation in Corbin, Montana, and another 50 t/day plant at Georgetown, Colorado. The Bunker Hill & Sullivan Mining and Concentrating Co. planned a plant at Kellogg, Idaho, but the idea was not fulfilled, seemingly because of financial difficulties. The process is known as the Malm process and was described by Traphagen (1909) and Herrick (1910).

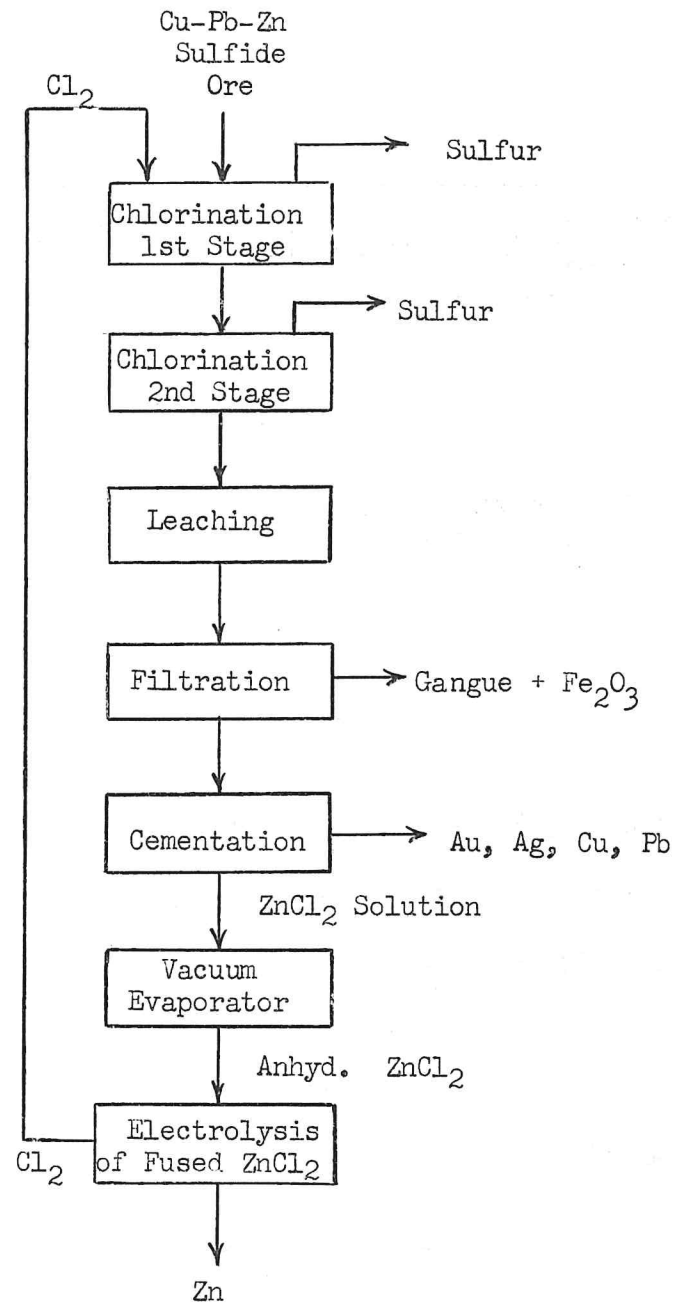


Fig. 4 - A Modified Malm process. (Ionides, 1916).

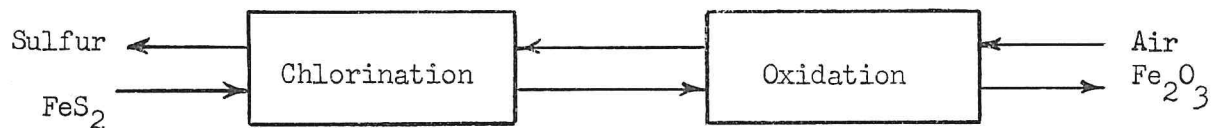


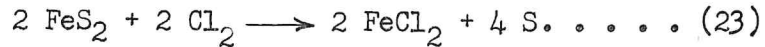
Fig. 5 - Recovery of Elemental Sulfur from Pyrite by Chlorination (Comstock-Wescott Process).

Ionides (1916) described a modified Malm Process, which was operated by the Midwest Metals Co. Chlorination was conducted in two stages--in the first stage by chlorine gas, and in the second by simple roasting so that any $FeCl_3$ formed in the first stage reacts with the non-ferrous metal sulfides. Fig. 4 shows a flowsheet of the process. Ores treated by this process had the following analysis:

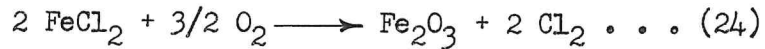
Pb	9.0%	
Zn	2.5%	
Cu	0.5%	
Ag	3.0	Oz/t
Au	0.05	Oz/t

The high-temperature process was applied to pyrite containing small amounts of lead and zinc (Levy and Gray, 1928). Later, the same process was applied to pyrite for the sole purpose of the recovery of elemental sulfur. Thus, in 1936 a plant was operated in Canada by the Comstock-Wescott Inc.,

Niagara Falls, in which pyrite was treated with chlorine gas in rotary kilns to volatilize elemental sulfur according to:



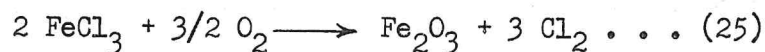
Ferrous chloride was oxidized by air to generate the chlorine for recycle:



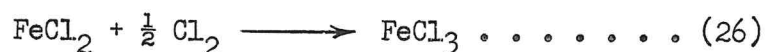
The plant was described by Rowe (1938) and the process is covered by numerous patents. Fig. 5 shows a flowsheet of the process.

The high-temperature process was studied further by Zielinsky (1931), Urazov et al. (1935), Pristoupil (1937), Ezdakov (1939), Chowdhury and Hossain (1945), Hayek and Kraus (1950), Khundkar and Quadr (1951) and Biswas et al. (1951-55).

Recently, a pilot plant was operated by Hohn et al. (1954-58) for the treatment of pyrite containing nickel and copper. It was possible to recover sulfur in the elemental form, and the nonferrous metals were converted to chlorides and leached from the iron oxide residue. Instead of rotary kilns, fluidized beds were used. The same authors also treated an ore containing 85.6% FeS_2 , 2.7% cassiterite, 6.3% zinc blende, and 5.4% gangue, and were able to get two separate fractions: 1) pigment grade Fe_2O_3 and 2) SnO_2 and ZnCl_2 fraction, which after leaching the zinc analyzed 63.8% gangue, 33.2% SnO_2 , and 2.9% Fe_2O_3 . This was achieved by admitting only the required amount of ferric chloride vapor to expel the sulfur from the ore, and the rest of the ferric chloride was oxidized in a separate chamber to recover chlorine and pigment grade ferric oxide:



Chlorine is then allowed to react with ferrous chloride in a third chamber to volatilize pure FeCl_3 :



Under these conditions, cassiterite is not chlorinated, and therefore can be concentrated in a separate fraction. Fig. 6 shows a flowsheet of the process.

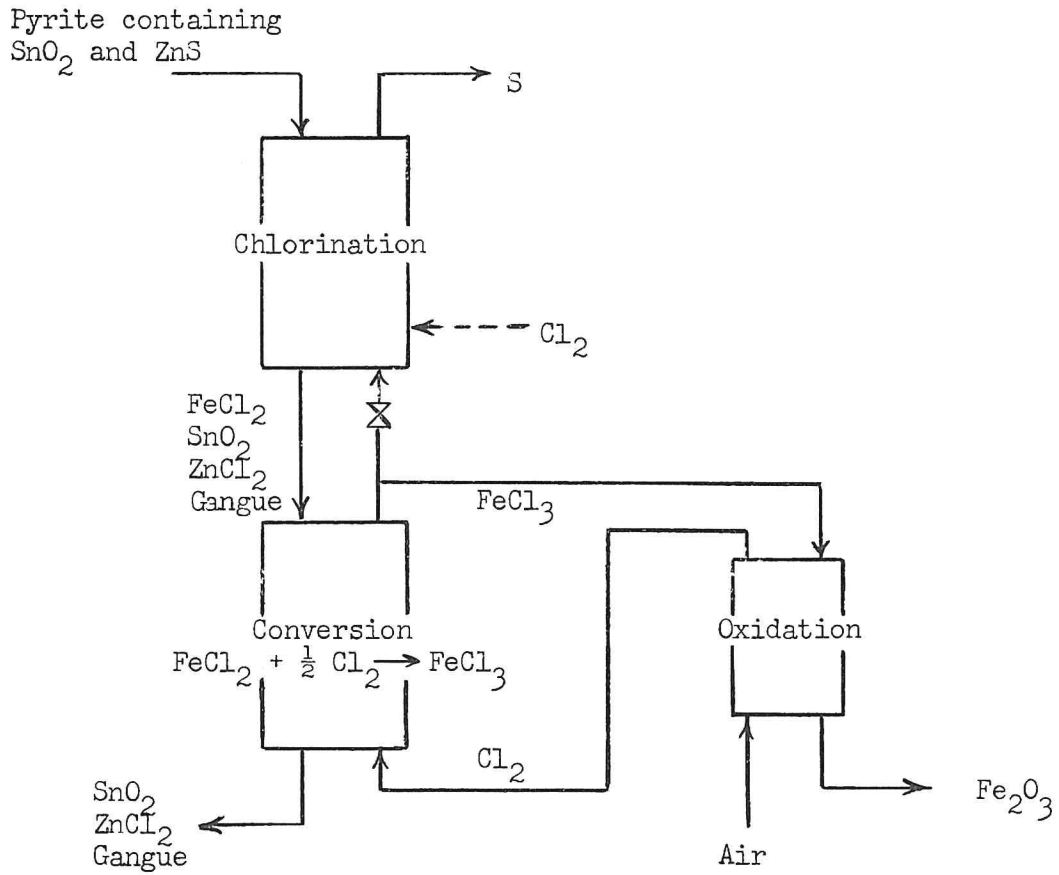


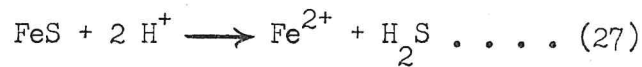
Fig. 6 - Recovery of Elemental Sulfur, Tin, and Zinc from Pyrite Ore (Hohn et al. 1957).

The US Bureau of Mines is showing interest in these processes, and work was conducted along similar lines by Kershner (1961), Donaldson (1962), and Starliper and Kenworthy (1964).

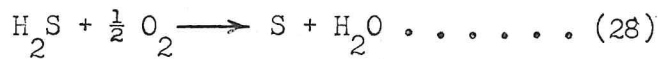
HYDROMETALLURGICAL METHODS

1. Production of Sulfur via H₂S:

Pyrrhotite or ferrous sulfide produced by the thermal dissociation of pyrite, reacts readily with dilute acids to liberate hydrogen sulfide:



The gas is collected and oxidized in a Claus furnace to elemental sulfur according to:



During the decomposition of FeS with acid, the non-ferrous metal sulfides are not dissolved, and therefore can be recovered from the unreacted residue. Fig. 7 shows a flowsheet of the process.

Lindenlaub (1954) treated a zinc blende concentrate :

Zn	50 %
Pb	4
Fe	6.5
S	31

with 60% H₂SO₄ at 150-160°C. Reaction gas containing 30-40% H₂S was passed through a Claus kiln. The residue, after washing and filtration, contained all the lead, silver, and gold, and was processed for recovery. The filtrate was treated with H₂S to precipitate copper and cadmium, which are then processed and recovered electrolytically. Spent electrolyte was concentrated and recycled.

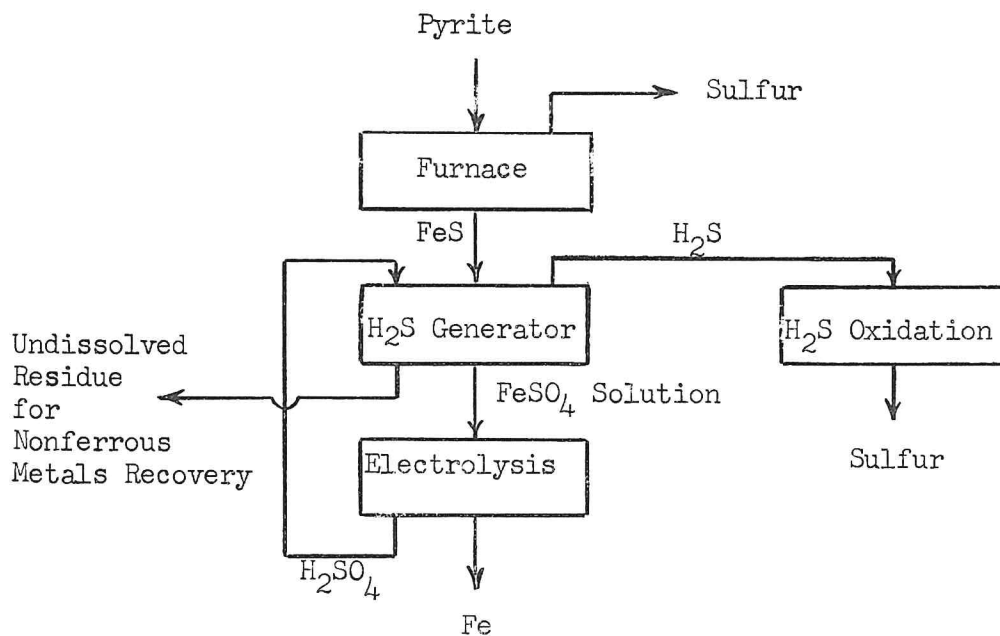
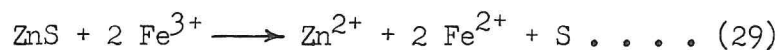


Fig. 7 - Recovery of Elemental Sulfur from Pyrite Via H₂S Formation.

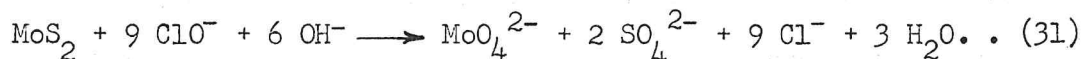
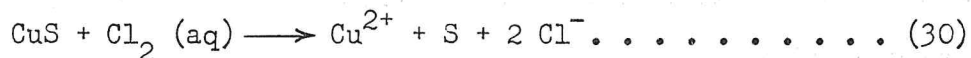
2. Aqueous Oxidation of Sulfide Ores:

Ferric sulfate or ferric chloride solutions dissolve metal sulfides at room temperature with the liberation of elemental sulfur:

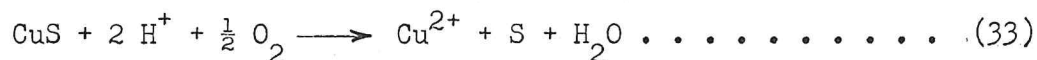
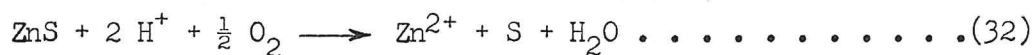


Ferrous ion formed is oxidized by air back to ferric, thus allowing the recycle of the leaching agent. The main difficulty with this process is the hydrolysis of ferric ion and precipitation of hydrated ferric oxides, if the pH of the solution exceeds 3.5. Other oxidizing agents were suggested

to replace ferric ion. Thus chlorine water, sodium hypochlorite, nitrates, etc., were used, e.g.

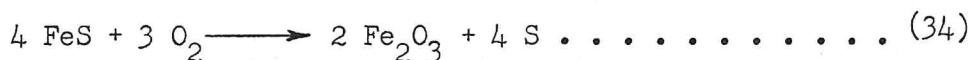


Recently, air or oxygen under pressure was used as the oxidizing agent, at temperature below 120°C, the m.p. of sulfur. Such oxidizing agents have the advantage of not contaminating the leach solution with foreign ions, e.g.:



Elemental sulfur is recovered from the unleached residue by melting and filtration.

Pyrrhotite was also treated in a similar way (Downes and Bruce, 1955), but owing to the hydrolysis of ferric ion even at low pH, elemental sulfur and hydrated ferric oxide were obtained:



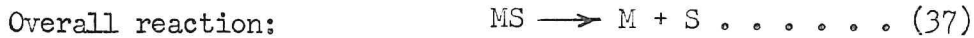
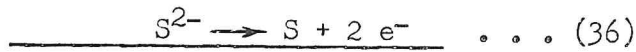
Nonferrous metals present in the pyrrhotite go into solution and can be recovered. Ferric oxide, which is now free from non-ferrous metals, can be readily dried, sintered, and charged to blast furnace.

ELECTROLYTIC METHODS

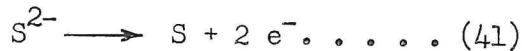
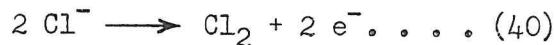
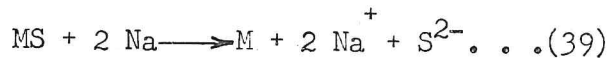
1. Electrolysis of Molten Sulfides

In the process for the chlorination of sulfide ores described earlier, elemental sulfur and the metal chloride were obtained; the latter was electrolyzed in the fused state to metal and chlorine, which was recycled.

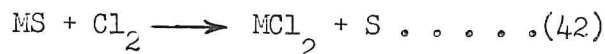
In 1906, it was found by Townsend, and Valentine and Betts that the sulfide can be electrolyzed directly in the molten state to yield the metal and elemental sulfur. As most metal sulfides have either a high melting point or decompose upon melting, electrolysis is usually carried out in a metal chloride-alkali chloride bath to which the metal sulfide is added. The decomposition potential of the sulfides is usually much less than that of the chloride components of the bath, thus allowing the deposition of the metal at the cathode and elemental sulfur at the anode:



Because the solubility of most metal sulfides in metal chlorides at 700 to 800°C is usually low, electrolysis is conducted when the sulfide is either in form of suspension in the electrolyte or as a molten layer floating above the electrolyte. The deposition of the metal at the cathode, and sulfur at the anode takes place possibly by secondary reactions as follows:



Chlorine liberated at the anode does not interfere with the process, as it will react with the metal sulfide as follows:



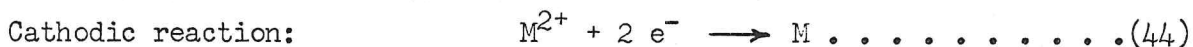
Theoretical studies on the electrolysis of sulfides were published by Truthe (1912) and Savelsberg (1940) and laboratory experiments as follows:

<u>Sulfide</u>	<u>Reference</u>
PbS	Valentine and Betts (1906) Isgaryshev and Grigor'ev (1936) Angel and Garnum (1945) Winterhager and Kammel (1956) Guld'in et al. (1960) Kammel (1961)
Cu ₂ S Sb ₂ S ₃	Hoar and Ward (1958) Vivian (1936) Kammel (1961)

Richardson (1936) described briefly a semi-commercial plant in Wales recovering lead 99.95% purity and sulfur 99.5% pure from galena by this method. The plant was operated by the National Smelting Company (1934-35), who has many patents in this connection.

2. Electrolysis of Sulfide Anodes in Aqueous Phase:

The previous method of electrolyzing sulfides in the molten state has the disadvantage of operating at temperatures in the range 500-900°. A much simpler method is to cast the sulfide in form of anodes, and electrolyze it using an aqueous electrolyte at room temperature. The process can be represented by the equations:



The metal deposits at the cathode, while elemental sulfur remains adhering to the anode in form of a sludge, which can be melted, separated, and recovered. This process has a great advantage over conventional methods of metal recovery. Thus, in the case of nickel extraction it would be possible to cast the nickel matte in form of anodes, and recover the metal and the sulfur directly in one step, instead of roasting the matte to oxide, reduction of the oxide to metal, and then casting the metal in form of anodes for electrolytic refining. It should also be noted that this direct electrolysis

of sulfide anodes not only by-passes the roasting and reduction step, but it is possible to recover the sulfur directly in the elemental form.

As early as 1882, Marchese patented a process based on the above idea, in which a matte having the composition: Cu 15, Pb 14, Fe 41, and S 25%, was electrolyzed. According to Engelhardt (1932) the process was applied on a commercial scale by the Società Anonima Italiana di Miniere di Rame e di Elettrometallurgia at Casarza near Genua, Italy, and by the Aktiengesellschaft für Bergbau-, Blei- und Zinkhüttenbetriebe at Stolberg, Rheinland, Germany. The process was unsuccessful, however, owing to the contamination of the electrolyte during electrolysis. Borchers et al. (1906) developed the process further by using the white metal (Cu 78.2%, S 19.6%) instead of the matte, and a pilot plant was operated by the Mansfeldschen Kupferschiefer Company at Eisleben, Germany. It appears, however, that the process never went into commercial operation. Further studies in this direction were carried out by the Russian workers Loshkarev et al. (1945), Chizhikov et al. (1948, 1956, 1963), Furtunatov and Mikhailovskaya (1951), and Klochko and Mironova (1954).

The International Nickel Company of Canada, to apply the process to nickel matte, erected a pilot plant in 1951 and full scale plant at the Thompson Refinery in Manitoba in 1964. In this method nickel sulfide matte is melted at 980°C and cast into anodes having the dimensions 28½" x 43½" x 1 3/4". The anodes are allowed to cool in the molds to 540°C and then promptly placed in a controlled cooling box to maintain a desired cooling rate. This step is a critical operation in preparing the anodes, otherwise the castings crack, owing to the phase transformation in Ni₃S₂ that takes place at 505°C. The analysis of anodes and electrolyte is as follows:

<u>Anodes</u>		<u>Electrolyte (pH = 4)</u>	
	%		g/l
Ni	76.0	Ni ²⁺	60
Cu	2.6	SO ₄ ²⁻	100
Co	0.5	NaCl	100
Fe	0.5	H ₃ BO ₃	20
S	20.0		

Boric acid acts as a buffer to minimize variations in the pH. The anodes corrode smoothly and uniformly, and the sludge formed is granular and porous and remains adhering on the anode. The anode increases in thickness during electrolysis, owing to the formation of elemental sulfur. By the end of electrolysis the thickness is usually doubled. Also, the cell voltage rises from 3.0 volts at the start to 5.0 volts at the end of electrolysis. The temperature of electrolyte is 55-60°C. The anodes can be electrolyzed to about 10% scrap, and the efficiency during the anode dissolution is 95%. To facilitate the handling of the scrap anodes containing the sludge, they are enclosed in bags during electrolysis.

During electrolysis, Fe, Cu, Co, As, and Pb originally present in the anodes go into solution, while the precious metals are collected in the sulfur sludge. The electrolyte is regenerated by first oxidation and hydrolysis of iron, then selective oxidation with chlorine and hydrolysis of cobalt, arsenic, and lead, and finally cementation of copper by nickel powder. The nickel content of the electrolyte and the pH are adjusted and it is recycled (Fig. 8).

The sulfur sludge recovered during electrolysis, after filtration and washing to remove the electrolyte, analyzes as follows:

Elemental sulfur	97.0%
Sulfide sulfur	0.7
Nickel	1.25
Iron	0.6
Copper	0.3
Selenium	0.15
Precious metals	variable
Moisture content	20.0

The first step in the purification of the sulfur sludge (Fig. 8) involves melting and filtering. The filtrate is pure sulfur containing only 0.15% Se. The residue, analysing 50% sulfur, contains all the precious

metals and base metal values. Sulfur is purified from selenium by distillation in a 60-trays bubble cap tower, whereby a product containing 5 ppm Se is obtained at the top of the tower. At the bottom of the tower a fraction is drawn containing 20% Se. The residue containing the precious metals is melted and cast into anodes; these analyse as follows:

S	27.2%
Ni	50
Cu	20
Fe	2
Ag	0.5
Precious metals }	0.3

These anodes are electrolysed in the same way as the primary nickel sulfide anodes. During electrolysis a sludge is obtained containing 90% S and 10% precious metals. This sludge is again melted and filtered to separate sulfur from the precious metals concentrate. Fig. 9 and 10 show a general view of the INCO Thompson Refinery and the electrolytic plant.

Similar work is under way at Montana College of Mineral Science and Technology in Butte, Montana, USA, in which the white metal (Cu_2S) is cast in form of anodes and electrolyzed in CuSO_4 solution to deposit copper at the cathode and collect sulfur at the anode.

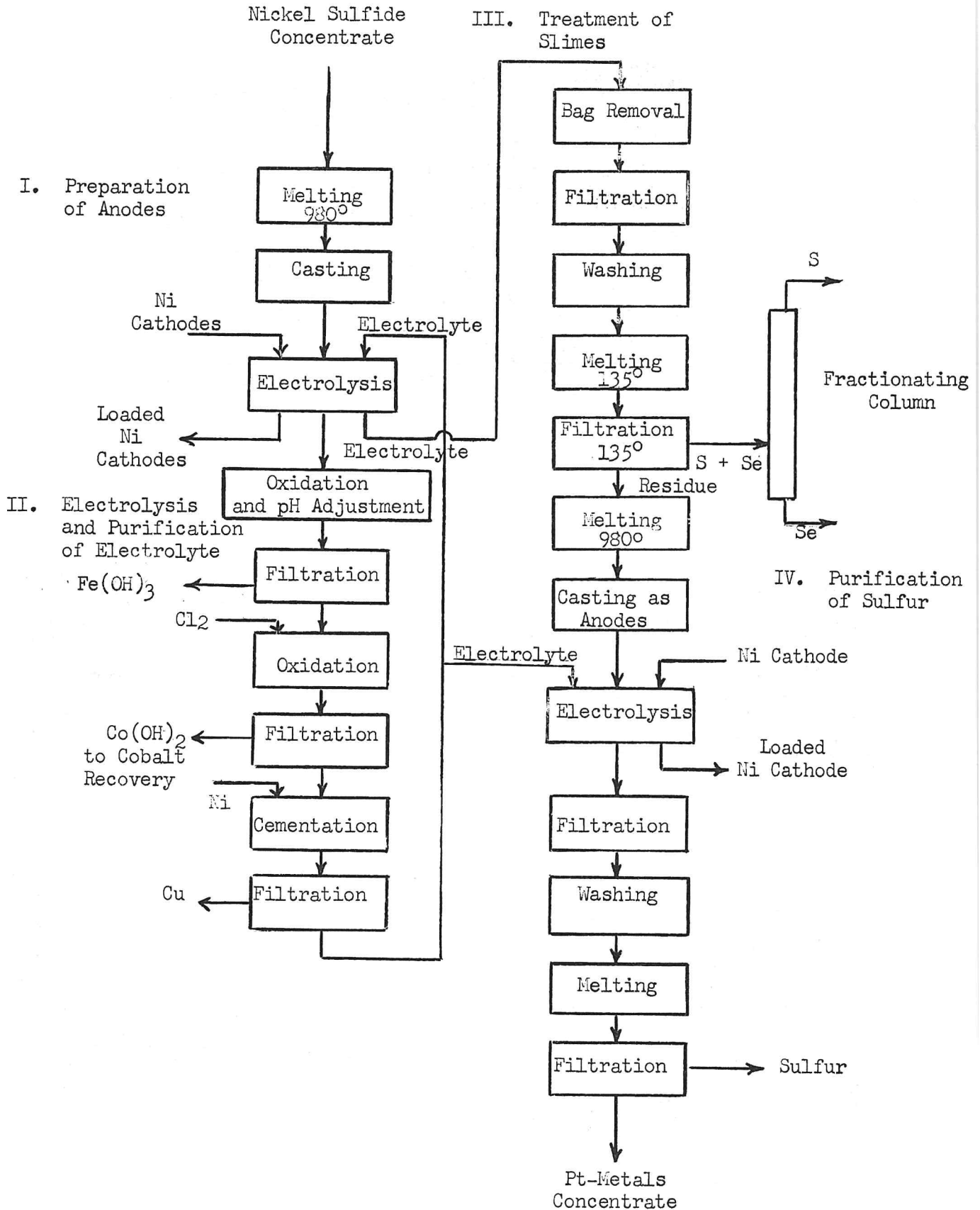


Fig. 8 Recovery of Nickel and Elemental Sulfur by Electrolysis of Nickel Sulfide Anodes (INCC Process).

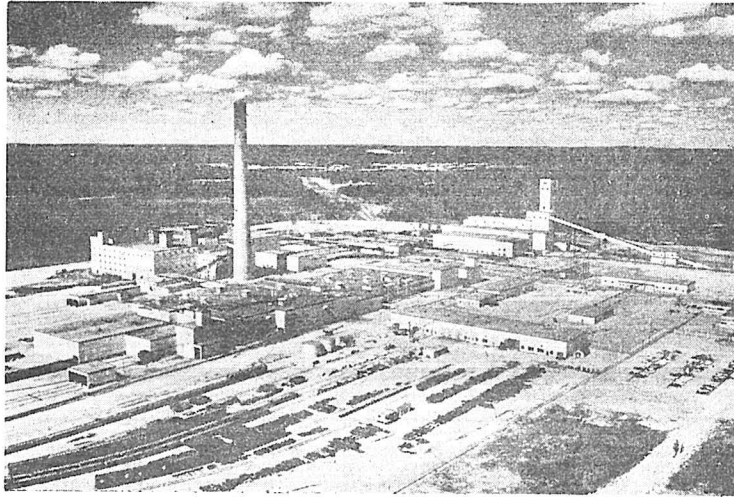


Fig. 9- The International Nickel Company Thompson Refinery, Manitoba, Canada. General View.

- | | |
|--------------------------------------|---|
| (1) Transformers | (14) Nickel carbonate manufacture |
| (2) Rectifiers | (15) Cobalt oxide purification |
| (3) Corroded anode handling | (16) Cobalt oxide production |
| (4) Sulphur filtration | (17) Chlorine unloading |
| (5) Tankhouse service area | (18) Ni shot storage and grinding |
| (6) Plating tanks | (19) Soda ash unloading and storage |
| (7) Nickel hydrate production area | (20) H ₂ SO ₄ storage |
| (8) Anolyte well tanks | (21) Vacuum pumps |
| (9) Iron and arsenic removal | (A) Holding furnaces |
| (10) Cobalt removal | (B) Anode moulds |
| (11) Copper removal | (C) Annealing boxes |
| (12) Purified electrolyte head tanks | (D) Sulphur burn-off kiln |
| (13) Shearing and shipping area | |

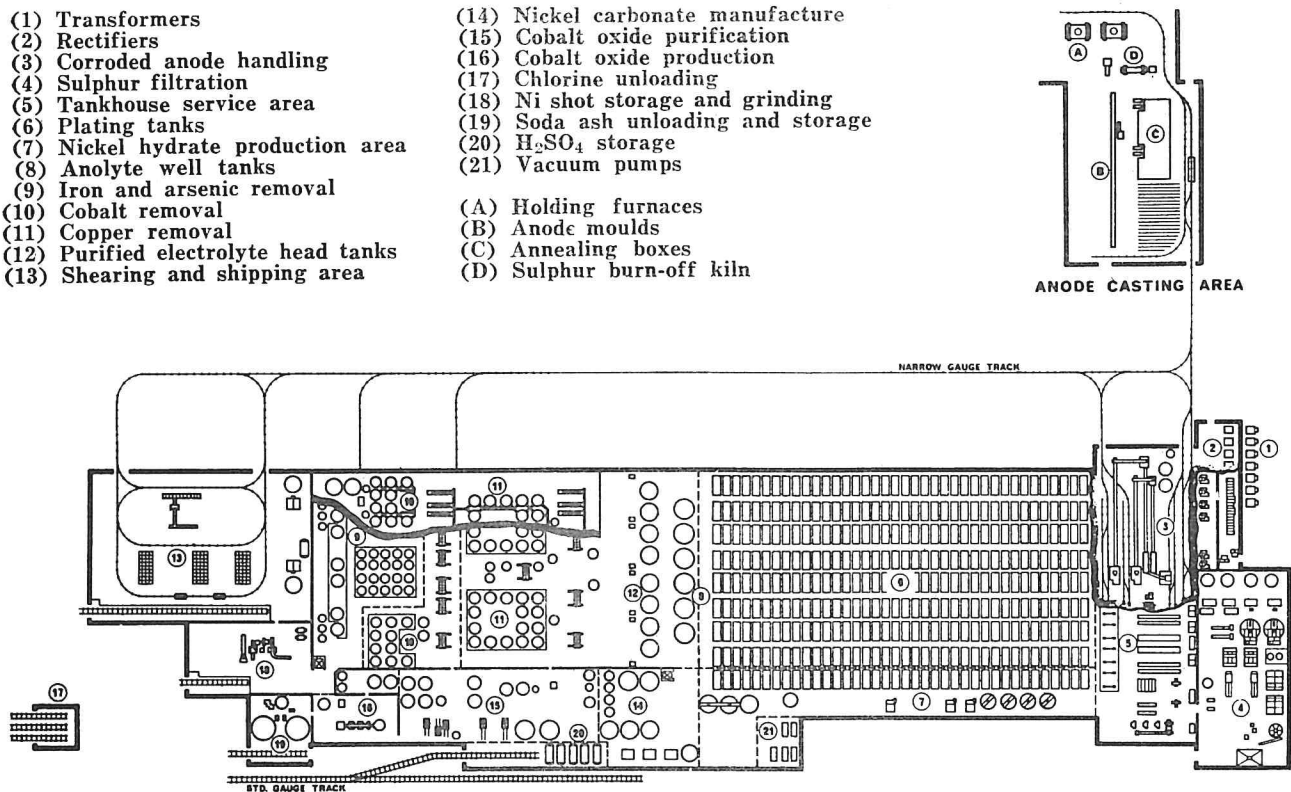


Fig. 10- Nickel Sulfide Electrolytic Plant, Thompson Refinery. (Spence and Cook 1964).

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