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NEW LEAD SMELTING PROCESSES<sup>1/</sup>

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## New lead melting processes

### Summary

1. In spite of the metal's relative nobility the metallurgy of lead is not so simple as one would presume. This is caused both by its occurrence as sulphide and by the low melting points and the high volatility of the metal and its compounds.

2. The old Scotch hearth process offered a simple way to take advantage of some of these circumstances and enabled the direct production of lead from high grade galena ore. Its adaptability to low grade ores and ordinary concentrate was, however, not so good, and the blast furnace process became the most important smelting procedure for lead and practically the only used around 1950.

3. The benefits of the hearth process were not forgotten, and when the supply of rich galena concentrates has become more abundant there has appeared some new processes grounded on the same basic principle, the roast reaction which can be expressed as the partial oxidation of galena to metallic lead:



4. Of the new processes the round hearth process is a direct conversion of the old straight hearth process whereas the self-fluxing smelting has some new features. These two are both characterized by excess oxygen in the basic reaction written above and therefore require a slag reduction treatment as a substantial complement to the smelting proper.

5. Another type of processes works with deficit oxygen in the basic reaction and must eliminate sulphur from the bullion. The most important process of this type is the Boliden process where the partial oxidation is performed as flash smelting in the upper part of an electric furnace; the residual sulphur is removed in a converter.

6. The blast furnace process for zinc (the ISF process) gives lead as a large-scale byproduct and has turned up as an important lead produce. Therefore, it must also be discussed as a new lead process.

7. To make an objection-free comparison of the newer lead processes both among themselves and with the standard blast furnace smelting is very difficult if not to say impossible. There are always some important parameters which are not comparable such as size, composition of galena concentrate, prices on local facilities (labour, fuel, energy), relations between these, local regulations on air and water pollution and possibilities to evaluate zinc and other minor metals in the raw material.

8. In spite of their simplicity and apparent ability to extract lead at low production costs the roast reaction processes have definite limitations in their adaptability. The most important is their exclusiveness to sulfidic concentrates and specially to high grade ones. For the round hearth process and the self-fluxing smelting, described above as the roast reaction processes with oxide excess, comes further the fact that they operate in small furnaces; it is not clear that a larger production can be performed in larger units so probably this requires enlarged number of units which increase the costs of both labour and equipment. Finally, these two processes seem to be suitable only for concentrates with basic gangue.

9. The Boliden process (and eventually also the Outokumpu method) should be competitive for medium and big smelters. It requires medium-grade concentrate, the capital costs are relatively high but the need of labour is small even if it must be skilled in order to handle the complicated equipment. If electric power is cheap and coke expensive, there is no doubt that this process, perhaps in combination with the basic ideas of the Outokumpu process, is very applicable for the treatment of good concentrates. A drawback is the handling of large quantities of flue dust which, however, can be mechanized to a great extent.

10. For large tonnages of concentrates of low and average grade the blast furnace process still is a good alternative. It has the greatest adaptiveness of all processes and is suitable also for oxidic minerals like cerrusite  $PbCO_3$  and anglesite  $PbSO_4$ . It requires both expensive fuel as coke and is not very low in costs for labour, repairs and construction but like the old blast furnace for pig iron it has proved very reliable, so even if existing lead plants should be rebuilt today most of them should still use the blast furnace method.

11. There is one important limitation and that is if the lead concentrate can be treated together with sufficient quantities of zinc concentrates. In this case the ISF process can produce lead to a cost far below that of every other lead producing process.

1 General survey on methods for lead production

11 General principles

11-1 In nature the most common lead mineral is galena which has the composition of lead sulphide; usually it is easy to concentrate to 50 - 60 % Pb and sometimes even up to 70 or 80 % Pb. From sulphide concentrates lead can in principle be produced in three different ways.

- a. by roasting and following reduction of the oxide - roast reduction methods (fig. 11 a)
- b. by roasting and simultaneous reduction of the oxide by unreacted sulphide - roast reaction methods (fig. 11 b)
- c. by melting with a metal such as iron having higher affinity to sulphur than lead - precipitation method (fig. 11 c)

11-2 In its simplest form the three methods can be illustrated by the flow-sheets in fig. 1 in which F means a treatment in some kind of furnace or other apparatus.

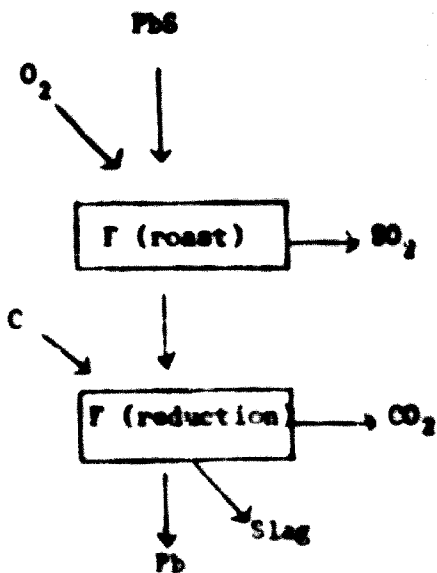


Fig. 11 a The roast reduction process

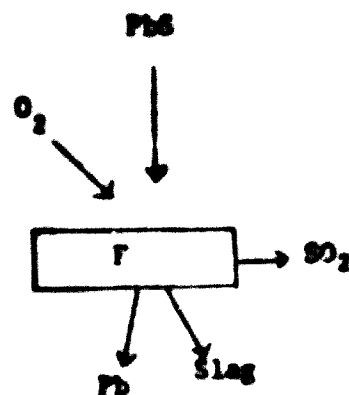


Fig. 11 b The roast reaction process



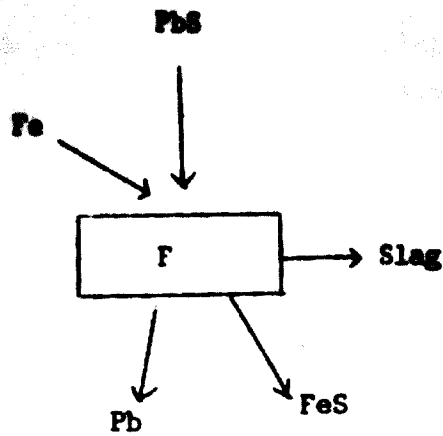


Fig. 11 c The precipitation process

Fig.-s 11 a, b, c Principal flowsheets for the basic processes for lead production.

11-3 If lead occurs in other minerals of oxidic character the material is treated according to method 1.

11-4 Of these methods the third nowadays is hardly utilized as an independent method and will therefore not be discussed. The roast reduction method is not the simplest but nevertheless the most applied and will be discussed first. The roast reaction methods have an apparent simple flowsheet but are applicable only under certain conditions; therefore, they have been subject for the special interest of metallurgists looking for new ideas for lead production from rich galena concentrates.

## 12 The system lead-sulphur-oxygen

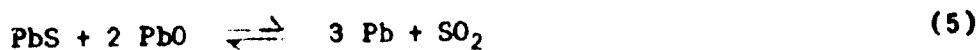
12-1 In order to form a background for the later discussions it is necessary to know a little of the thermochemical behaviour of lead and of some of its compounds such as oxide and sulphide.

Following reactions are relevant:

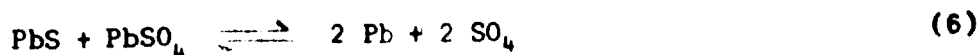


12-2

As combination of (3) and (4) basic sulphates are formed. A combination of (2) and (3) gives the roast reaction:



Another kind of roast reaction is a combination of (2) and (4):



12-3

As - at least at lower temperatures - the solubility of the solid or liquid substances Pb, PbO, PbS and PbSO<sub>4</sub> in each other is rather small, their activities will equal unity and we can without great error write the constants of these reactions as

$$\begin{aligned} k_1 &= p_{\text{O}_2} & k_4 &= 1/p_{\text{O}_2}^2 \\ k_2 &= p_{\text{SO}_2}/p_{\text{O}_2} & k_5 &= p_{\text{SO}_2} \\ k_3 &= p_{\text{SO}_2}/p_{\text{O}_2}^{3/2} & k_6 &= p_{\text{SO}_2}^2 \end{aligned}$$

12-4

As  $k$  is a function of temperature we have a system of the three variables  $k$ ,  $p_{\text{O}_2}$  and  $p_{\text{SO}_2}$ , and by giving  $p_{\text{SO}_2}$  certain values such as 0.1 at and 0.01 at we can make diagrams showing areas which indicate the conditions of existence for lead and the different compounds. These are given in figs 121 and 122.

12-5

From the area of PbS we must for extraction of lead move into the field of Pb, and the figures indicate how this possibly can be attained.

## 2 Roast reduction methods

### 21 The standard blast furnace method

21-1

In the classic roast reduction process, the flowsheet of which follows fig. 111, the galena concentrate is first roasted in a sintering procedure in order to form an agglomerated charge for the blast furnace. Into this the sinter is fed together with coke and fluxes and smelted under reducing conditions so that the lead and nobler metals are reduced to the metallic state whilst zinc and iron remain in the oxidic state and together with gangue and added fluxes form a slag.

$P_{SO_2} = 0.1$

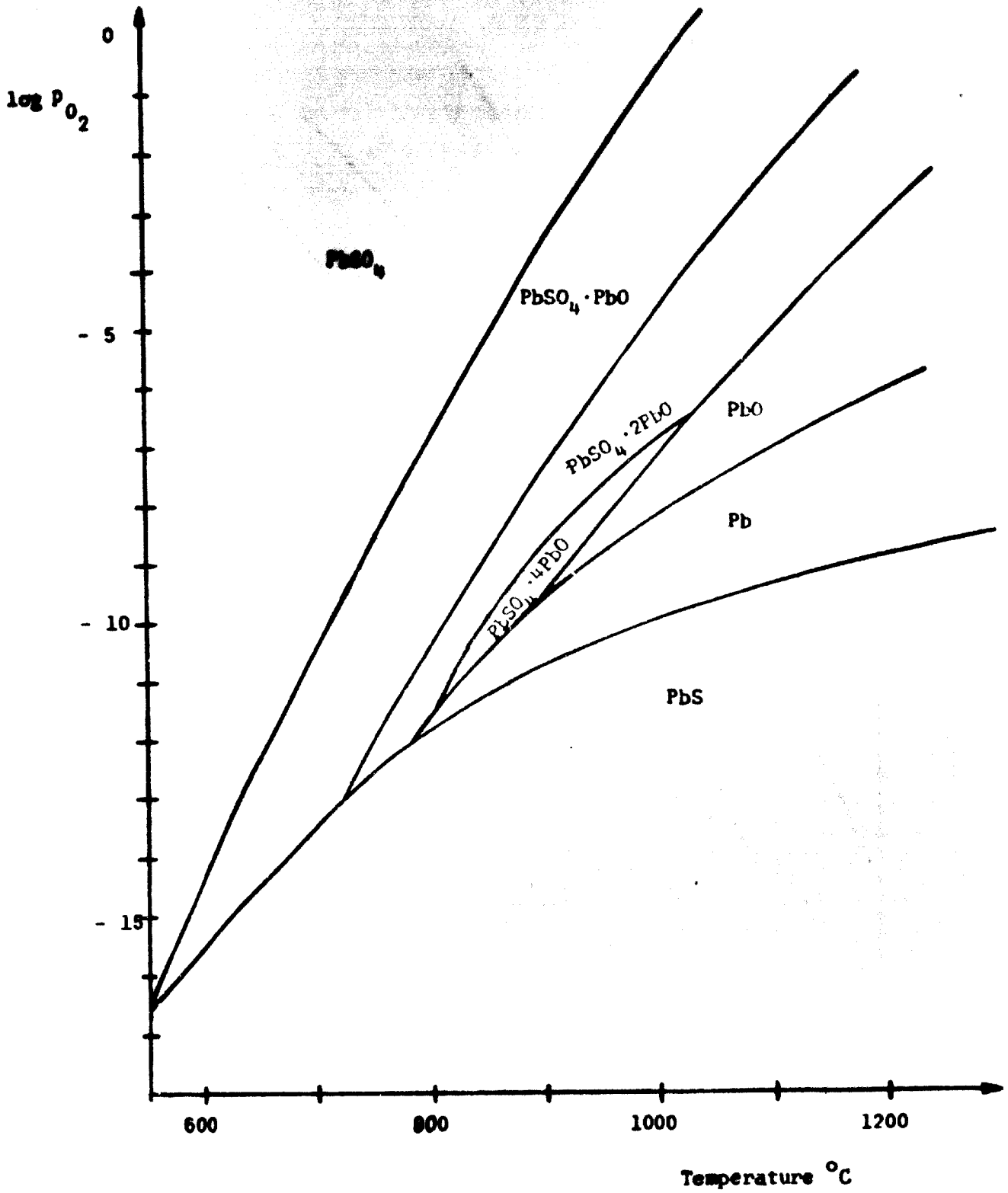


Fig. 121. The system Pb-S-O at  $P_{SO_2} = 0.1$  at

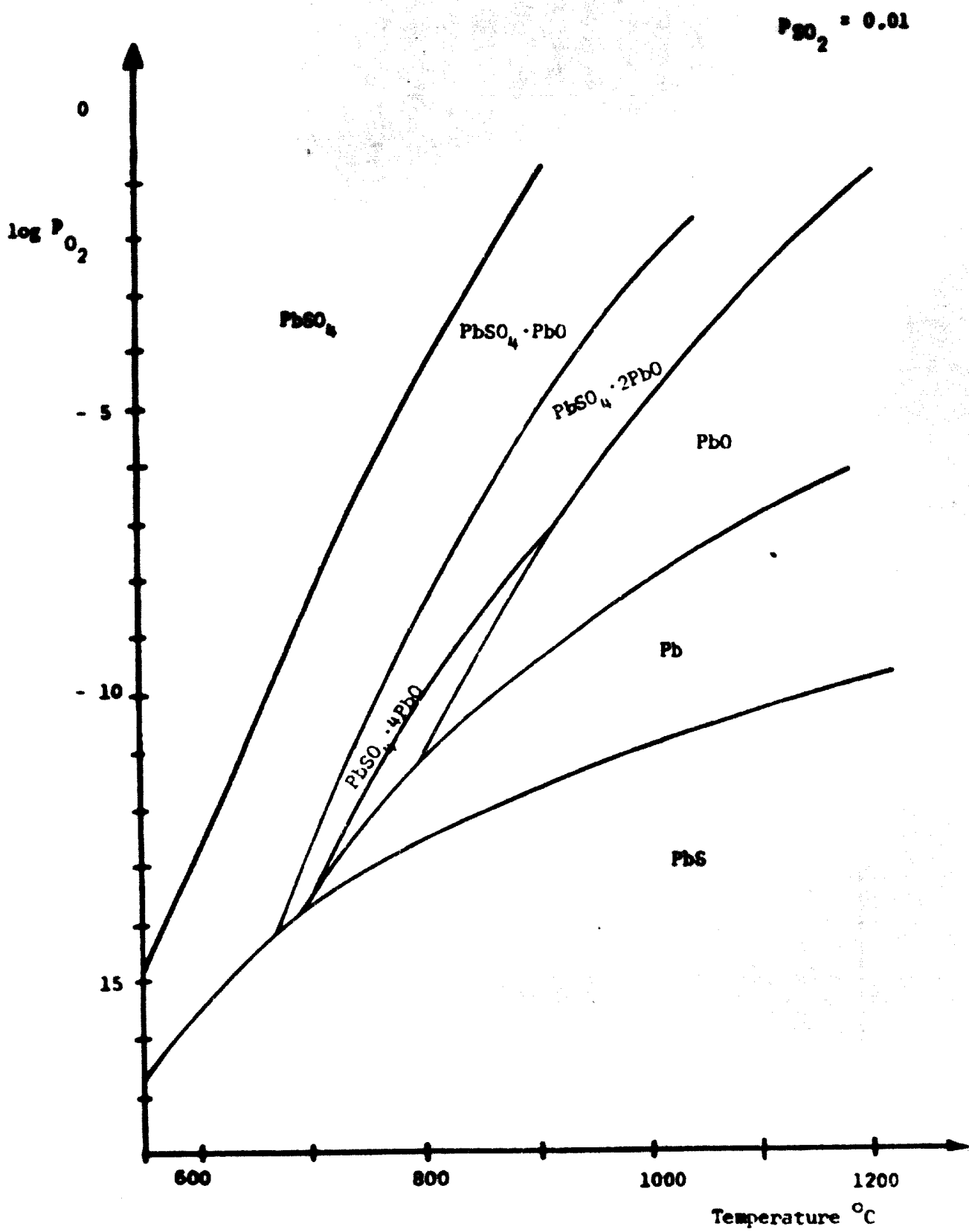


Fig. 122. The system Pb-S-O at  $P_{SO_2} = 0.01$  at

21-2

This process can be regarded as a universal process applicable for practically all kind of raw materials for lead. Therefore, it is the mostly used and without comparison the most important process to day; there is plenty of experience in it and the operations do not require too much skill for their performance.

21-3

The blast furnace process was developed for the relatively poor lead ores from which lead was extracted in the passed centuries. It works best with not more than about 40 - 50 % Pb in the charge; it is not so well suited for the usual concentrates with a lead content of 60 % and more.

21-4

The trouble with richer materials starts already with the sintering. As can be seen from fig. 121, galena can be desulphurized only above about 850°C, and at this temperature eutectics between PbS, PbO and basis sulphates exist which make the sintering charge sticky and pasty, so it must be diluted with other substances in order to maintain the porosity and permeability of the bed. Further, you cannot prevent the roast reactions (5) or (6).

to occur at elevated temperature in a rich mixture so there is a remarkable formation of molten lead which flows down to the grate and solidifies mainly in the wind boxes. Part of these troubles can be overcome by using updraft sintering in stead of downdraft, but the only way to eliminate them completely is to dilute the charge.

21-5

Also in the blast furnace it is advantageous to have a relatively low percentage of lead (about 50 %) in the charge. If very low there are considerable losses both in slag and in flue dust; lead is very volatile at temperatures in the melting zone and the top of the charge column must be kept cold (about 100°C) so that the lead can be recovered. A high grade charge is difficult to treat because of the low permeability at elevated temperature as discussed above; further tuyere temperature is higher because there is less slag to consume heat; this also causes an elevated formation of flue dust.

21-6

The general way to overcome the difficulties with the high grade raw materials is also here to dilute them, usually with slag, but it cannot be the right way to make the ore beneficiation more or less needless. There should be a better way to make use of the high grade achieved at the beneficiation and therefore the interest has been concentrated on roast reaction methods which - contrary

to the roast reduction method - only can make full use of high grade galena materials. Before that, however, we must discuss a zinc smelting process which acts as a new process for lead extraction.

## 22 The ISF process

### 22-1

The Imperial Smelting Furnace (ISF) process is properly a zinc production process giving lead as a by-product. It must be discussed here because a great number of ISF plants have been built recently thus taking over an essential part of the lead production which otherwise should have been performed mainly in standard blast furnaces.

### 22-2

Briefly, the ISF process (fig. 212) implies the melting of lead-containing sintered sphalerite in a blast furnace with preheated coke and preheated blast. The furnace burden is reduced, and under the predominant conditions zinc vapour can exist at elevated temperature and low partial pressure. The exit gases go to a condenser where they are rapidly cooled by droplets of circulating cool liquid lead on which the zinc vapour condenses and dissolves in the lead. The lead absorbs the heat, it is now cooled outside, the zinc is separated by liquation, and the lead is returned again to the condenser. In the blast furnace the charge lead is reduced and tapped with the molten slag, these two phases being separated in a forehearth.

### 22-3

Although originally designed as a zinc producing method the ISF-process has proved itself able to treat lead concentrates additionally with no substantial changes in the equipment, the quality and recovery being on the same level as in a conventional lead smelter. A ton of lead can be produced simultaneously with about 2 tons of zinc. This means, as will be shown later that within its capacity this process can produce lead from galena concentrate much cheaper than any other process.

## 3 The roast reaction processes

### 31 General introduction

#### 31-1

As mentioned in the opening part the roast reaction process should be a very simple one if not certain complications had intervened. First, the fundamental

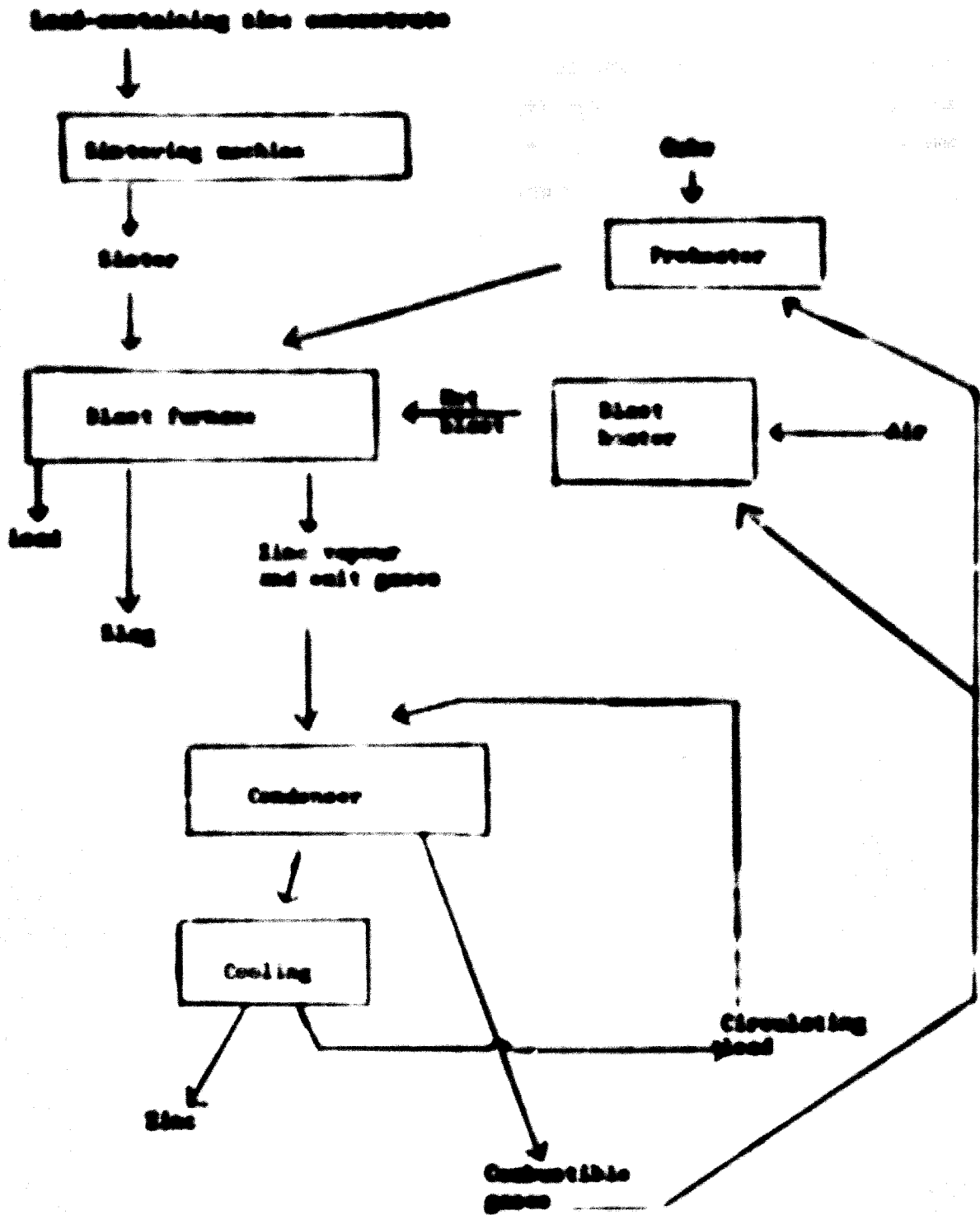


Fig. 21.2 Flowchart for the ZIF-process for lead and zinc

roast reactions (5) and (6) are never complete. However, by reacting with an excess of either  $PbS$  or  $PbO$  ( $PbCO_3$ ) the residual amount of  $PbS$  respectively  $PbO$  will become relatively small but must be specially treated. Therefore, the practical performance of the roast reaction is more complicated than the simple flowchart given in fig. 112. The original way, more or less taken over by some modern processes, is given in fig. 31. In a roasting procedure a. a part of the lead sulphide is converted into lead oxide or sulphate, and this material is then reacted in a furnace b. where the major part of the lead is recovered and a slag formed which has to be reduced in a final reduction furnace c.

### 31-2

All existing roast reaction processes are based on this scheme, and it has been possible to combine some of the steps entirely or partly. There are two species of processes, one working with sulphide excess in step b., so there is residual lead sulphide in the bullion and very little lead in the slag, and one with oxide excess in step b., i.e. the lead is free of sulphur but the slag contains so much lead oxide that it must be reduced in step c.

## 32. The sulphide excess processes

### 32-1

There are two technical processes working with excess sulphide in bullion from the roast reaction. One is the Boliden process which is applied in full scale, and the other the Outokumpu method which has been worked out in pilot plant scale.

### 32-2

Experimentally and not without success F.T. Fuller of St. Joseph Lead Co. has performed reaction (2) by blowing galena with air into a molten lead bath. In the Japanese Sagami process excess sulphur is said to be removed by soda ash; no details of this process are known.

## 321. The Boliden process

### 321-1

The flowchart of the Boliden roast reaction process is shown in fig. 321. Galena concentrate of high grade (65-75 %  $Pb$ ) is together with circulating flue dust and some limestone blown with slightly preheated air into the upper part of an electric furnace provided with electrodes submerging in a slag bath. The furnace can be considered as a flash smelting furnace with facilities for additional electric heating.

### 321-2

In the suspended material the reaction (2)  $PbS + O_2 \rightarrow Pb + SO_2$  takes place; as can be seen from fig. 321-22 lead can exist in contact with  $SO_2$  at elevated temperatures if the partial pressure of  $SO_2$  is not too high.



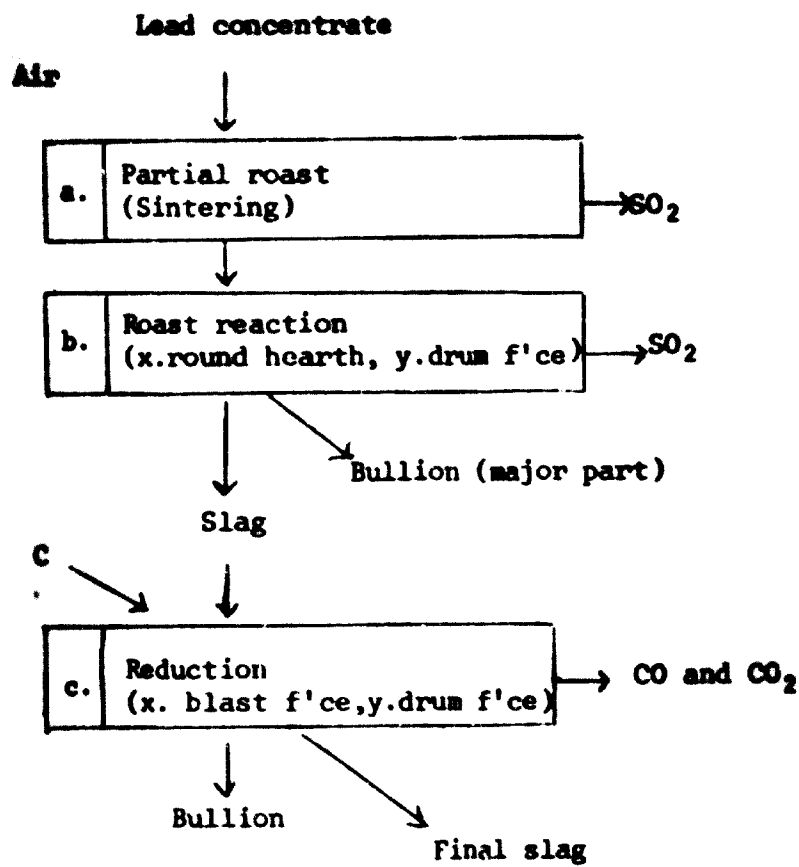


Fig. 31 Original version of the roast reaction process  
x. round hearth      y. self fluxing

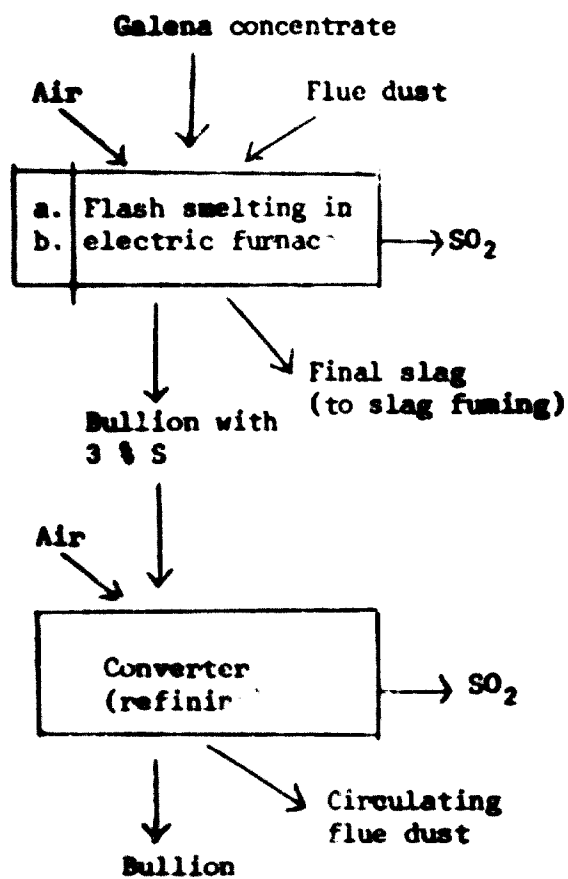


Fig. 321 The Boliden process

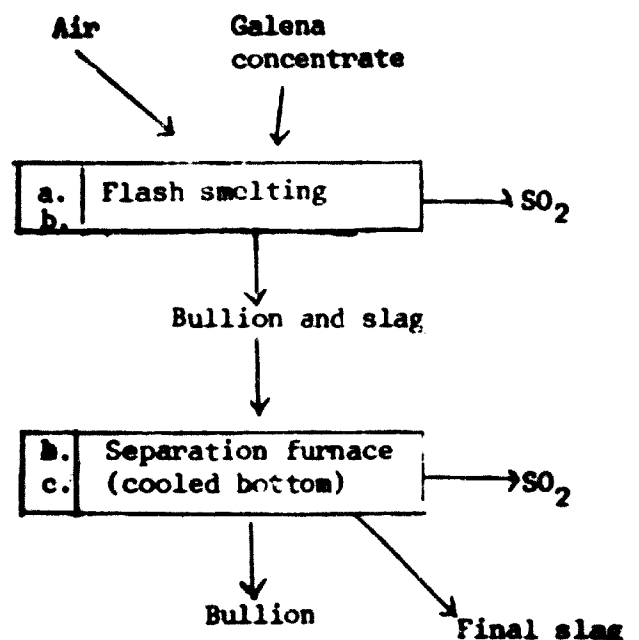


Fig. 322 The Outokumpu method

321-3

The reaction is, however, not complete, and it has been found more convenient to leave some unburnt sulphur in the reaction mixture by using deficit of air. As very little lead is oxidized, it is possible to produce a final slag with so little lead that it can be discarded or taken to a slag fuming plant. A small quantity of coke is added on top of the slag in order to depress the lead content of the slag. From the furnace a bullion with 3 % S is tapped; it is taken to a converter and the sulphur removed by blowing air through the melt a short time.

321-4

The temperature in the flame in the upper part of the furnace is rather high and as lead sulphide and lead are very volatile a substantial part of the input goes out of the furnace as vapours lead sulphide and lead and as dust. For concentrate containing 100 t of lead there is a formation of furnace dust with a lead content of 36.2 t and converter dust with 3.9 t lead or 40.1 t in total which quantities must be fed back into the furnace. The dust consists mostly of lead sulphate and reacts with lead sulphide in the flame.

321-5

The bullion is tapped at intervals in a launder and flows directly into a converter of Pierce-Smith type where it is blown with air. Most of the sulphur is removed as  $SO_2$  partly by roast reaction in the molten bullion, partly as volatilized PbS which is burnt above the surface; secondarily, lead sulphate is formed in the dust. After this treatment there is still a small amount sulphur left but when the blown bullion is cooled a coarse dross is separated which contains lead sulphide and some metallic impurities and which is returned to the electric furnace.

321-6

The slag contains about 34 %  $CaO$ , 24 %  $SiO_2$ , 17 % Zn and 4 % Pb; the quantity is small so in spite of the relatively high lead content only 1.1 % of the input goes this way. The zinc and lead contents are recovered when the slag is treated together with the copper matte slag in a slag fuming plant.

321-7

In its present version the Boliden process was developed in the early 1960 -es. A furnace of 8000 KVA with four electrodes smelts 235 t of concentrates in 24 hours and produces lead with a recovery of 98 %. It has successfully been tested with concentrates down to about 60 % Pb, but indications give an idea of its applicability also for concentrates of still lower grade.

### 322 The Outokumpu method

#### 322-1

Outokumpu Oy in Finland has developed a method where galena concentrate is burned in a flash smelting furnace with air in approximately stoichiometric quantity, the heat balance being improved by the use of preheated air instead of electric heat as in the Boliden process. As in the latter there is a formation of sulphur-containing bullion in which the lead sulphide cannot react with the lead oxide in the slag according to the low activity of lead sulphide in the bullion. Bullion and slag are taken into a separation furnace.

#### 322-2

The bottom of the gas-fired separation furnace is cooled so there is a pronounced vertical temperature gradient in the metallic phase. The cold metal in the bottom cannot dissolve sulphur, and this metal is continuously tapped out. The liquated sulphide flows up and causes a concentration of lead sulphide in the top layer, and when the concentration has increased to the level where the roast reaction can take place,  $SO_2$  escapes and there is no accumulation of sulphide. This effect is further increased by the presence of copper which facilitates the realization of the roast reaction.

#### 322-3

In principle, this method is very near the idealized flow-sheet given in fig. 112. The method has been tested with good results in a pilot plant with a capacity of 2 tons of concentrates per hour, but lack of material has prevented the construction of a full scale plant.

### 33 The oxide excess processes

#### 31-1

In older times, when the supply of coarse high grade galena ore was better than it is today, the ore was treated according to fig. 31 in small reverberatory furnaces where the ore first was partly oxidized at moderate temperature (step a.), after that was the temperature raised so that the roast reaction occurred (step b.) and the lead could flow out, and finally the remainder reduced with coal or charcoal (step c.) in order to recover as much lead as possible. This three-step procedure was modernized to the two step Scotch hearth process (see 331) and this further developed into the round hearth process (see 332). The self-fluxing process (333) is another modification.

### 131 The Scotch hearth process

131-1

Instead of separate roasting and reheat reactions these two steps are performed simultaneously in the ore hearth or Scotch hearth (Buckan hearth) process. The hearth was a shallow tray containing lead. On the surface was a heap of preferably lumpy ore with at least 60 % Pb plus some coal, into the heap air was blown from one side, from the other side the mixture was intensely stirred by a poker. The coal was combusted in order to maintain the temperature at 900-1000°C (the reaction is weakly endothermic), the air oxidized part of the sulphide to oxide or sulphate and the poking brought remaining sulphide in close contact with oxide or sulphate on the reaction surface. At 900-1000°C the solubility of oxide/sulphate and sulphide in each other and in metallic lead are rather low, and as the reactants must meet in a common phase stirring is required to bring them in the same phase. An increase in temperature should give higher solubilities and a better rate of reaction, but already in this area of temperature the volatilization of lead and its compounds, especially the sulphide, are very high, so a considerable amount of lead went into the fine dust and must be returned to the furnaces. Much manual labour even if the poking can be carried out mechanically and intense formation of dust (15-25 % of the input) made the work costly and unpleasant.

131-2

Eventually the percentage of lead in the mixture decreased and it became more difficult to attain the activities of lead oxide and sulphate required for the reaction with the sulphide. The reaction stopped when the lead content has dropped to 30-50 % and the residue, called gray slag, must be removed and the lead recovered in a blast furnace. The finer the raw material to the hearth, the higher the percentage of lead and sulphur in the slag. When the supply of high grade coarse galena ore has decreased, the old hearth process has practically disappeared.

### 132 The round hearth process

132-1

The hearth process exists, however, still in a somewhat changed modification the finest of which is given in fig. 132. The old Scotch hearths were stationary and straight and the poking mechanism had to be a valve. The new device consists of an annular hearth, resting on rolls and rotated by a rim

of cogs driven by a toothed wheel. The air blows in from below through a central pipe connected to the circular window which is a part of the water-cooled inner wall of the hearth. Punching of the tuyeres is made automatically.

202-2

Poking and removal of slag is performed mechanically at one station and when a certain sector passes this station the blast is automatically switched off. The lead formed is tapped from the surface level in a collecting ladle. Such factors as blast, rotation speed, poking intensity and addition of ore and coal can be varied according to the nature of the raw materials. The hygienic conditions for the workers are very satisfactory.

202-3

A furnace with hearth dimensions outer diameter 12 feet and inner diameter 7 feet can with 42 tuyeres in 24 hours melt 81 t ore with 75 % Pb (61 t lead) and produce 41.7 t dross (67.5 % recovery). 21 t gray slag with 50 % Pb is taken out (17 % of input) and 13.6 t flue dust with 70 % Pb (15.5 % of input) is formed and returned. 5 - 5.5 t coal is burned. If the raw material consists of fine concentrates, a slight sintering together with the flue dust is very advantageous and decreases the formation of dust on the hearth. If the quantity of slag is small it can best be reduced in a short drum furnace.

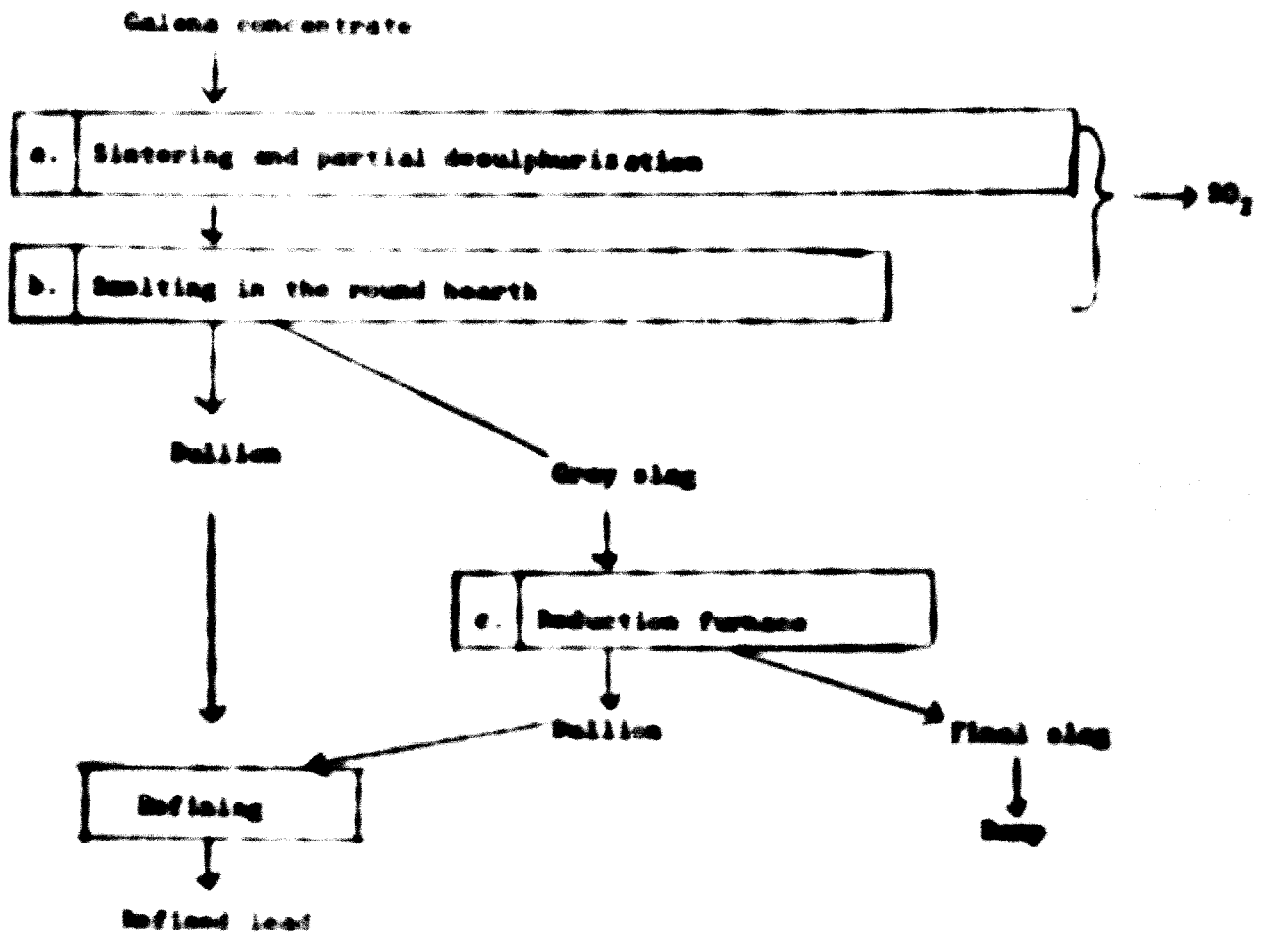


Fig. 202. Flowchart for the round hearth process.

332-4

The round hearth process is used only by the Bleiburger Bergwerks Union (BBU) in Austria. The present smelting capacity is about 80 t concentrates with 75 % Pb in 24 hours, but it is said to work satisfactorily down to about 20 t a day. All operations can be mechanized, the requirements for equipment and buildings are small, so if the material is suitable this method should be very convenient for a small plant.

### 333 The self-fluxing smelting

333-1

The self-fluxing process can be considered as another development of the ancient reverberatory smelting. It is still a three-step process, but the roasting step a. is performed on a sintering machine and the roast reaction b. and the reduction of residue c. are both performed in the same oil-fired short drum-furnace.

333-2

As mentioned above there is a considerable roast reaction occurring already at the sintering of a rich galena concentrate. If the sintering machine works with updraft the lead droplets solidify when they meet the cold air coming from below, and the metallic lead in the sinter does no harm to the sintering process. When the gangue is basic the activity of lead oxide in the sinter and in the slag is increased and the recoveries are rather good.

333-3

The concentrate with about 70 % Pb is first sintered on a straight Dwight-Lloyd machine with updraft sintering and a sinter with about 75 % Pb is produced of which one third is metallic lead; the  $\text{SiO}_2$  - content is 2.3 % and  $\text{CaO} + \text{MgO}$  1.5 %.

333-4

The sinter is taken to short drum furnaces with an inner diameter of 2.37 m and a length of 2.63 m. The furnaces are oil-fired and the charge heated to about  $1160^\circ\text{C}$  at which the roast reaction is completed, the lead separated from the only partially molten slag and tapped off. In order to melt completely and reduce anthracite and soda ash are added and the heating continued; now the furnace acts as reducing furnace. The total lead recovery is about 94 %.

333-5

The self-fluxing smelting is applied by Rudnik svinca in Topilnica in Mežica, Slovenia, Yugoslavia, where a galena concentrate with about 70 % Pb is smelted in quantities of about 100 t in 24 hours. There is one updraft sintering machine:

and two operating 2.5 m<sup>3</sup> short drum furnaces, each furnace having a capacity of producing 39 t of bullion in 24 hours. The method seems very suitable for a small or medium-sized smelter especially when coke is expensive.

#### 4 Comparison of the different methods

4-1

Of the methods mentioned the old Scotch hearth method will not be taken into consideration, and the Outokumpu method although very attractive has never been tested in full industrial scale so there is not sufficient basis of experience for its comparison with other methods.

4-2

To make a comparison free of objections is very difficult if not to say impossible. There are always some important parameters which are not comparable such as size, composition of galena concentrate, prices on local facilities (labour, fuel, energy), relations between these, local regulations on air and water pollution and possibilities to evaluate zinc and other minor metals in the raw material.

4-3

Dr. W. Schwarz of Lurgi, Frankfurt, has made comparisons of some different processes for lead production, and on these as basis I have tried to draw up comparable figures in a table. For each process I have attempted to analyze how the production costs divide in material, fuel, electric energy, labour and maintenance, and finally how much capital costs can be estimated to.

4-4

In this comparison the ISF process is the most difficult to analyze. Here lead is produced as byproduct with practically no extra construction costs, and the extra consumption of fuel for the lead is negligible. Addition of a ton of lead concentrate decreases the capacity for treating zinc concentrate about 94 kg, but both lead in zinc concentrate and zinc in lead concentrate are recovered. In a small lead plant the production of sulphuric acid is often questionable, but in an ISF-plant all sulphur is evaluated. For larger quantities of lead concentrates the ISF alternative is hardly to be considered.

4-5

Table 41 gives consumption figures for the different processes. Such expenses as costs for overhead, for licences, for assaying, for other staff than direct labour besides repair and maintenance etc. have not been taken into consideration as they differ much from one site to another. The figures in column S refer to

a small plant with a throughput of about 10 000 t/yr (30 t a day) and L to a larger plant of about 50000 t/yr (150 t a day).

4-6

In table 42 each main item in tab. 41 has got a characteristic relative number 0 to 5 where 0 means a not significant cost and 5 a very dominating cost. The sum of these numbers gives a relative estimation of the competitiveness of a process and how it varies with the size of the production. Finally, fig. 41 tries to illustrate these variations graphically.

## 5 Conclusions and summary

5-1

In spite of their simplicity and apparent ability to extract lead at low production costs the roast reaction processes have definite limitations in their adaptability. The most important is their exclusiveness to sulfidic concentrates and specially to high grade ones. For the round hearth process and the self-fluxing smelting, described above as the roast reaction processes with oxide excess, comes further the fact that they operate in small furnaces; it is not clear that a larger production can be performed in larger units so probably this requires enlarged number of units which increases the costs of both labour and equipment. Finally, these two processes seem to be suitable only for concentrates with basic gangue.

5-2

The Boliden process (and eventually also the Outokumpu method) should be competitive for medium and big smelters. It requires medium-grade concentrate, the capital costs are relatively high but the need of labour is small even if it must be skilled in order to handle the complicated equipment. If electric power is cheap and coke expensive, there is no doubt that this process, perhaps in combination with the basic ideas of the Outokumpu process, is very applicable for the treatment of good concentrates. A drawback is the handling of large quantities of flue dust which, however, can be mechanized to a great extent.

5-3

For large tonnages of concentrates of low and average grade the blast furnace process still is a good alternative. It has the greatest adaptiveness of all processes and is suitable also for oxidic minerals like cerussite  $PbCO_3$  and anglesite  $PbSO_4$ . It requires both expensive fuel as coke and is not very low in costs for labour, repairs and construction but like the old blast furnace for pig iron it has proved very reliable, so even if existing lead plants should



be rebuilt today most of them should still use the blast furnace method.

5-4

There is one important limitation and that is if the lead concentrate can be treated together with sufficient quantities of zinc concentrates. In this case the ISF process can produce lead to a cost for below that of every other lead producing process.

Table 41. Basis of cost calculations for smelting of galena concentrate with 72 % Pb.

S small plant, L large plant; ISF considered not fit for use for large tonnages.

Per t concentrate	Process	Blast furnace		ISF	Round hearth		Self-fluxing		Boliden	
		S	L	S	S	L	S	L	S	L
Direct labour (man-hours)		9.7	3.2	1.5	11.6	3.4	7.1	3.3	4.0	1.5
Fuel in kg										
Oil		13.3	6.4	-	85	51	83.3	74.0	-	-
Coke		168	150	-	13	13	-	-	12.5	12.5
Coal		-	-	-	150	150	24	24	-	-
El.energi in KWh		154	135	58	70	60	161	43	750	596
Materials										
Limestone		44	44	113	80	90	-	-	200	200
Soda ash		32	32	12	15	15	25	25	-	-
Maintenance (man-hours)		5.5	2.2	1.1	3.4	2.0	6.0	4.0	3.0	1.0
Capital costs \$		16.1	8	4.5	12.1	8.9	16.9	14.0	15	7.5

Table 42. Relative characteristic cost of different processes.

S small plant, L large plant.

Process	Blast furnace		ISF	Round hearth		Self-fluxing		Boliden	
	S	L	S	S	L	S	L	S	L
Labour	5	2	1	5	3	4	3	3	1
Fuel	5	5	-	4	4	3	3	-	-
El. energy	1	1	-	1	1	1	1	5	5
Materials	1	1	1	1	1	1	1	2	2
Maintenance	5	2	1	3	2	5	4	3	1
Capital costs	5	3	1	4	4	5	5	5	3
<b>Sum</b>	<b>22</b>	<b>14</b>	<b>4</b>	<b>18</b>	<b>15</b>	<b>19</b>	<b>17</b>	<b>18</b>	<b>12</b>

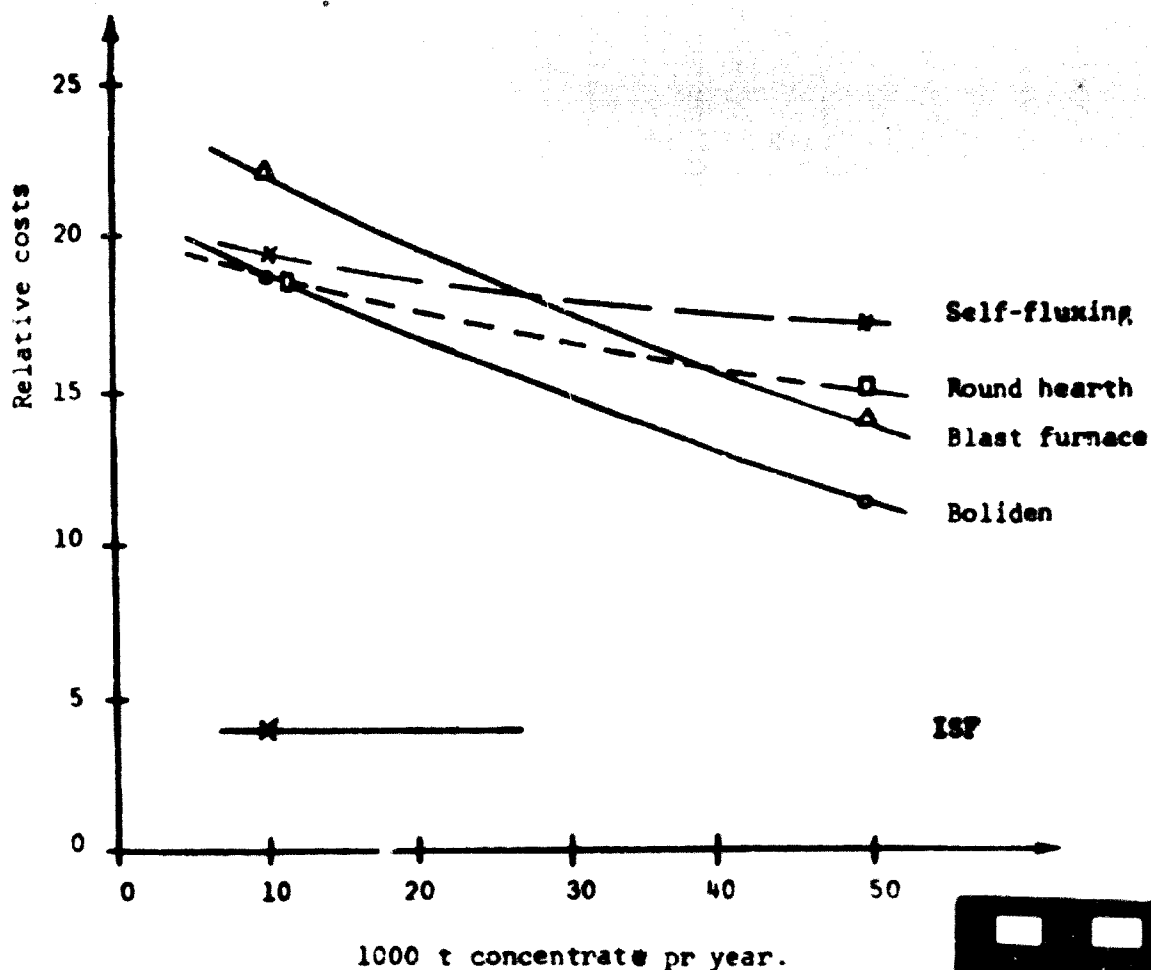


Fig. 42. Relative costs of different lead processes.



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