United States Patent

Themelis et al.

[15] **3,663,207**

[45] **May 16, 1972**

[54] DIRECT PROCESS FOR SMELTING OF LEAD SULPHIDE CONCENTRATES TO LEAD

[72] Inventors: **Nickolas J. Themelis,** Beaconsfield, Quebec; **John R. Rawling,** Lively, Ontairo, both of Canada

[73] Assignee: **Noranda Mines Limited**

[22] Filed: **Apr. 21, 1970**

[21] Appl. No.: **30,557**

[30] **Foreign Application Priority Data**

Oct. 27, 1969 Canada 065897

[52] **U.S. Cl.** **75/77,** 75/78, 75/23

[51] **Int. Cl.** **C22b 13/02**

[58] **Field of Search** 75/78, 77, 23

[56] **References Cited**

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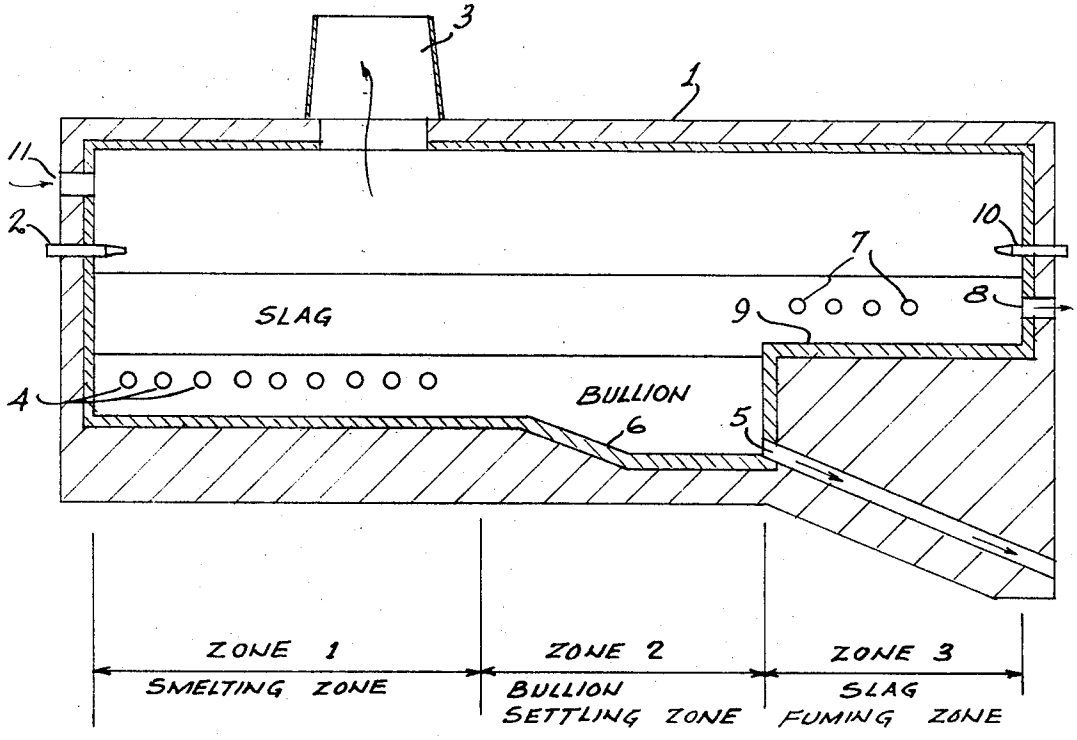
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Primary Examiner—L. Dewayne Rutledge
Assistant Examiner—G. T. Ozaki
Attorney—Mason, Fenwick & Lawrence

[57] **ABSTRACT**

A process for continuously smelting and converting lead sulphide concentrates to molten lead or bullion in a furnace comprises introducing pelletized lead sulphide concentrate onto the surface of a molten bath in the furnace and smelting the same therein, introducing oxidizing gas into the furnace to convert the lead sulphide concentrates to molten lead, tapping off molten metallic lead, treating the slag in a slag fuming zone to reduce zinc oxide in the slag to zinc vapor, introducing an oxidizing gas above the slag in the slag fuming zone to oxidize the zinc vapor to zinc oxide fume, passing the fume to a fume collecting area and thereafter withdrawing substantially zinc free slag.

10 Claims, 1 Drawing Figure



INVENTORS
NICKOLAS J. THEMELIS &
JOHN R. RAWLING
BY
Mason, Fenwick & Lawrence
ATTORNEYS

DIRECT PROCESS FOR SMELTING OF LEAD SULPHIDE CONCENTRATES TO LEAD

This invention relates to a process for the continuous smelting of lead concentrates to lead bullion in a single furnace.

Present methods for the recovery of lead from lead sulfide concentrates commonly involve mixing the concentrates with flux, return lead sinter and various residues such as dust which mixture is then roasted on a sintering machine to eliminate most of the sulfur and produce a strong porous sinter which is a suitable feed for a lead blast furnace. Hot sinter is then transferred to a lead blast furnace in which reduction of the lead oxide in the sinter to metallic lead takes place by means of carbon and carbon monoxide provided by coke charged with the sinter. Combustion of the coke also furnishes most of the heat required for smelting. Lead bullion is tapped from the furnace, drossed and subsequently refined. A lead slag is also withdrawn from the furnace and may be treated in a fuming furnace to recover its lead and zinc content.

The reactions occurring in the sintering process principally involve the oxidation of metallic sulfides such as PbS,ZnS, and FeS to form oxides, and are strongly exothermic. As noted above, the concentrates are commonly mixed with cold return sinter which serves to limit the temperature rise during the sintering process by its diluting effect and also to achieve the desired sinter quality. In contrast, in the blast furnace a large part of the heat required for smelting and reducing the oxide sinter is provided by the combustion of fuel (coke). The two-step sintering and blast furnace smelting process for the recovery of lead from lead concentrates is overall thermally inefficient. The process also requires the immediate handling of sinter between the sintering machine and blast furnace and in recirculation to the sinter machine. Furthermore, the provision of separate sintering machines and blast furnaces involves the expenditure of substantial capital and operating costs and the provision of a single unit to recover metallic lead from lead sulfide concentrates would make possible considerable savings in cost.

The present invention provides a process which eliminates the sintering operation and in which the smelting operation is conducted within a single unit in such a way that a large part of the heat required for the smelting of the lead concentrates is provided by the exothermic heat of the oxidation reactions involved.

According to the present invention there is provided a process for the continuous smelting and converting of lead sulfide concentrates to molten lead or bullion in a furnace containing a molten bath and having a charging end and a slag tapping end, said process involving gradual reactions along the length of the furnace and including the steps of:

a. feeding pelletized lead sulfide concentrates also containing minor amounts of zinc sulfide through the charging end of the furnace and onto the surface of the molten bath in a smelting zone of said furnace;

b. smelting said concentrates in the smelting zone whereby heat contained in the molten bath is transferred to the pelletized concentrates;

c. controlling the flow of the molten bath so that it flows along the length of the furnace from the smelting zone towards a bullion tapping means in the furnace;

d. introducing an oxidizing gas into the molten bath to convert the lead sulfide concentrates to molten lead with concurrent formation of sulfur dioxide whereby the smelting and converting reactions are effected essentially in the smelting zone and the exothermic heat of the converting reaction is utilized for smelting;

e. allowing the molten metallic lead to settle in a bullion settling zone remote from the charging end;

f. drawing off bullion from the bullion settling zone through a bullion tapping means intermediate the charging end and the slag tapping end of the furnace;

g. treating the slag flowing past said bullion tapping means in a slag fuming zone with a reductant whereby to reduce zinc oxide in the slag to zinc vapor;

h. introducing an oxidizing gas above said slag in the slag fuming zone whereby to oxidize said zinc vapor to zinc oxide fume and passing said fume to a fume collecting area; and

i. withdrawing substantially zinc-free slag from said slag tapping end.

A major advantage of the process provided by the present invention is the considerable saving in fuel costs and the elimination of the need for coke achieved by the direct smelting of lead concentrates to metallic lead in one furnace. Other significant advantages of the process are that a saving of capital cost is achieved by replacing the sintering machines and a lead blast furnace with one unit with a consequent reduction in other operating costs such as labor and maintenance costs. Another advantage of the process is that the continuous nature of the process lends itself to a high degree of instrumentation and automation. Yet again the process offers the advantage of continuous production of sulfur dioxide in the flue gas exiting the furnace at a relatively high concentration thus making possible the economic recovery of sulfur dioxide which may be utilized in, for example, the production of sulfuric acid.

The converting reaction of the lead sulfide concentrates to metallic lead is an exothermic reaction and, as stated above, this exothermic heat of converting reaction is efficiently utilized in accordance with the process of the present invention in the smelting operation by which the pelletized lead sulfide concentrates which are introduced into the furnace are melted. Presently available processes are not as thermally efficient as the process of the present invention. Furthermore, the process of the present invention makes possible the production of an essentially clean or metal-free slag which can be discarded.

In order that a clear understanding of the process of the invention may be gained the process will be described by reference to the accompanying drawing which is a diagrammatic vertical cross section taken longitudinally through a furnace suitable for carrying out the process of the invention.

As can be seen from the FIGURE the length of the furnace, indicated generally at **1**, is divided into three separate zones, namely zones **1**, **2** and **3**. These zones are respectively a smelting zone (Zone **1**), a bullion settling zone (Zone **2**) and a slag fuming zone (Zone **3**).

The reaction vessel **1** comprises a refractory lining (not shown) and a charging port **11** is provided at the entrance to Zone **1** for the introduction of pelletized lead sulfide concentrates. An exhaust stack **3** is provided for the removal of gases and fume from the vessel **1** in the upper surface thereof. Tuyeres **4** are provided in Zone **1** as indicated for the introduction of air or other oxidizing gas into the bath in Zone **1**. A lead bullion tapping port **5** is provided at the end of Zone **2** for the withdrawal of lead bullion settling from the slag in Zone **2** and the floor or hearth of furnace **1** is provided with a depression **6** in Zone **2** which aids in the separation of the bullion from the slag. Tuyeres **7** are provided in the slag fuming zone for the introduction of a reducing gas or pulverized coal into the slag contained within Zone **3** and a slag tapping port **8** is provided for the removal of slag from the furnace **1**. Separation of the bullion from the slag is aided also by the provision of an elevated portion **9** in the slag fuming zone, said portion **9** being raised above the level of the floor or hearth of furnace **1** in the smelting zone thus providing a barrier to the flow of bullion through the slag tapping port **8**. An air inlet **10** is provided in the slag tapping end of the furnace **1** for the injection of air above the slag in order to burn gaseous reduction products resulting from the introduction of a gaseous or pulverized solid reductant through tuyeres **7** into the slag. These gaseous reduction products comprise zinc vapor, lead vapor and carbon monoxide and these are oxidized to form zinc oxide and lead oxide fumes and carbon dioxide and to generate additional heat for utilization in the process. A supplementary burner **2** may be included in the charging end of furnace **1** to provide an additional source of heat for the smelting process.

Pelletized lead sulfide concentrates together with a silica or lime flux are charged continuously into the furnace 1 through charging port 11. The furnace 1 contains a molten bath of lead bullion with a supernatant layer of slag as illustrated in the drawing. The concentrates and flux are fed into the furnace 1 in such a manner, as described in more detail hereinafter, that they are distributed uniformly on the bath surface throughout Zone 1 where they are enveloped by the slag and smelted at a very high rate through the efficient transfer of heat generated in the molten bath. Air is introduced through tuyeres 4 in order to effect conversion of the lead sulfide concentrate and of iron sulfide in the concentrates in accordance with equations (1), (2), (3) and (4) below:

$$PbS + O_2 \rightarrow Pb + SO_2 \quad (1)$$

$$FeS_2 \rightarrow FeS + \tfrac{1}{2}S_2 \quad (2)$$

$$FeS + 3/2\ O_2 \rightarrow FeO + SO_2 \quad (3)$$

$$\tfrac{1}{2}\ S + O_2 \rightarrow SO_2 \quad (4)$$

Reactions (1), (3) and (4) are highly exothermic. Any zinc sulfide contained in the lead concentrates is oxidized in accordance with equation (5) below:

$$ZnS + 3/2\ O_2 \rightarrow ZnO + SO_2 \quad (5)$$

Although not desirable, some lead sulfide may be oxidized in accordance with equation (6) below:

$$PbS + 3/2\ O_2 \rightarrow PbO + SO_2 \quad (6$$

Reactions (5) and (6) are also exothermic. As the molten material flows past tuyeres 4 it is gradually and continuously depleted of ferrous sulfide, zinc sulfide and lead sulfide. In order to ensure complete conversion of the lead sulfide to metallic lead the charging of concentrate and flux is restricted to the smelting zone (Zone 1) and separation of the bullion from the slag is completed in bullion settling zone (Zone 2).

Slag is formed in the smelting zone by the reaction of ferric oxide, zinc oxide and lead oxide with silica flux in accordance with equation (7) below:

$$2MeO + SiO_2 \rightarrow 2MeO{\cdot}SiO_2 \quad (7)$$

If lime flux is added the CaO will also combine with silica according to the above equation.

The slag and bullion flow from smelting zone 1 into the bullion settling zone as the result of displacement brought about by the continuous charging of concentrates and flux in zone 1 and the withdrawal of bullion from zone 2 and slag from zone 3. Zone 2 is relatively quiescent and the bullion settles out of the slag and a more complete separation of the slag and bullion is achieved than is possible in Zone 1 where turbulent conditions are generated by the introduction of air or other oxidizing gas through tuyeres 4. This introduction of air causes some mixing of slag and bullion at their mutual interface. Bullion separating out from the slag in Zone 2 is tapped from the furnace 1 through bullion tapping port 5 situated at the bottom of depression 6 in the floor or hearth of furnace 1. The tapping of bullion may be carried out on either a continuous or an intermittent basis.

The slag separated from the bullion in Zone 2 contains relatively high concentrations of both lead and zinc. Cleaning of the slag is effected in the slag fuming zone by the introduction of a suitable reductant, such as natural gas or pulverized coal through the tuyeres 7. Reactions which take place by the introduction of the reductant into the slag will depend upon the nature of the reductant, but when methane is used as the reductant reactions proceed in accordance with equations (8) and (9) below:

$$\tfrac{1}{3}CH_4 + ZnO \rightarrow Zn + \tfrac{1}{3}CO + \tfrac{2}{3}H_2O \quad (8)$$

$$\tfrac{1}{3}CH_4\ PbO \rightarrow Pb + \tfrac{1}{3}CO + \tfrac{2}{3}H_2O \quad (9)$$

The zinc vapor, lead vapor and carbon monoxide produced in accordance with equations (8) and (9) are burned by injecting air through aperture 10 in Zone 3 and produces zinc oxide and lead oxide fume and carbon dioxide and also generates additional heat which may be utilized in the process. The zinc oxide and lead oxide fume and carbon dioxide pass through stack 3 in the roof of furnace 1 together with other gases from zones 1 and 2 to a fume collection system where zinc oxide and lead oxide fume may be removed in accordance with procedures well known in the art. The treated being slag passing the tuyeres 7 under the influence of gravity is allowed to settle and is tapped from furnace 1 through tapping port 8 either continuously or intermittently as desired.

The continuous charging of lead sulfide concentrate and flux into the furnace 1 and the withdrawal of lead bullion and slag from the furnace through their respective tapholes 5 and 8 gives rise to a flow of material through the furnace 1. As illustrated in the drawing the flow of slag and bullion through the furnace 1 is generally co-current and is brought about through the influence of gravity. No special steps need be taken therefore to ensure the co-current flow of bullion and slag and the sequential series of reactions embodied in the reactions (1) to (9) is readily achieved in accordance with the invention by suitably dimensioning the furnace 1 to provide effectively for separate smelting, bullion settling and slag fuming zones.

In order that full utilization of the heat of conversion may be made in the smelting operation, the concentrates preferably are introduced into the furnace 1 throughout the smelting zone. Heat transfer to the concentrate charge is enhanced and high smelting rates are achieved by feeding the concentrates to the vessel in the form of pellets.

Lead sulfide concentrates produced, for example, by flotation, can be pelletized easily without any additive, although additives such as bentonite can be used and, after pelletizing, the pellets may be partially dried prior to being fed into the continuous smelting reactor. Whether or not the concentrates are dried prior to being fed to the furnace will depend on the relative costs of drying them before or after introduction into the furnace. It has been found that wet pellets from a pelletizing machine introduced directly into the continuous furnace tend to smelt individually without agglomerating into large masses of unsmelted material which could entrap gases or moisture thus causing minor explosions which are sometimes experienced when wet unpelletized concentrates are charged onto a molten bath.

Fluxes such as siliceous ore or limestone may be required for the production of slag and may be charged into the furnace. If economically feasible, it is preferable to grind flux to below 10 mesh (ASTM Standard Sieve) and incorporate the flux into the pellets of concentrate thus producing a type of self-fluxing pellets. This procedure simplifies control of the slag composition and eliminates the possibility of segregation of the flux in the continuous smelting furnace. If, however, incorporation of the entire amount of flux in the pellets is not feasible, it may be desirable to separate fines of less than 10 mesh size from the flux, and to incorporate these in the concentrate pellets thus minimizing loss of flux fines from the reaction vessel. In addition, dust or fume collected from the flue gas which exits the furnace, and crushed lead dross and other residues may also be conveniently incorporated in the pellets.

The feeding of the concentrate pellets into the furnace is such that a large area of the molten path in the furnace is covered with pellets and a relatively thin layer of pellets is formed on the surface of the bath. The slag and lead bullion can readily and effectively contact and envelope the pellets throughout the smelting zone, the contact being further enhanced by the mixing and agitating action of air or other oxidizing gas introduced through tuyeres 4. The pellet charge thus presents a very high surface area per unit weight for effective heat transfer from the molten bath to the pellets and makes possible efficient utilization of the heat evolved in the bath by reactions (1), (3), (4), (5) and (6) summarized above. While the heat evolved in these exothermic reactions may be sufficient to melt the concentrates and to maintain the molten bath at a suitably high temperature, it may be necessary to supply additional heat in the smelting zone. This can most conveniently be done by situating a supplementary burner 11 in the charging end of the furnace 1. The burner 11 may be of any suitable type and fuels which can be used include coal and natural gas. It has been found that the pellets tend to smelt individually on the surface of the molten bath

without agglomeration into large masses of unsmelted material and very high smelting rates are obtained, for example, of the order of 200 pounds of concentrate pellets per square foot of surface per hour. When flux is not incorporated in the pellets, or is incorporated in part, as, for example, fines, it is advantageous to feed the balance of the flux with the concentrate pellets through charging port **2** in such a way that a large proportion of the surface area of the molten bath in zone **1** is covered with a relatively thin layer of pellets interspersed with lumps of flux. In this way, both the pellets and the flux present a very high surface area per unit weight to heat transfer from the bath. It has been found that there is little or no tendency for the concentrate pellets or flux to agglomerate into large masses and this method of feeding results in greatly increased rates of smelting and hence of oxidation to metallic lead.

A preferred apparatus for feeding the pellets onto the bath is a slinger belt, the speed of which is variable over a given time interval and which may be subjected to a swinging action over a small arc, in a horizontal plane, thus resulting in the spreading of pellets over a wide surface area of the molten bath. However, any other suitable charging apparatus may be employed in the process of the present invention. As the lead sulfide concentrates and flux introduced into the furnace **1** melt as a result of their intimate contact with the molten bath, two separate layers are formed, one of slag and the other of lead bullion. As the lead bullion layer flows toward the discharge end of the furnace **1**, it is continuously subjected to conversion with an oxidizing gas stream, for example air, injected through tuyeres **4**. It is not essential to introduce the gas continuously and some stoppage in introduction is not critical provided the furnace is rotated during such stoppage to expose the tuyeres. However, it should be borne in mind that the overall process of the invention is a continuous process and the introduction of oxidizing gas required for this process is normally continuous. The introduction of oxidizing gas oxidizes the metallic sulfides either to the metal, as in the case of lead, or to metallic oxide as in the case of iron and zinc, in accordance with the equation summarized as (1), (3) and (5) above. The metallic oxides then combine with the silica flux to form a silicate slag which, being less dense than the bullion and sulfide layer, tends to separate out as a supernatant layer.

As a consequence of the reaction between the metallic sulfides and the oxidizing gas stream, the bullion becomes continuously depleted in metallic sulfide until at some downstream point the entire lower layer consists essentially of metallic lead with only a minor proportion of sulfur present. The lead bullion may also contain any silver and gold present in the concentrates, and impurities such as copper, antimony and bismuth.

The lead bullion has a higher density than the silicate slag and tends to settle out as a liquid phase in the quiescent bullion settling Zone **2** of the furnace **1**. The bullion settling into the depression **6** in the floor or hearth of furnace **1** is withdrawn from the furnace either continuously or intermittently through the bullion tapping port **5**. The upper slag layer flows past the tapping port **5** and over the elevated portion **9** in the floor or hearth of furnace **1** into the slag fuming zone. In this slag fuming zone the slag is subjected to contact with a suitable reductant, such as natural gas or pulverized coal which is introduced through a series of tuyeres **7**. The reaction of the reductant with the slag which contains appreciable quantities of zinc oxide and lead oxide, results in the formation of gaseous reduction products, namely zinc vapor, lead vapor and carbon monoxide, which are emitted from the slag. These gaseous reduction products are burned by injecting air through aperture **10** in the tapping end of furnace **1** in order to form zinc oxide fume, lead oxide fume and carbon dioxide and to generate additional heat. Should it be necessary to supply additional heat to the slag fuming Zone **3** this can be done conveniently by introducing a fuel/air mixture through tuyeres **7**. The fuel may be the same as the above mentioned reductant.

The gaseous and vapor products issuing from the molten bath, including Zone **1** and Zone **2** and particularly from Zone **3**, are collected from furnace **1** through a suitable hood over stack **3** and passed through a waste heat boiler where it is cooled. The gas may be cooled further in gas coolers and is then passed into a baghouse or cottrell plant. Precipitated dust and fume from the waste heat boiler and baghouse or cottrell may be recirculated to the furnace if desired, or treated accordingly to known processes for the recovery of zinc, lead and other metal valves therein. The cleaned gas is further treated for the recovery of sulfur dioxide. If desired the slag removed from furnace **1** through slag tapping port **8** may be discarded or, if it contains appreciable quantities of zinc, lead and other metal valves warranting further treatment to recover the same, may be further treated in a separate slag fuming furnace. The metallic lead withdrawn through tap hole **5** is cooled to cause the separation of dross from the liquid lead. The dross may be crushed and recirculated to the furnace if desired, while the lead is subjected to conventional lead refining in order to produce pure metallic lead and to recover precious metals such as silver and gold and by-product metals such as copper, antimony, tin and bismuth.

The process of the invention for the production of lead bullion on a continuous basis from lead sulfide concentrates will now be described in more detail by reference to material balances, heat balances and reactor sizes based upon a charge of 200 tons of lead per day (all tons quoted are short tons of 2,000 pounds). This will make possible a more ready appreciation of the advantages to be gained by working in accordance with the present invention.

I.—FIRST STAGE
(Smelting)

(a) Materials balance:	
Wet concentrates, tons	690
Assumed composition, percent:	
Zn	10
Pb	29
Fe	15
S	30
SiO_2	9
H_2O	7
Total	100
Flux added, tons	25.3
Assumed composition SiO_2, percent	100
Slag produced, tons	316
Volume of slag (specific gravity assumed 3.6) cu. ft.	2,820
Assumed composition, percent:	
FeO	42.0
ZnO	27.2
PbO	3.0
SiO_2	27.8
	100.0
Bullion Produced, tons	196.5
Volume of bullion (specific gravity assumed 9.5) cu. ft.	664
Assumed composition, percent:	
Pb	97.0
S	4.0
Total	100.0
Air required (assumed 95% O_2 utilization) s.c.f.:	
Oxidation of sulphur	2.27×10^7
Oxidation of iron	0.33×10^7
Oxidation of zinc	0.19×10^7
Oxidation of lead (negligible)	
Total requirement	2.79×10^7
Air blowing rate $=\frac{2.79\times10^7}{1,440}=1.95\times10^4$ s.c.f.m.	
Number of tuyeres required (2 in. I.D. at 250 s.c.f.m. per tuyere) $=\frac{1.95\times10^4}{250}=78$	
Length of smelting zone required (determined by number of tuyeres at 6 in. spacing) ft	39
(b) Heat balance:	
Assumed temperature of smelting zone, ° F	2,100
1. Heat required, B.t.u.:	
(a) Evaporation of moisture and superheating water vapour to 2,100° F	$+2.04\times10^8$
(b) Heating charge (concentrate and flux) from 77° F. to 2,100° F. (assume 450 B.t.u./lb.)	$+6.45\times10^8$
(c) Heating air from 77° F. to 2,100° F	$+12.20\times10^8$
(d) Compensation of heat losses from reactor	$+1.87\times10^8$
Net heat required	22.56×10^8

Imponderables including heat of vaporization of volatile components (10% of net heat required) B.t.u	2.26×10^8
Total heat required, B.t.u	24.8×10^8
2. Heat generated by smelting reactions at 2,100° F., B.t.u.:	
(a) Oxidation of iron sulphides (assume all iron in charge as FeS_2)	-10.6×10^8
(b) Oxidation of zinc sulphides	-3.96×10^8
(c) Oxidation of lead sulphides	-1.68×10^8
Total heat generated by smelting	-16.2×10^8
3. Additional net heat supplied by combustion of fuel at 2,100° F.:	
$(24.8-16.2)\times10^8$ B.t.u., equals	-8.6×10^8
Net heat supplied per hour is therefore 36,000,000 B.t.u./hr. (This could be supplied, for example, by burning 3.5 tons of coal at 42% of available heat.)	

II.—SECOND STAGE

(Bullion settling and slag fuming)

(a) Length of bullion settling and slag fuming zone. The length of reactor required for slag treatment and settling may be determined on the basis of an estimated mean residence time of slag in those zones of 2 hours. The estimated slag fall is 117 cu.ft./hr. and the length of the second zone of the furnace is estimated as 18 feet.	
(b) Materials balance:	
Slag input from first stage, tons	316
Reducing gas requirement, s.c.f. (i.e. 1.67 s.c.f.m.)	24.2×10^4
(Assume 95% ZnO in slag is reduced to zinc vapor with CH_4 according to the reaction $CH_4+3ZnO \longrightarrow 3Zn+CO+2H_2O$)	
Air requirement for gas phase oxidation Zn and CO, s.c.f.	2.31×10^5
Slag output, tons	234
Assumed composition, percent:	
FeO	57
ZnO	2
PbO	4
SiO_2	37
Total	100
(c) Heat balance:	
1. Heat Required, B.t.u.:	
(a) Heat required in slag bath (including heat of reduction and sensible heat to reducing gases and reaction products)	$+2.85\times10^8$
(b) Heat losses	$+0.86\times10^8$
(c) Heat required to raise air temperature from 77°–2,100° F.	$+1.01\times10^8$
Total heat required	$+4.72\times10^8$
2. Heat generated, B.t.u.:	
(a) Heat generated by combustion of zinc vapor and CO above bath	-4.84×10^8
3. Excess heat generated $(4.84-4.72)\times10^8$ equals	-0.12×10^8

Although this heat balance indicates that the second stage process is autogenous, it is implied that the heat generated above the bath in combustion of Zn and CO is transferred to the bath at 2,100° F. If this heat transfer process was not efficient additional fuel would have to be burnt in the bath by introduction of a fuel/air mixture through the tuyeres 7.

We claim:

1. A process for the continuous smelting of lead sulfide concentrates to molten lead bullion in a single furnace containing a molten bath and having a charging end and a slag tapping end, said process involving a gradual reaction along the length of the furnace and including the steps of:

a. feeding pelletized lead suflide concentrates containing minor amounts of zinc sulfide through the charging end onto the surface of the molten bath in a smelting zone of said furnace;

b. smelting said concentrates in the smelting zone by heat transfer from the molten bath to the concentrates within the smelting zone;

c. controlling the flow of molten bath in said furnace so that it flows along the length of the furnace from the smelting zone towards the slag tapping end of the furnace;

d. introducing an oxidizing gas into the molten bath sufficient to effect a gradual conversion of the lead sulfide to molten lead, the smelting and conversion reactions being effected essentially in the smelting zone and the exothermic heat of the conversion reaction being utilized for smelting;

e. allowing said molten lead to settle in a bullion settling zone remote from the charging end;

f. drawing off molten lead from the bullion settling zone through a bullion tapping means intermediate said charging end and said slag tapping end;

g. treating the slag flowing past said bullion tapping means in a slag fuming zone with a reductant whereby to reduce zinc oxide in the slag to zinc vapor;

h. introducing an oxidizing gas above said slag in the fuming zone whereby to oxidize said zinc vapor to zinc oxide fume and passing said fume to a fume collecting area; and

i. withdrawing substantially zinc-free slag from said slag tapping end.

2. A process as claimed in claim 1 including the step of supplying to the charging end of the furnace auxiliary heat sufficient to make up the heat balance of the smelting process.

3. A process as claimed in claim 2 including the step of introducing silica flux into the furnace with the lead sulfide concentrates to be smelted.

4. A process as claimed in claim 3 in which the flux is ground to an average particle size below 10 mesh (ASTM Standard Sieve) and is then incorporated in the pelletized concentrate prior to the introduction thereof into the furnace.

5. A process as claimed in claim 3 wherein the pelletized concentrate is separately mixed with the silica flux prior to feeding into the furnace.

6. A process as claimed in claim 1 wherein the pelletized concentrate is fed into the furnace while wet.

7. A process as claimed in claim 1 wherein the concentrates are fed into the furnace in such a manner that substantially the entire area of the molten bath in the smelting zone is covered with the uniformly thin layer of pellets, the feeding of the pellets being so controlled as to prevent the pellets from forming masses of unsmelted material.

8. A process as claimed in claim 1 including the step of supplying to the slag tapping end of the furnace auxiliary heat sufficient to make up the heat balance of the slag fuming process.

9. A process as claimed in claim 1 wherein said reductant is a reducing gas.

10. A process as claimed in claim 1 wherein said reductant is pulverized coal.

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