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The Metallurgical Treatment of an Argentiferous Galena from Burleigh Tunnel. Colorado

By

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The tunnel from which this ore comes, is worked by a company of Massachusetts capitalists into the base of Sherman Mt. near Georgetown Colorado. It is run for the purpose of discovering new lodes of ore, and affording drainage and transportation of material to veins now being worked. The location is on the valley of Clear Creek on the South Pass of the mountain, about three miles from Georgetown. A number of tunnels have been started but few of them seem to be much value on account of the amount of worthless rock that has to be taken out before ore is reached. In 1871 the tunnel had been driven 1200 feet at the rate of 500 feet per year, being 8 ft. wide by 8 ft. and rises one foot in two hundred. It is nearly at right angles to the outcropping veins.

Since 1871 the tunnel has been driven to 1830 feet from the entrance through solid rock, and has cut five lodes, the ore taken from four exceeds in value that taken from mine. One lode 1700 ft. from the portal measures 16 ft. wide and assays 70 oz. of silver to the ton.

Mr. Burleigh thinks that they will soon strike richer lodes, and so is shipping improved machinery in order to double the rate of running the tunnel.

Mr. Lincoln, who visited the place in 1871, says that the country rock is granitic, quartz and feldspar are found alone and all varieties of proportions in combination. Pure hornblende is also found in large deposits.

The ore I had to work came from Lode No. 2 (situated about 900 feet in the tunnel) which is 13 ft. wide, dipping 69° with a strike N 30° W, containing ore in streaks an inch or two in width. The ore is an argentiferous galena, having zinc blende of a dark yellowish color and crystalline structure. Besides this zinc blende are pyrite, quartz, feldspar, chalcopyrite and traces of mica.

The ores of that region are mostly auriferous near the surface, but the silver seems to increase and gold disappear as you go down. This ore has no antimony or arsenic which is of advantage, since these two metals are of great trouble in working. The ore assays $35.73 per ton that is 27.7 oz. at $1.29 per oz. of silver, and has 61.37 % of lead which is 1227.4 lbs per ton. On account of the high price of transportation and working this does not afford a profit in that region. The lead also is not of value there, but when there are means of getting it to the States cheaply it will make this a desirable ore. This lead at six cents per pound is worth $73.64. A mineral analysis of this ore gives:

Zinc Blende (ZnS)	8.32 %
Galena (PbS)	75.86
Pyrite (FeS_2)	12.04
Chalcopyrite	1.53
Gangue (SiO_2 & Feldspar)	0.95
	98.70

The galena was both coarse and fine grained, the pyrite was fine grained.

The ore was principally in large pieces.

I made two determinations of the ore as follows.

			Average
Sulphur	20.56	20.67	20.61
Lead	61.48	61.27	61.37
Zinc	5.54	5.62	5.58
Iron	6.06	6.09	6.07
Copper	.49	.56	.52
Gangue	5.96	5.94	5.95
Silver	.095	.095	.095
	100.185	100.245	100.195

The amount of ore I had to work with was 1274 5/16 lbs. This was crushed in a 5 by 2 1/2 Blake crusher to the size of a walnut, and then put through a pair of rolls, until it passed through a 1/12 inch sieve. The loss in doing this was 16 5/16 lbs., and it took (not counting stops for accidents) two hours and fifty three minutes. Before crushing it was broken into pieces the size of one's fist, and samples of the minerals taken.

In order to analyze a true sample of the ore, I put the whole through a sampler, and saved out 24 1/2 lbs. The sampler used was made of two (one inverted) sheet iron cones with their apices adjoining. The top one had a hole in the apex the size of a cent through which all the ore fell onto the lower cone. The lower cone had four slits in it, under which were four boxes to catch the ore that fell through, and as only a fraction fell through the slits, that was taken as a sample. A lot of ore was saved here for the blast furnace run, and so I really had eleven hundred pounds to use.

	1274 5/16
	174 5/16
	1100
Sample,	21 1/2
For Blast furnace	136 1/2
Lost	16 5/16
	174 5/16

If this ore were smelted as it now stands, it would be an expensive operation for the sulphide of lead is more volatile than the sulphate or oxide (which we get by roasting), and the amount of sulphur to be worked off would require a large amount of flux, so that in practice it is rarely smelted as galena.

Now we have to get rid partly of the sulphur, not wholly since that would be hard on account of the stability of sulphate of lead. When the galena is heated at a low temperature, it is converted into oxide and sulphate, the oxide being volatile causes a loss, while the sulphate is quite permanent.

Zinc blende is also reduced to oxide

and Sulphate, the sulphate is reduced at a white heat to oxide, which goes into the slag and makes it thick and glassy (Percy). Much of the sulphur goes off as sulphurous acid SO_2, with lead zinc and silver when roasting.

The zinc especially is trouble-some in the whole operation, in the high heat, it causes a loss of from 5 to 60 o/o of silver (Plattner), and the trouble with thickening of slag in the blast furnace is caused by it.

For the roasting two reverberatory furnaces were used, one long and flat bottomed, the other shorter with a bottom sloping to a depression at the side. The bed of the long one is 2 ft. 7 in. by 5 feet with a 2' 1" x 1' 7" horizontal grate, having a surface of 425 sq. in.

The hollow bed furnace measured 3 feet by 4 feet with a grate sloping 30°, with a surface of 540 sq. in.

The fires were started and the furnaces well warmed and in working order, before my ore was put on to them, roasting the Newburyport galena and Vershire copper ores.

A long red flame, spreading over the ore, affords the heat to drive off the sulphur, but not agglomerate the ore, if well stirred. For stirring, iron hoes with long handles are used, every five minutes or so. The ore was charged in lots of 85 pounds each, both furnaces having the same sized charge, and were run $3\frac{49}{60}$ hours on each lot. One man was allowed to a charge at each furnace, and the run lasted from two o'clock P.M. Wednesday, when a charge was put in the long furnace, until six o'clock P.M. Thursday when both furnaces were emptied.

During this time thirteen charges were used, the long furnace running $27\frac{20}{60}$ hours, and the hollow bed $23\frac{15}{60}$. To do this in the reverberatory with the hollow bed, $358\frac{3}{8}$ lbs of bituminous coal were used on 510 lbs. of galena, and in the long furnace $296\frac{1}{2}$ lbs of coal were used on 590 lbs of ore; that is, one pound of galena was roasted four hours by 0.59 pounds of coal.

The loss in weight was $\frac{1100}{1056}$ 44 lbs.

No. of charge Long F.	H. B. Furnace	coal	time
1		69 7/8	4 hours
	2	52 3/4	3-40
3		22 3/4	4
	4	83 3/4	3-45
5		31 1/2	3-40
	6	27 1/2	3-55
7		49 1/4	3-50
	8	58 7/8	4.
9		36 1/8	3-55
	10	77	3-50
11		53	3-50
	12	59	3-35
13		34	3-45
		654 7/8	3.49 av.

In the hollow bed the charges used $59\frac{35}{48}$ lbs of coal each, and in the long $42\frac{5}{14}$.

The Sulphur in the roasted product 9.66 % was a loss of 10.95 %, but for each atom of Sulphur driven off, either one atom of oxygen has taken its place to form oxide, or four combines with the sulphur to form sulphate, so the weigh can't be reduced so very much.

$$\begin{array}{r} 20.61 \\ 9.66 \\ \hline 10.95 \end{array}$$

The amount of lead in the roasted product was, by two analyses, 58.05 % and 58.35, average 58.2, and silver by two assays .096 % which is 14.76 oz in the lot a loss of 0.48 oz of silver in roasting.

$$\begin{array}{r} 1056. \\ 58.2 \\ \hline 614.59 \end{array}$$

Amt. of lead in raw galena	675.07
" " " " roasted "	614.59
Loss " " " " "	60.48

As this ore was rich in lead and had little Silica, the Flintshire process of Smelting in the reverberatory to metal, matte and Slag was tried. In this process the undecomposed PbS is acted on by PbO, and lead separated and the $PbSO_4$ is also acted on by PbS and Pb separated, as follows $PbS + 2PbO = 3Pb + SO_2$, and $PbS + PbSO_4 = 2SO_2 + 2Pb$. If much Silica were present it would prevent this reaction by the formation of a Silicate of lead.

The ores on which this is used, generally have over 75. % of lead in them, and often are nearly pure galena. The heat has to be more intense than for roasting, in order to decompose the sulphate and charcoal, by furnishing carbon to unite with the oxygen, helps break it up. The fire was lit at 2-25/60 o'clock A.M., and at three the fire was 8 inches deep and a good flame on, at 4-45 there was a strong roasting heat and the furnace was ready, but I was not, so my charge was put in until 8-10, and when this was pasty and bubbling powdered coal was added, but in vain, for I could not get enough heat to melt the ore. So I gave up trying to reduce it and only heated to a pasty mass in a few minutes, and withdrew the charge. Other charges were only heated for an hour, and were stirred harder than in roasting, because to keep a fresh surface to the blaze is harder work when the ore is pasty than when dry.

Time start.	Time stop.	Time	Ore	Reagent Coal.
8-10	10-55	2-45	132 3/8	10 3/4
11.	12-45	1-45	123 7/8	10
12-55	1-55	1.	185 1/8	
1-10	2-10	1.	203 7/8	
3-30	4-30	1.	208 7/8	
4-35	5-30	.55	201 5/8	
			1056	

The amount of coal used for heating was 546 5/8 lbs. This was done in the hollow bed furnace. The total time of agglomerating was 9 hours 20 minutes. The loss of sulphur was $\frac{9.66}{6.52}$ 3.14 o/o, of lead was

Lead in roasted 614.59
" agglomerated 604.52
10.07

10 lbs., the agglomerated ore having 60.68 o/o of lead in it, there being 996 1/8 lbs of ore, the loss of weight being

1056
996 1/8
59 7/8

59 3/4 lbs. Some minute pieces of lead were found in this ore, .0125 grms. in 361. grms.

The causes of failure seem to be, not enough heat, and the zinc blende, which with the silica also would make a very infusible slag though 10 o/o might have been run down perhaps if a higher heat had been obtained. The slags from this process are rich (having often 55. o/o of lead) and are run down in the blast furnace in practice. This was then crushed to the size of a walnut, and placed aside, until I could run the blast furnace.

Blast Furnace.

To get the lead away from the other ingredients of the ore, tap cinder is used. This furnishes iron for the sulphur and silica, the sulphur having a stronger affinity for iron than for lead, the silica combining to a fusible slag which flows off. It is important to add enough tap cinder and not too much, for if an infusible slag is formed the furnace will be choked up and the operation stopped. The iron takes the place of the lead, or precipitates it, so this is called the iron reduction process.

To know how much cinder

we need, an analysis has to be made, and I got the following results. The cinder is call tap cinder from puddling furnace, and this I used cause from the "Bay State Iron Works" in South Boston. As it is not easy to get this cinder in the right proportion, limestone is added, to partially take the place of iron, and magnetite. so these had to be analysed as follows.

SiO_2	17.09	16.95
FeO	72.68	72.76
S	.54	.61
Al_2O_3	5.97	6.16 (6.05)
CaO	1.35	1.30
P_2O_5	3.82	3.68
	101.45	101.46

	SiO_2	FeO	Al_2O_3	CaO	S
Tap Cinder	17.0	72.75	6.06	1.9	.58
Lime stone	0.6	0.00	0.0	55.6	0.40
Magnetite	5.75	80.00	—	—	—
Galena	5.95	7.79	—	—	6.52

A slag of silica 30 to protoxide of iron 56, we know is a good one from previous runs, so we try to get that ratio leaving enough iron to form a matte with the sulphur. First I find the amount of this slag that exists in the tap cinder, and see how much extra FeO I have for the silica and sulphur of the ore.

30 : 56 :: 17 (amt. of SiO_2 in cinder) : x   x = amt. of Fe used by cinder itself   x = 31.7 FeO

Therefore $\begin{array}{r}72.75\\31.7\\\hline 41.05\end{array}$ 41.05 o/o of the FeO in the cinder is at my disposal. Next I want to know how much sulphur combines with FeO to form the matte. The atomic weight of the sulphur : the atomic weight of FeO :: one lb. of sulphur : x the weight of FeO required to flux it :: $32 : 72 :: 1 : x$, $x = 2.25$ FeO. Now we have in the 1000 lbs. of ore 65.2 lbs. of sulphur, therefore $\begin{array}{r}65.2\\2.25\\\hline 146.7\end{array}$ 146.7 lbs of FeO are wanted for the sulphur. 41.05 o/o of the tap cinder can be used for this, therefore it will take 357.3 pounds of cinder for our matte.

To use up our SiO_2 in the ore, 59.5 pounds $30 : 56 :: 59.5 : x$ $x = 111.06$ = the amount of FeO wanted to flux the silica of the ore, of this 77.9 lbs are furnished by the ore itself, $\begin{array}{r}111.06\\77.9\\\hline\end{array}$ 33.16 lbs, then we will get from the cinder $41.05)\underline{3316.}$ 80.7. The amount of cinder in all is $\begin{array}{r}357.3\\80.7\\\hline 438.\end{array}$ 438 pounds. The pounds of the constituents are given here.

	SiO_2	FeO	Al_2O_3	CaO
Cinder	74.4	318.6	26.5	8.2
Ore	59.5	77.9		
Total	133.9	396.5	26.2	8.2

= 564.8 pounds in slag and matte. then subtract the matte which is 146.7 $\begin{array}{r}564.8\\146.7\\\hline 418.1\end{array}$ and we have 418.1 pounds of slag, and calculating the percents from the weights given, we see how near the wanted slag they come, and then add lime and magnetite to fix them right. I did not have to use any magnetite for I got a good proportion without. Add to the weight of the Slag 27 lbs of lime, and we have a weight 445.1 of which 30.08 is SiO_2. This is the calculated slag.

SiO_2	30.08
FeO	56.12
CaO	7.91
Al_2O_3	5.89
	100.00

To supply the lime (as there are 55.6 o/o of lime in this marble I used) it took $55.6)\frac{27}{48.56}$ 48.56 lbs of marble.

Taking 30 lbs. of ore to one charge, dividing the limestone and cinder up proportionately, we have for charge 30 ore, 13 cinder, 1 3/8 limestone, and 10 coke, taken from amount of coke used in previous runs. One pound of galena was also added to act as a purifier, just how, I don't know.

Next, the furnace is a small cupola, being 4 ft. 9" from bottom to feed door, the walls are 1 foot thick and are lined with fire brick on the inside. The inside measures 11 in. by 11 in.

The feed door is 11 in by 1 ft. 3 in and the front is 11 in by 1 ft 11 in. The very bottom of the inside shaft has a bed of broken chips of fire brick firmly pounded down to a thickness of 5 inches and on this, layers of brasque are pounded and hardened by a red hot iron. The layers are built up until the top is 12 inches from the bottom of the shaft, when a basin is scooped out approaching the sides and back until within an inch, being 3" deep in front and sloping upward at the back to the level of the top. A channel is cut leading out to the front and then the front is bricked up.

Only the small channel runs out of the furnace, and it was expected that the ore would be reduced in the lower part of the furnace, and the metal, matte, and slag run into the basin; the slag being lighter would float, and as soon as the basin filled, would begin to run off and then, when the lead came out, a hole would be cut, tapping the lowest point of the basin and let the lead out.

This operation to be kept up all along through the operation. The slag running out in a thin stream over an iron nose into a "buggy" will show metal when it appears by the different thickness and color. The buggy is simply an inverted pyramid set on legs, it is 12½ in square at the top, and 3 in square at the bottom; it is made of iron and is dragged around with an iron hook. Before using it has to be "swabbed", with a thin layer of fire clay, put onto the iron, when of a milky constituency. The blast is supplied by three ½ inch tuyeres, one at the back and one on each side driven by a no. 0 Sturtevant blower, with a pressure of four oz. of blast on each tuyere, given by 5508 revolutions per minute

The furnace was all fixed the day before and a fire started on the inside at half past one P.M., in order to warm up the furnace before using. Three hods of charcoal and four of coke kept the fire going until the next day when we were ready to put the blast on and begin. Before the ore was put in, some Revere Copper slag was put into the furnace just to start it going.

Time	Remarks	Time	Remarks
9.35	Blast on	2.40	Buggy 18. Probably good
10.	Flame at tap hole	" "	" 19. Lead
10.30	Fed slag	" 45	" 20. Perhaps lead
10 45	Slag at tap hole	" 55	" 21. " "
11.	Slag running freely	3-15	" 22. " "
11.05	Buggy 1. 22 lbs	" 25	" 23. Lead
" 15	" 2. 18 1/4 lbs.	" 35	" 24. Poor
" 35	" 3. 28 1/4 lbs.	" 52	" 25. "
" 40	Metal at tap hole.	4-10	" 26. Lead
" 47	Buggy 4. Foul.	" 15	" 27. Lead on top
" 57	" 5. "	" 22	" 28 Foul on top. Separatin
12-12	" 6. "	" 36	Blast off.
" 16	" 7. Steep and dirt.	" 37	Tapped. #29. Foul
" 20	" 8. Lead.	" 42	Blast on #30. Lead
" 23	Blast on.	" 55	Buggy #31. not good.
" 28	Slag appeared.	5 10	#32. Foul.
" 40	Buggy 9. Foul.	" 11	Blast off. #33. Steep.
" 50	" 10. "	" 15	Tapped.
1-05	" 11. Settled.	" 17	Blast on
" 20	" 12. Foul on top	" 21	#34. Lead.
" 35	" 13. Probably settled	" 25	Blast off.
" 45	" 14. Fairly settled	" 30	Tapped.
" 50	" 15. Tapped. Lead.	" 36	Blast on. #35 Lead & Settl
" 52	Blast on.	" 53	#36 Steep. Little lead.
" 58	Slag at tap hole.	" 58	#37 Lead
2-10	Buggy 16 Probably good	6+	Stopped.
" 25	" 17 " "		

Feed Door Record.

Time	Charge	Height
mch. 8. P.M. 1-30	3 hods charcoal.	
4-	4 " coke.	
mch. 9. A.M. 9-35	Blast on.	
" 50	1 Coke.	
" 55	1 Coke.	
10 20	1 "	
" 30	23/8 Revers 1 Coke	2 ft.
" 40	25 " "	2 1/4 ft.
" 58	I	2 1/4 "
11-03	I	2 1/2 "
" 15	I	3 "
" 26	I	3 1/4 "
" 39	I	3 1/2 "
" 57	I	3 1/2 "
12.05	I	3 2/3 "
" 30	I	3 3/4 "
" 45	I	3 3/4 "
1-10	I	" "
" 30	III	" "
" 50	II	" "
2-05	III	4 1/2 "
" 29	III	4 "
" 45	III	4 1/4 "
" 52	III	5 "
3-24	III	5 "
4-12	IV	4 1/2 "
" 38	IV	4 1/2 "
5-17	IV	4 1/2 "
" 33	IV	4 1/2 "
5-58	IV	4 3/4 "

Charges

	I.	II	III	IV
Ore	30	30	30	30
Cinder	13	13	13	13
Limestone	1 3/8	1 7/8	1 7/8	1 9/16
Galena	1	2	2	5
Coke	10	8	10	8

To find the amount of coal used.

10 charges of	I	100
6 " "	III	60
1 " "	II	08.
5 " "	IV	40
		208 lbs

Put in first, four hods of coke and three hods charcoal which are equal to $\frac{14}{4}$ 56 lbs coke and $\frac{6}{3}$ 18 lbs charcoal 208+56=264 coke

The slag ran in too thick and viscid a stream to keep molten long, and it did not allow the lead to settle out much inside the furnace, but took it along with it, so as it was thick, the lead did not settle when collected in a buggy outside, but cooled in with the slag.

The table shows that none of the buggies made a complete separation, but had the lead scattered all through the slag. We put charcoal onto the hot slag in the buggies to keep it cold until it could settle, but it did not do much good.

The furnace was stopped by the babbitt metal melting out of its case on the shaft that ran the blower, then when the front of the furnace was rocked in, the coke was found to be nearly at the top of the shaft. This shows that most of the air was used up near the tuyeres by the coke, and the ore was reduced far above, so that the lead trickled through the hot coals and much was volatilized in this way. This shows that we had been using too much coke for this ore, although it had not been so much for any ore that was worked before it was. The slag was not fairly hot at any time. The matte and slag did not separate, but were mixed so that no matte was visible, though Percy says that in some ores rich in zinc blende, the slag has the matte mechanically mixed with it, only visible to the microscope.

To this zinc blende they ascribe most of the trouble, for it forms an oxide or silicate, very infusible, which goes into the slag and thickens and cools it.

From this run I had slag stuff (brasque) rich in lead globules, slag from which most of the lead had settled, lead, coke, and unsmelted charges which the fire had not touched. It took two days to clean up the lote, and break off what lead there was in limfie. The slag was weighed, crushed, put through the rolls and sifted through a 1/4 in. sieve. Then to get out what lead ~~that~~ had gone through the sieve in globules, the spitzlutte was used, but this made no separation, on account of the weight of the matter.

being too near that of lead to work well. The end bumping table was next used to concentrate into a smaller bulk the lead portion, but after working the lot, it was found that there was too little lead to work for. So it was dried and sampled, and a portion saved.

The rich stuff was saved to be worked with that from second run, this is the stuff that was dug out in making the tapping hole to let the lead out.

The portion of the slag that passed over the sieve was mostly lead, and was saved as such. The coke was used in my second run, the unsmelted ore also. The lead was put with the lead from second run.

Before my second run the furnace was used by other men, and was relined, and some 3/4 inch tuyeres put in in place of the 1/2 inch, giving a blast of about 8 ounces. A new way of working was also tried, the bosh instead of being on the inside, was made outside the furnace.

The inside of the furnace was fixed the same, without cutting out the basin, being of small changes than to the front of the furnace. The basin was made in a large iron box attached to the front of the furnace, the bottom was built of firebrick and then steep pounded in, and a basin of 1¼ ft. cut out. In the box were (on the front part) three spouts, one leading to the bottom of the basin a little above the level of the floor, the other two a little below the level of the top of the basin. The slag and metal were to flow into this basin, the lead and matte settle, then as it filled up the slag would flow off in a continuous stream, and the lead and matte tapped out of the lower spout into an ingot mould. This basin was heated at the expense of much charcoal, a fire being made in it the day before the using and a fire being kept on the slag all the time. Only one of the upper spouts was used. As soon as the ore was reduced, instead of staying in the furnace as before, it ran out into a colder place and less lead was consequently lost. The over flowing slag ran into a ...

Feed door Record

Time	Interval	Charge	Depth	Rise	Remarks
Mar 26. 1.45		80 lbs coke			Fire Lit. 18 hods of charcoal
Mar 27 9 25		12 1/4 coke	1 1/4		Blast on. used on front
" 33	7 mts.				" " basin add two
" 39	6 "	14 1/2 "	1 1/2 ft.	1/2 ft.	
" 43	4 "	13 3/8 "	1 1/2 "	1/2 "	more for night before
" 46	3 "	13 "	2 "	1/2 "	
" 47	1 "				Blast off.
" 56	9 "				" on. = 20 hods = 12 lbs
10-18	22 "	14 coke	2 "	1/2 "	
" 33	15 "	I	2 "	1/2 "	coke
" 40	7 "	I	2 "	" "	
" 47	7 "	I	2 1/4 "	" "	
" 52	5 "	V	2 1/2 "	" "	
" 58	6 "	V	2 1/2 "	" "	Dull flame in shaft.
11-05	7 "	VI	3 "	" "	
" 15	10 "	VI	3 1/4 "	3/4 "	No flame after charge.
" 26	11 "	VI	3 1/4 "	3/4 "	" " " "
" 36	10 "	VI	3 1/4 "	3/4 "	" " " "
" 54	18 "	II	2 3/4 "	1/4 "	Ore +
12-06	12 "	III	3 "	1/2 "	No " " "
" 17	11 "	III	3 "	" "	" " " "
" 32	15 "	III	3 "	" "	" " " "
" 43	11 "	VII	3 "	" "	" " " "
" 55	12 "	VII	3 1/4 "	" "	" " " "
1-04	9 "	VII	3 1/2 "	1/2 "	" " " "
" 19	5 "	VII	3 1/2 "	3/4 "	" " " "
" 24	5 "	VII	3 3/4 "	1/2 "	Began to use charge of things
" 27	13 "	VIII	4 "	3/4 "	from Exp. St. [illegible]
" 31	4 "				Blast off
" 35	4 "				" on
" 48	13 "	VIII	3 3/4 "	3/4 "	
2 00	12 "	IX	3 3/4 "	3/4 "	
" 18	18 "	IX	3 1/2 "	1/2 "	
" 30	12 "	IX	3 3/4 "	1/2 "	
" 48	18 "	XI	3 1/2 "	1/4 "	
3-07	19 "	XI	3 "	1/4 "	
" 48	41 "		1 3/4 "		
4 14	26 "	XII			

Charges		
		80
		12 1/4
		14 1/2
		13 3/8
		13
3	I	= 24
2	V	= 20
4	VI	48
1	II	10
3	III	30
5	VII	40
1	XII	13 7/8
3	IX	0
		333 lbs. left
2	XI	0
2	VIIII	0

Ore	I	II	III	IV	V	VI	VII	VIII	IX	XI	XII
Ore		30	40	45			40	60	60		
Galena		1 3/8	1 3/4	1			1 3/4				
Cinder		13	17 1/3	19 1/2			17 1/3				
Coke	8	10	10	10	10	12	8	8	9	8	13 7/8
River slag	30				40	40		~~7~~	~~7~~	30	
Lime		1 3/8	1 3/4	2			1 3/4				
	1:4	1:45	1:6	1:6.9	1:4	1:35	1:7.5	1:7.5	1:7	1:4	

Tap Record.

Time	Patent	Buggy	Remarks
1051	1		Slag in sight, running into basin from furnace.
1127	36		Slag running into Buggy.
" 30	3	1	Clear Slag
" 35	5	2	" "
" 41	6	3	" "
" 47	6	4	" " Settled.
" 55	8	5	" "
12 –	5	6	" "
" 06	6	—	Lead fumes
" 07	1	7	Settled
" 19	12	8	Not settled
" 33	14	9	" "
" 46	13	10	" "
" 57	11	—	Tapped.
" "	0	11	Not settled.
1-08	11	—	Slag ran again
" 16	8	12	
" 20	4	13	
" 30	10	—	Tapped. Blast off.
" 32	02	14	Settled Blast on
" 40	8	—	Slag appeared
" 48	8	15	
2-00	12	16	
" 08	8	17	
" 15	7	—	Tapped
" 16	1	18	
" 23	7	—	Slag ran
" 37	14	19	Poorly settled
" 48	11	20	
3 00	12	21	Tapped.
07	7	—	
20	13	—	Slag appeared
30	10	22	Contains Reverse Slag
45	15	23	
54	9	24	
4 03	9	25	
10	7	26	
20	10	—	Slag disappeared
25	05	—	Tapped. Blast off
30	05	27	1/2 Full..
35	05	—	Tapped
40	05	—	Front Broken in

The faults of the run were too much coke, slow rate of slag flow and charging, and hot tuyeres. The low state of the coke for first half hour also retarded operations. Below 2 ft. This last made the lowering of the amount of coke in the charges a doubtful thing at first, but it soon corrected itself but was not appreciated in time, and the reduction in the ratio of the fuel to other materials from 1 coal to 3 Slag at 11 o'clock, to 1 coal to 8 1/2 ore at

12 1:6 Ore
12-43 1:7 1/2
2 – 1:8 1/2

two o'clock though steps in the right direction, did not succeed in putting the furnace in good order. A longer run would have fixed this. This was a much better run than the last, the lead all settled in the basin together with a lot of matte, and the slag, that ran into the buggies had no specks of lead in it, though it did not form a cake from which the matte separated, and a sample could be taken of the slag for analysis. The lead only had some matte on it. There were a foot and a half of coke in the furnace when the front was broken in, from which all slag had been melted out. The basin in front had a large lot of matte in it, from which a sample was taken, and a little lead, the slag was in a good lump, and would have given a good sample, if there had been no severe slag in it.

The slag was picked over the following day and examined but no sample of slag good enough for analysis found, so the lot was thrown away.

Owing to this mixing of matte and

slag, no analysis of the slag could be made to compare with the calculated slag. The rich steep of both runs was ground and what passed over a ½ in sieve was mostly lead the remainder was sampled and analysed and thrown away.

Products. Steep 312½ lbs holding 10.56 o/o of lead. Slag from first run having globules, and lead chemically combined 323½ lbs analysing 7.84 o/o of lead, slag from second run, no globules, 5.1 o/o of lead.

782 lbs lead slag and sieve - 433 lbs. sieve 349 lbs.

$$\frac{312\tfrac{1}{2} \times 10.56}{33.000} + \frac{323\tfrac{1}{2} \times 7.84}{25.36} + \frac{349 \times 5.1}{17.79} = 76.15 \text{ lbs}$$

The matte had 1.82 o/o of lead $\frac{15.8 \times 1.82}{28.75}$ $\frac{.28}{76.43}$ 76.43 lbs. of lead in slag and waste products. The lead in ingots and pieces was 493⅞ lbs, but this had matte, slag, copper etc amounting to 56.0 lbs. leaving me 437⅞ lbs of lead.

Lost in slag 76.43 plus the lead I have 437.87 gives me 514.3 lbs. of lead accounted for. the rest went up the chimney as lead fume.

Lead in agglomerated ore	604.5
Lead in slag and pieces	514.3
Lead that flew away	90.2 lbs.
	76.4
Lead fume plus loss by slag equals	166.6 lbs

In the blast furnace runs, one man was in charge of the feed door and two men at the tapping door.

As no slag sample could be made I took 13 lbs. of the mixed slag and matte, and fused them under charcoal in a plumbago crucible. To do this I started a fire, in a 12 in by 12 in sized furnace, with hard coal and while the coals were still black, placed the crucible on them upside down. As soon as the bottom was warm, the cover was placed on the furnace, until the edges of the crucible were red hot, when the crucible was turned over onto its bottom and packed in with coal, then I put in the charge of slag etc and luted the cover on to the furnace with ashes, to keep the air from getting at the top of the fire. The slag was first mixed with some fine coal dust to help the fusion. This fire was started at one o'clock, at 1-15 the crucible was put in, at two the charge was put in, at four the charge was fusing and at five twenty it was fused and taken out of the furnace. The crucible is lifted out with iron tongs, and the fused mass poured into a buggy and cooled. On this fusion, I used one hod charcoal and two of anthracite coal.

The fused mass gave on top a layer of slag, then one of matte, then one of a cast iron, then one of lead.

From this, the slag must have chang'd considerably for most of the cast iron must have come from the slag, and at such a high fusion much of the zinc and lead must have been distilled, so the slag would show comparatively little of the real composition of the true run slag. The proportions in the fused sample were by weight, Slag 67.06, Matte 16.94, Iron 15, and lead one, the proportions were taken from the weight of the cake which was three pounds less than charge. The per cents calculated on the first charge were as follows

Slag	51.58	Slag, matte and iron	86.14
Matte	13.03	Lead	.76
Iron	11.53	Lost	23.10
	86.14		100.00

An analysis of the slag gave

SiO_2	45.13
FeO	26.35
$P_2O_5 + Al_2O_3$	13.09
CaO	9.60
ZnO	3.50
PbO	1.25
MgO	1.72
	100.55
Sulphur	.76
	101.31

Of the SiO_2, $Al_2O_3 + P_2O_5$, FeO I got two results, respectively 45.13, 26.29, 13.05. The determinations of CaO, ZnO, MgO were made for me by Mr. Powell who kindly helped me, for I had not the time to make them myself. This fusion goes to show that there was much matte formed and that the iron can be smelted out from the slag. After all the fusing it is seen there was still some of the troublesome zinc left in the slag, and very likely some in the matte.

All the lead was now collected, consisting of ingots from good tappings in second run, lead that had settled in the buggies, and pieces that had been collected in sifting. These were fused, or refined under charcoal in plumbago crucibles. When melted the slag and dirt are skimmed off from the lead and fused separately. The lead is poured into ingot moulds if clean, but if not, it is poured into a buggy and the matte or dirt allowed to collect on the top and pounded off when cold, and then the ingot is remelted and cast into ingots of six or ten pounds weight.

This refining also drives off most of the sulphur that has gone into the lead.

The pieces of lead were so irregular in shape, that it was not possible to get all the slag off the cakes that came from the first run. Often I could not get the whole charge into the crucible at first, but had to wait until part had melted, before I got it all in.

The crucibles held nearly sixty pounds of lead, but forty was what I generally put in, as sixty pounds of molten lead are hard to manage when the heat of the furnace is as great as it had to be for this purpose. I used the same furnace that I used in getting my slag sample, and another one measuring 14 in. by 14 in. It took three days, using both furnaces, to do this refining, part of the time in fusing the skimmings (from the crucibles) which were rich in lead.

Furnace No. 2

Time	Interval	Notes	Charge	Remarks
8-50		Fire lit.		March 31. 1876
9-00	10.	Crucible in		Furnace #2 14 by 14.
" 10	10.	" warm		5 hods Anthracite = 120 lbs 6 lbs charcoal
" 35	25.	Lead in	36 5/8 lbs	Slag Lead
10-25	50.	Began to skim		
1-15	2 hours 50	Crucible in		Lead 36 5/8
2-30	1-15	Lead in	52 lbs	Ingot Lead 52
3-30	60	Poured		36 5/8
" 35	5	Lead in	36 5/8	Slag Lead. 44
4-40	65	Poured		7 18 1/4
" 45	5	Lead in	44	Ingot Lead. Lead taken 194 1/2 lbs.
5-25	40	" Out		
8-40		Crucible in		April 1st
9-10	30	Lead in	44	Remelting Ingot Remelted 44
10-25	75	" out		31 3/4
" 30	5	Slag in	7 lbs	Slag with little lead
2-55	265	" Out		Poor Fusion
3-00	5	Matte in	18 1/4	
4-10	70			
5-05	65	Poured	Good heat and fused	
9-17		Fire Lit		April 4 3 Anthracite 24 72 lbs
" 55	38	Skimmings in	41 5/8	1 Charcoal 6 lbs
11-15	80	Melting		
12-00	105	Out.		
2-30	30	Lead in	31 3/4	Ingot from skimmings.
4-30	120	" Out		

Furnace No. 1

Time	Interval	Notes	Charge	Remarks	
8.55		Fire lit.		March 31. Furnace 12 by 12	
9-05	10	Crucible in		Coal 7 hods anthracite and	
" 20	15	" over		one of Charcoal. 168 and 6 lbs.	
" 45	25	Lead in	40 lbs	Ingot.	40
10.48	63	" Out			25
" 50	2	" in	25	Ingot.	40 5/8 47 1/4
12.15	85	" out			44 6/8 45
" 20	5	Lead in	40 5/8	Slag.	20 7/8 15 1/4
1-05	45	" Out		Lead Slag and matte	278 3/4
" 10	5	" in	47 1/4	Ingot. Furnace #2	194 1/2
2-00	50	" Out			473 1/4
" 10	10	" in	44 6/8	Ingot.	
" 45	35	" Out		Of this	44 48
" 50	5	" in	45	Ingot.	54
5-15	145	" Out			31 1/4
8.30		Crucible in		April 1st.	177 1/4 lbs
" 50	20	Lead in	48	Remelting Ingot.	were remelted
10-15	85	" Out			From these
" 20	5	" in	54	Remelt. Ingot	fusings I
11-05	45	Lead out			got matte
" 20	15	Matte in	20 7/8		Lead and
1.30	130	" out			Slag.
" 40	10	Slag in	15 1/4		

Taken out about 5 o'clock.

There were 410½ lbs. of lead in ingots, 15 5/8 lbs of matte and 41 5/8 lbs of skimmings, making 467 6/8 lbs in all, a loss of 5½ lbs by volatilization during the process of refining. From the 473/8 lbs started with, I got 441 lbs of ingot lead, 15 5/8 lbs of matte, and 11 1/8 lbs. of slag. It took for the whole operation 360 lbs of coal and 18 lbs of charcoal.

473/8
467 6/8
— 5½

Sweating.

This lead still had in it such impurities as copper and sulphur. These can be separated by heating the ingots enough to melt the lead, but leave the alloys of copper etc. solid, so that the liquid can be drained off into moulds. What solid particles are carried down with the lead, can be skimmed of while liquid with a stick, and thrown onto the heater.

The heater used was an iron plate, set on four legs and heated by some large Bunsen burners. The sides are inclined toward the middle of the plate, where there is a channel, and the whole plate is slightly inclined toward the lower end, in order that the lead may run easily.

Under this central channel is a row of the burners, and under the upper part of the plate are eight burners in four rows, which melt the lead. The lead being melted runs down the channel and after it has been going a little while, a coating of litharge forms over it, keeping much of the dirt back. A ladle is placed at the lower end to catch the lead, and is kept hot by another bunsen lamp. The plate is 2 ft. 6 in. by 10 in. and has to be "swabbed" before using. About 40 lbs. of lead can be put on at once and when running well, about ten ladles full (weighing 7½ lbs.) can be run in an hour. The lead is kept from the cool air (while melting) by a sheet iron cover, which fits over it. This operation took 11.3 minutes per ladle. The amount of lead refined and ready for the next operation was 390½ lbs., scum 48 lbs. making a loss of
441
438½ 2½ lbs. The lead analysed 98.95 o/o fine, by two analyses 98.85 and 99.05.

The scum was smelted along with the matte from the refining, with enough galena to form a sulphide of copper (CuS) with the copper in the scum. As I had not time to analyse the scum, I took it for granted that all the copper, not in the matte, was in that, and so added galena enough to form Cu_2S with it.

The matte had 24.83 % of sulphur, 1.82 % of lead, and .024 % of silver, 9.53 of copper.

$$\begin{array}{r} 9.53 \\ .15.8 \\ \hline 1.50 \end{array} \qquad \begin{array}{r} 11.00 \\ 5.2 \\ \hline 5.72 \end{array} \text{ lbs}$$

There were then 4.22 lbs. of copper not in the matte, and let x equal the amount of sulphur granted, then we have $x : 4.22 :: \overset{\text{at. wht of}}{32} ^S : \overset{\text{at. wht of } Cu}{63.4}$ $x = 2.12$ lbs.

The galena has 20.6 % of S, therefore 10.2 lbs. of galena are wanted $.20.6)\,\frac{2.12}{10.2}$

So, for this fusion, we have 48 lbs of scum 15.8 matte and 10.2 galena. I made two charges of it, and used four hods coal $\frac{24 \atop 4}{96}$ 96 lbs and one hod of charcoal 6 lbs.

At 9.10 Fire lit - 9.30 crucible in - 10 - Lead in - 1.10 lead out - 1.20 second charge in - 3.50 charge out.

First charge gave 10½ matte and 21½ lead

Second charge gave 8¼ matte and 23½ lead

Making a total of 18¾ matte and 45 lead

The scum then lost $\begin{array}{r} 45. \\ 6.2 \\ \hline 38.8 \end{array}$ $\begin{array}{r} 48. \\ 38.8 \\ \hline 9.2 \end{array}$ 9.2 lbs. of dirt, sulphur etc. The 6.2 lbs came from the galena. In the 9.2 lbs. are the slag etc from the galena. These 45. lbs were sweated and

gave 39 1/4 lbs of refined lead. So for the next operation I had 390 1/2 + 39 1/4 = 429 3/4 lbs of pig lead. This assayed .197 ozs of silver.
The last residue was 5 1/4 pounds but as it was very impure it was left alone.

Zincing

In order to cupel the lead and get the silver out, it is first necessary to concentrate the silver into a smaller lot of lead. I used the Parkes's process of zincing as follows. if the lead is melted nearly to driving, and then molten zinc is stirred in, the zinc will take the silver away from the lead, and as the mixture cools the zinc will crystallize and cool on the top, where it can be skimmed off, and the lead left in the pot comparatively poor in silver. Of course the zinc will take very much lead with it, but much can be sweated off. Here I differed from the regular way when the zinc had been stirred in for ten minutes, I ran it into ingots, and sweated them, the zinc requiring a higher temperature than lead for melting, does not melt, but stays, along with some lead on the iron where it can be scraped off. I melted up 226 3/4 lbs. of lead in an iron pot (which had previously been thoroughly "Swabbed"), to the required point, and put in one per cent of zinc, which I melted in a hessian crucible at a side fire. The zinc was heated, so that when exposed to the air, it burned to zinc white. The lead must be well stirred while the zinc is being put in, or it (the zinc) will float on top (being lighter), and not extract the silver.

Then I zinced the remainder 203 lbs., as I had the first lot. The first charge filled the pot too well, to be stirred easily.

After this zincing, I sweated the whole lot, and zinced twice with 1 o/o of zinc each time, and sweated after each zincing. Adding the zinc in this way, is better than adding it all at once, because it requires less zinc, and is more effective.

The silver varied as follows:

	Lead	1st Zincing	2nd " Zinc.	3rd "	Zinc Crusts
Silver	.197 o/o	.0525 o/o	.0035 o/o	.001 o/o	.97 o/o
Amount	12.33 oz	3.09 oz	.195 oz	.050 oz	9.73 oz

The oz show the amount of silver in the lead after sweating, the per cents at the same time also. To get the amount of silver in the pig lead, it was so poor, that I had to take 100 grammes for analysis, to get a button that I could weigh.

1st Zincing			2nd Zinc.		3rd Zinc.	
Time	Interval	Note	Time	Interval	Time	Interval
9.30		Fire lit and lead in	10.		9-15	
11-20	1 hour 50 mts	Zinc in	1.	3 hours	11	1 hour 45 mts
11-35	15 mts.	Charge out	1-10	10 mts.	11-10	10 mts.
11-40	5 "	2nd Charge in	1-20	10 "	11-15	5 "
1-05	85 "	" Zinc Out	2 -	40 "	12-05	50 "
1-15	10 "	" Charge out	2-10	10 "	12-15	10 "

3 charcoal, 3 coke for melting zinc, and six hods of bituminous coal were used.

After zincing I had 84 3/8 lbs. of crust, and 356 1/2 zinced lead, which would make only 7/8 lb. loss in this operation, that is the lead and zinc and silver in the crust should weigh 85 1/4 lbs. instead of 84 3/8. I had to use the heater for sweating and do the zincing at the same time, so I could not take the time it took to sweat the ingots all through, as I had to stop sweating often to look after.

429 3/4
356 1/2
73 1/4
12 zinc
85 1/4

the zincing. For the first twenty ladles full, it took an average of 7 9/10 minutes each.

Distilling. Before the crust can be cupelled, the zinc must first be got rid of, and this is done by the distillation of the zinc in plumbago crucibles.

The charge is melted down, and lumps of charcoal put on top, the fire is raised to a good red heat with the cover on, and left for about half an hour, until the charge is well heated up, then the cover is partly taken off to allow air into the crucible. The oxygen unites with some zinc to form zinc white, but most of this is reduced by the carbon, and zinc and carbonic acid formed, both of which go up the chimney; if any lead should oxidize it is also reduced, and being less volatile than the zinc, it does not go up the chimney but goes back into the charge. In this way all the zinc is driven off, and some lead if the heat is too high. When all the zinc is driven off the lead in the crucible has a bright look, and colors show on its surface. The two corner furnaces were used.

Time	Interval	Furnace No. 1
9.55		Fire lit
10-10	15 mts	Crucible in
" 55	45 "	Lead in 40 lbs.
5 50	6 hours 55	Finished
12-35		Fire lit Furnace #2.
12-45	10 mts	Crucible in
1-15	30 "	Lead scum in 44 3/8
5.40	4 hrs. 25	Out, but not done.
10.07		In again next day
5 30	7 hrs. 23	Out 11 hrs 48 mts.

For distilling I used 3 hods charcoal, and five hods of coal, respectively 18 and 120 lbs. From this I got 68 7/8 lbs of lead ready for cupel, and 4 3/4 lbs. of dirt and scum most of which was no good, making the amount driven off ten 6/8 pounds. This process depends on the action of the charcoal, and the low point at which zinc volatilizes.

84 3/8
73 5/8
10 6/8

This finished the rich lead, the poor lead ought to have the zinc driven off, but I did not have time to do it, so it had to be given up. In order to cupel my lead, I had to make a cupel.
First I ground up some calcined bone

in order to get some boneash. I took then 90 parts of boneash, 5 parts of clay and 5 parts of litharge, and mixed them when dry, and ^them^ moisteyed ^them^. To make one cupel, I used 50 lbs. bone ash 2.7 lbs. clay and 2.7 lbs. litharge. The holders were nearly ft. square, only they had rounded corners, and were deep. After pounding the mixture into the cupel, and drying for a few days, I broke the arrangement getting it into place. The second cupel made at the same time as the first, was found to be too dry and crumbling to cut, so the mixture had five parts more clay added to it, and was pounded into the moulds, when damper than at first.

After drying this two days on a steam table, it was cut out, and put into place in the cupel furnace.

This one was very hard and good, and held its form well. The lowest point in the basin, scooped out in the cupel, was about 1½ inch below the top of the ring holding it. The fire in the furnace was made at quarter past six, and the cupel was heated from then until , when lbs of lead were put in an a blast

started, blowing over the surface. At
the lithage was seen to be eating
into the cupel, and the operation was
stopped. The air from the blast oxid-
izes the lead, forming lithage, which floats
on top of the lead, or is absorbed by
the cupel. That floating, is allowed
to run off in a channel so as to allow
a fresh surface of lead to the blast. In that
way the lead is all got rid of, and
the silver left on the cupel.

When I smashed this cupel, I found
that the lithage had combined with the
mixture, forming a slag with the clay,
and had thus eaten about 3/4 of an inch
into the cupel. The lead was crystalline
in structure when broken.

So I have not been able
to finish this operation for lack of
time, all except this last thing being
done though, it is possible that I get time
to do it, after this is handed in.

Product.	Weight of product.	Per cent of Lead.	amount of Lead.	Loss of lead.	Per cent of Silver.	Ounces of silver in Product.	Loss of Silver in Ounces.
Galena.	1100 lbs.	61.37	675.07		.095	15.24	
Roasted ore.	1056. "	58.20	614.59	60.48	.096	14.76	.48
Agglomerated.	996.25	60.68	604.52	10.07	.086	12.48	2.28
Crude lead.	437.87	—	437.87	166.65	—		
Refined lead.	429.75	98.95	425.24	12.63	.197	12.33	.15
Cupel "	68.87	99.03	68.2	2.54	.97	9.73	2.6
Unrefined Pig.	356.50	99.44	354.5 422.7		.001	.05	

675.07
422.7
252.37

The total loss of lead is 252.37 lbs. i.e. 37.38 o/o of the lead has been lost in the whole process.

15.24
9.73
5.51

The total loss of silver is 5.51 oz a loss of 36.15 o/o. The loss of silver in the zincing 2.55 oz, is not good, and is due to the sample I took, the loss of that silver was very likely in the blast furnace. The sample was taken from the tops of the cakes from the refining, and the top is richer than the bottom of the mould in silver, so the refined lead shows a richer product than it ought. The slag and residue did not give

any silver when I assayed a few grammes.
The per cent of lead in the unrefined pig is only approximate, it has about two pounds of zinc in it.

Coal used in:	Charcoal.	Coke	Anthracite	Bituminous	Pounds Heated	Amount of coal per pound of substance heated.
Roasting.				654.87½	1100.	.59 lbs
Agglomerating				546.62½	1056	.51 "
1st Furnace run	30.	308.			996.25	.77 "
2nd " "	120.	309.				
Refining Furnace #1.	6.		120.		631.	.46 "
" " #2.	6.		168.			
Skimmings	6.		72.		73 3/8	1.06 "
Slag Sample	6.		24.		13.	2.30 "
Smelting Matte	6.		96.		74.	1.31 "
Zincing	18.	42.		120.00	429.75	.41 "
Distilling	18.		120		84.75	1.37 "
Total	216.	659.	600.	1321.5		

Therefore 2796.5 pounds of fuel were used on the ore before cupelling.

1321.5
600.
659.
216
2796.5