12.

Mr. Shockley,

Class '75,

Merrimac ore.

Mr. Oxnard,

Class '75,

Burleigh tunnel ore.

Omit

Working of an ore from Newbury.

Before proceeding to describe the actual working of the ore it would seem advisable to devote a short space to a few facts in regard to the new mining region that has been so lately discovered.

Newburyport is a seaside town, or rather it is situate at the mouth of the Merrimac river, of some 13000 inhabitants it is about forty miles from Boston and is easily reached by either the Boston and Maine or by the Eastern R.R.

The mines of Newburyport are however not in Newburyport but are in Newbury which is an adjoining town.

The story of the discovery and the subsequent history of the deposits I relate as I have heard it.

There is of course a legend current that the deposits were worked in the past. The story here is that they were worked by the revolutionary heroes, who obtained lead for their bullets from this source.

There seems to be no foundation for this belief. In the year 1868 a Byfield man, by name Rogers, while walking over Highfield Pasture saw something glisten, he picked it up and as it seemed to be heavy he thought it likely to contain metal; following out this idea he took it to a Mr Albert Adams a farmer of the neighborhood. Mr Adams shared his suspicions and that he might be able to judge for himself he took up the study of Geology and Mineralogy.

In a short time he not only became convinced that the substance was an ore but that it was an ore of lead. Acting on this he bought the land where it was found of its owner-Mr Jacques for $350. He then began searching on it and in a short time dug up about six tons of float-

Omit

ore consisting mainly of galena and quartz and also containing gray copper, (tetrahedrite); pyrite, chalcopyrite and siderite.

Of course his discovery was soon known and in Aug. 1874 the mine was bonded to Dr Kelley and Mr Chipman for $100,000. and since then the land has passed into their hands. Dr Kelley, Mr Shaw and Mr Boynton own a lot in partnership this lot adjoins the Chipman Lode. Since these purchases and sales there have been many rumors as to transactions having taken place.

I have been told by one of the proprietors of the Chipman Lode that two thirds of it have been sold at the rate of a million dollars. i.e. for $666,666, and I have heard many rumors to the same effect but this price was not generally believed in at Newburyport. All that has been said in

Omit

regard to discovery of lands has reference to the Chipman Lode.

The Boynton lode is next to the Chipman on the same line and is undoubtedly the same lode.

Mr Boynton's claim is 2431½' square feet
he has a small house erected over the mouth of his shaft and at the present time is engaged in taking out ore and sinking. At the Chipman Lode there is also a small engine house but at present they are only straightening the shaft. The following are a few statistics I was enabled to obtain while there.

	Chipman Lode.	Boynton.
Depth of shaft.	75'	75'
Men employed.	10.	10.
Ore on hand,	700T.?	300 T.
Size of shaft.	8'×8'	4'×10'.

In the Boynton shaft they have run off a level at a depth of 30' along the vein to the line of the Chipman Lode, 31½' distant, they are at present

omit

stoping out this level and continue to sink the shaft to a depth of 100' when they will drift off in both directions along the vein.

The Kempton party are prospecting with a diamond drill at a distance of half a mile from the Chipman lode. They have found no vein as yet. To give an idea of the ideas of some persons in regard to these deposits I will say that they are often asked if they do not take out drill cores of solid silver!

There are a great many people prospecting in this region but I am informed by a gentleman who is largely interested in mining lands, that no well defined lode has been found save the Chipman.

I visited while there a lode near Rowley some five miles from omit.

Newburyport, in a southerly direction. At this place, which has been named Bartletts Lode, I saw a great amount of tetrahedrite and a wall rock the same as that of the Chipman Lode.

Other deposits have been found at West Newbury, Byfield, Amesbury, Salisbury and others. There is some talk of erecting a smelting works and thus avoiding the expense of shipping to New York as is done at present. The geology of Newbury is a subject I have not been able to get much information on.

The country rock is a gneiss with an outcrop 70°-80° east of north and a dip of 30° to the northwest,

The vein has a strike of north 72° east. An examination has been made of the wall rock by Miss Swallow and to her kindness I owe the following facts.

Omit

South Wall.

The south wall is a pale yellowish green or greenish white rock showing white quartz grains, minute pentagonal pyrites and greenish grains of some mineral under the microscope Sp. G. = 2.766 Hardness 2.5, slightly fusible and but little affected by acids.
An analysis gave.

No good means was	SiO_2.	66.53
found for separating	Al_2O_3.	25.09
the quartz grains from the	Na_2O	.39
combined silica so it is	K_2O.	4.67
an assumption to say	H_2O.	2.64
that the rock is composed		99.32

of quartz and Agalmatolite.

But making this assumption we have

Over.

Omit

Quartz	22.56			
SiO_2	43.97	Agalmatolite.	SiO_2	56.93
Al_2O_3	25.09		Al_2O_3	33.00
Na_2O	.39		Na_2O	0.51
K_2O	4.67		K_2O	6.11
H_2O	2.64		H_2O	3.45
	99.32			100.00

FeO. CaO. MgO. Trace.

This agrees perfectly physically and chemically with Dana's Agalmatolite.

North Wall.

The north wall is a very compact fine grained grayish green rock fusible to a black slag, Its spec. grav. is 2.71 hardness not determined.

Under the microscope there seemed to be three different minerals viz. I. A light yellowish green mineral resembling the south wall.

II. An olive green transparent mineral somewhat bladed or cleavable in one direction.

III. Very small opaque fragments appearing black.

Omit

The mineral II alone seem to dissolve in HCl. HNO_3 or H_2SO_4 The following are the various analyses made.

	Soluble Portion.	Insoluble Portion.	Rock.
SiO_2	26.25	66.44	15.01
Al_2O_3	39.03	26.73	22.32
CaO	4.51	6.10	2.58
$K_2O.Na_2O$	1.50	.80	.85
FeO	13.19		7.54
CO_2	5.60		3.31
$S.$	.25		.14
$H_2O.$	9.67		5.53
	99.90	100.07	Residue. 42.59
			99.37

What these minerals are is difficult to say, though the insoluble portion may be quartz and agalmatolite from which the alkali and part of the alumina have been removed by acids. The soluble portion has evidently quite an amount of siderite.

Print

Rom The ore that was worked was float ore from the Chipman Lode. The ore presented the usual appearance of a float ore being covered with oxide of iron & various decomposed minerals

An examination was made of the mineral species and the following were detected.

Galenite, quartz, siderite, limonite, hematite, pyrite, chalcopyrite, tetrahedrite, malachite, azurite, zinc blende, crocoisite, kaolin. The principal mineral constituents are, galena, quartz and pyrite.

An analysis of the tetrahedrite was made by Miss Swallow and to her I owe also the remarks on the siderite. There are three varieties of siderite in the ore. viz. I A dark almost black partially decomposed variety containing traces of CaO, MgO, MnO and PbO, also CuO.

Print

omit

II. A yellowish gray carbonate very similar in composition to I, but without the lead or copper.
III. A nearly white variety with a vitreous lustre resembling felspar, on heating becomes fused on the edges, gives little water, decrepitates turns black and becomes magnetic. An analysis of this variety gave.

Quartz impurity	.72
$FeCO_3$	46.84
$MnCO_3$	6.49
$MgCO_3$	8.83
$CaCO_3$	34.32
H_2O (undetermined)	1.80
	100.00

This differs considerably from any mineral described by Dana, the nearest approach is Menheim's Altenberg mineral viz.

$FeCO_3$	64.04
$MnCO_3$	16.56
$CaCO_3$	20.12

This corresponds to a formula; $8FeCO_3 + 2MnCO_3 + 3CaCO_3$. The Newburyport is; $8FeCO_3 + MnCO_3 + 6CaCO_3 + 2MgCO_3$.

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A partial analysis of the ore gave:—

		Calculated minerals	No 1.	No. 2
Silica	16.18	Quartz.	13.61	13.61
Lead.	45.39	Sphalerite.	1.21	1.05
Copper	1.33	Chalcopyrite.	3.84	1.65
Zinc.	.81	Pyrite.	14.81	13.31
Iron.	8.34	Kaolin.	5.17	5.17
Alumina,	2.23	Siderite.	.53	.53
Sulphur	16.70	Galenite.	52.68	52.55
Carbonic acid.	.20	Limonite.	5.10	5.72
Ferric oxide	5.10		96.95	
		Tetrahedrite		2.15
				97.74
Silver	.20			
Gold.	.002			
Undetermined (As etc.)	3.52			
	100.00			

No 1 is calculated directly from the ore analysis. Now if we assume the galena to have .19% Ag. (and this assumption is warranted by the assays) we shall have the galena in the ore (52.41%) taking .10% of the silver and, if the

Print

remainder be assumed to constitute the 4.65 o/o of the tetrahedrite, (which the assay shows to be the case) we shall have the basis for the results given in the second column.

The value of a ton of this in currency is (gold 1.15):– 90.7 lbs. lead $ 54.46

58.32 oz. silver $ 86.61

.58 oz gold $ 13.89

Total $ 154.96

An examination was made of the various qualities of galena to see how the silver was distributed. The results of the assays are:–

		o/o	oz.	$
Coarse galena	from 30' below surface	.35	102	131.65
Medium "	from ore	.09	29.16	37.61
Fine "	" "	.22	65.90	85.01
Tetrahedrite	" "	4.65	1357.19	1750.77
"	" Chipman Shaft.	2.30	670.68	865.17

Other tetrahedrites have varied from $ 500 – 1300.

omit

Working of Ore.

Weights = Kilogrammes.

Ore. 816.4
- 324.97 — crushed rolled and sampled
  - loss 1.36
  - roasted. (317.15) — 251.61 — Smelted in Blast Furnace. with addition of fluxes
    - fume etc = 65.54
    - Cake lead. 82.54
    - Foul Slag 90.71
    - Slag 220.42
  - sample. 6.46
- 501.43 boxed up.

Foul Slag. 90.71 Crushed and Sifted.
- Sifted portion. 58.95 Ran through Spitz Lütte
  - Lead.
  - Residue.
- 31.76 Unsifted Portion. Jigged
  - Lead.
  - Residue.

Lead (from both) — smelted
- Lead. 8.10 Put with cake lead.
- slag and refuse.

Cake Lead 90.71
- Impure Lead. 74.84
  - Refined Lead. — Zinced — sweated
    - Lead. 54.43 — Refined.
      - Pure Lead. 48.98
      - Skimmings Smelted.
        - Lead 4.98
        - Refuse.
    - Argentiferous Zinc.
  - Dross
    - Cupriferous Lead. 12.70 A
    - Refuse.
- Skimmings. — Smelted.
  - Cupriferous Lead. 7.7 B.
  - Refuse.

Cake Lead. cont. 48.98 — zinced and sweated,
- Lead. — Refined.
  - Lead. 30.39
  - Skimmings
    - Lead. 14.97 — Refined
      - Lead. 12.15
      - Skimmings.
    - Residue.
- Argentiferous Zinc.

Treatment Cupriferous Lead. A + B. 20.4 smelted + FeS. + PbS.
- Lead — Zinced and Refined.
  - Lead 15.25.
  - Argentiferous Zinc.
- Matte rich in Copper.

All the Argentiferous Zinc Dezinced = 12.47
- Cupelled 1.7
  - 11 grms. Ag.
  - PbO etc in Cupel.
- Cupelled 10.77
  - 87.86 grms Ag.
  - Pb etc ran through the cupel. — Rich Lead having.

Print

Details of Working Ore.

Weights = kilogrammes.

Lg/ The ore was in large lumps weighing from 10 to 30 k. The first operation was to break up these lumps and pick out the ore to be worked. At the same time pieces were picked out and examined for the minerals.

The weight of the whole lot was 816.41 k, the amount picked out 324.97 k, this being as near as possible a fair sample of the whole. The ore was passed through a small Blake's crushers and then through rollers; these operations were to crush the ore so it might be fit for roasting. The ore was then sampled, passed through a 1/20" sieve; the portion remaining on the sieve was rerolled until it was judged to be about 1/50. & The loss in crushing was .34 kilos., the sample taken out weighed 6.40 kilos., so that the ore now weighed only 311.13; the loss of 7.64 kilos. was due to sifting, rolling and to the fact that some

Long Print

of the ore was overlooked and remained behind. The ore was divided into eight charges, each of 39. kilos; it was roasted in two reverbratory furnaces, each charge being roasted four hours. A sample was taken at the end of two hours, and also when the charge was drawn, these samples were analyzed for sulphur; the results of these analyses will be found farther on.

After the ore was roasted it was put in the larger of the two furnaces and agglomerated. A sample was taken of this agglomerated ore and analyzed so as to furnish data for calculation of slag, etc. The object of agglomerating the ore, was to make it strong enough to stand the pressure of the fuel and fluxes.

omit

Tables showing the details of roasting and agglomerating will be found on the next page.

Omit

Insert (A)

Roasting of Ore.

No of Charge	Wt.	Time of Charging.	First Sample At.	Drew At	Coal.	Remarks.
1.	39.	8.30 P.M.	10.30. P.M.	12.45	~~////~~ /	Slight fusion at 10.15
2.	39.	1.15 A.M.	3.30 A.M.	6.15	////	
3.	39.	7.10	9.25	11.10	//	
4.	39.	11.20	1.20	3.20	/	
5.	39.	8.30 P.M.	10.30	12.35	~~////~~ /	
6.	39.	1.10.	4.	6	///	Slight fusion 4 Am.
7.	39.	6.50	9.15	10.50	//	
8.	39.	11.15	1.15	3.15	/	

The first four charges are in the large furnace the last four in the small.

Table showing o/o S removed by roasting.

No. of Charge.	1.	2.	3.	4.	5.	6.	7.	8.	average
S after 2 hours	7.36	7.85	6.60	7.09	not determ.	10.13	5.70	9.91	7.8
S after four h.	4.18	6.23	4.90	4.97	not determined	4.85	4.36	5.77	5.04

The table on the next page shows the details of the agglomeration.

No of Charge.	Time of Charging	Time of Drawing	Fuel.
1.	4.25	5.10	~~IIII~~
2.	5.15	6.00	
3.	6.35	7.15	
4	7.20	8.00	

Print Rena

Print.

The ore after being agglomerated weighed 25.6 kilos, a sample was taken and a partial analysis made, so that there might be data enough to calculate the amount of fluxes to be used. After these calculations were made, the ore was run in the blast furnace. The details as to the run will be found in the tables on pages 22 to 30 inclusive. A slag from the Revere Copper Works was used to start the furnace with. It was found after running some time that the proportion of flux and ore was too great for the fuel, so the charge was altered by taking only 3/4 the amount of ore and flux to the fuel after this the furnace worked well.

Lenny Print

¶ The calculations were to have a basic slag; the fluxes used were puddle cinder and limestone; the exact composition of the puddle cinder was not known at the time of the calculation but it was judged that it contained about 20% SiO_2 to 80% FeO and acting on this supposition the following charge was calculated:— ~~This gave a charge as follows~~

40 lbs Ore
25 lbs Puddle Cinder
3 lbs limestone.
3 shovels coke.

This charge was afterwards changed to

30 lbs Ore
18 3/4 " Puddle Cinder
2 1/2 " limestone
3 shovels Coke.

¶ This change was made in order to bring the layers of fuel nearer together.

The analysis of the puddle cinder afterwards made and of the ~~and~~ roasted agglomerated ore are below.

Puddle Cinder.		Ore.	
FeO	70.62	FeO	17.11
SiO_2	15.43	SiO_2	18.47
Al_2O_3	9.42	Al_2O_3	2.57
		S.	3.27

Now this gives in each charge the

Lucy

following amounts.

Compounds	SiO_2	Al_2O_3	CaO	FeO	S.
Ore.	7.39	1.03	0	6.84	1.31
Puddle Cinder	3.86	2.45	0	17.65	0.
Limestone.	0.	0	1.68	0.	0.

Lucy

After allowing sufficient iron to ~~Removing~~ the sulphur as FeS ~~and~~ the calcu lation of the per cents of the components in the slag ~~we have the following results~~ is as follows.

Compounds.	SiO_2	Al_2O_3	CaO	FeO	Pb.
Calculated	29.56	9.23	4.41	56.77	0.
Calculated from Analysis	32.92	9.39	5.02	52.66	0.
Analyzed.	30.54	8.71	4.66	48.86	2.44

omit & insert (B)

omit

The line marked 'Calculated' contains the per cents of the elements composing the calculated slag, the line headed 'Analyzed' contains the per cents found by analysis, and the line head 'Calculated from Analysis' is calculated from the analysis as if the

Omit

(slag only contained Al_2O_3, SiO_2, CaO, FeO.)

Leave Print The discrepancy between the calculated slag and the slag actually obtained is due to two causes; viz the dissolving of silica from the bricks and the charging of the Revere Slag at sundry times during the run. The analysis of the Revere Slag is:-

FeO	41.86
CaO	9.96
SiO_2	37.00
Al_2O_3	6.21

omit

Blast Furnace

Charging Door.

Time of Charging	Interval	Charge	Depth of material After Charging	Remarks
5.40 Mar 30 March 31st		3 hods charcoal 3 " coke shaving etc.		Blast a 8.56. AM.
9-5 AM		4 sh. coke.		Charge No 1. {2 sh. coke 4 Revere slag.
9-10	5.	4 sh. coke.		
9.25	15	Charge No 1.	1'	Charge No 2. 3 coke. 6 slag
9.30.	5	Ch. No 1.	1'-8"	
9.37	7	Ch. No 1.	1'-9"	Charge No. 3 {40 lbs. = ore. 25 lbs = puddle cinder 3 lbs. = lime. 3 shovels coke.
9.40	3.	Ch. No 1.	2'-3"	
9.52	12.	Ch. No 2	3'-4"	
9.57	5.	Ch. No 2.	3'-5"	Charge No 4. 2 shovels coke.
10.10	13.	" "	3'-7"	
10.40	10.	" 3	3'	
10.52	12.	" 4.	3'	
11.0	8.	" "	3'-3"	
11.7	7.	One small shovel sulphate soda.		
11.10	3.	Charge No 4.	3'-5"	
11.50	10.	4 shovels slag.	1'-8"	
11.50	0.		1'-8"	
11.56	6.	Charge No 5	2'-2"	
12.25	30.	Charge No 6.	2'-5"	

omit

Blast Furnace.

Charging door.

Time of	Interval	Charge	Depth of material in the furnace. After charging.	Remarks.
12.35	15	Charge No 6.	2'-5"	Charge No 5:
12.50	10	" "	2-8	3 shovels of coke
1.	10	" "	2-10	4 " " slag.
1.10	10	" "	2-10	Charge No 6.
1.20	30	" "	3-3	30 lbs Ore.
1.50	15	" "	3-3	18¾ lbs Cinder
2.05	15	" "		2¼ lbs limestone.
2.20	17	" "		3 Shovels coke.
2.37	15	" "		Last charge of ore
2.52	23	" "		25 lbs at 6.05
3.15	21	" "	3'-3"	
3.36	25	" "	3'-5"	
4.01	25	" "	3'-5½"	
4.26	34	" "	3'-5"	
5.	23	" "	3'-8"	
5.23	12	" "	3'-7"	
5.35	15	One hod foul slag.	3'-6"	
5.50	15	One hod foul slag.		
6.5	30	One hod foul slag.	4'-0"	
6.35			3'-0"	

omit

Blast Furnace.
Charging door.

Time of Charging	Interval	Charge	Depth of scale read in Furnace	Remarks.
6.43	8.		2'-3"	
6.45	2.	2 shovels coke.		
6.52	7.		2'-9"	
7-2	10.	5 lbs Na_2SO_4	2'-8"	
7.10	8.	"	1'-11"	
7.18	8.	5 lbs Na_2SO_4	1'-11"	
7.30.	12.		1'-9"	

omit

Blast Furnace. Run.

Tap Hole Times.	Interval	Plugged	Tapped	No. of Kettle	Remarks.
8.56					Blast On.
9.37					Slag appeared.
9.55					Flame came through breast.
9.57					Small amount matte.
11.37	5 minutes	plugged.			
11.42	1	1	tapped	12.	Small run.
11.43	5	plugged			
11.48	1½		tapped	13	Small run.
11.49½	5.	plugged			
11.54½	½		tapped	14	
11.55	5	plugged.			
12.00	1.		tapped.	15	lead appeared at first.
12.01	5.	plugged			
12.06	1.		tapped	16.	Small run.
12.07	5.	plugged			
12.12	1.		tapped	17.	Small run.
12.13	5.	plugged			
12.18	1½		tapped	18	Good run, little lead.
12.19½	5.	plugged			
12.24½			tapped.	19.	Very poor at first and then very fluid giving much lead

omit

Blast Furnace. Cont.

Tap Hole. Times.	Interval.	Plugged	Tapped	No. of Kettle	Remarks.
	3½				
12.28		plugged			
	5.				
12.33			tapped	20.	Good.
	2.				
12.35		plugged			
	5.				
12.40			tapped.	21.	Small.
	2.				
12.42		plugged			
	5.				
12.47			tapped.	22.	Small.
	2.				
12.49.		plugged.			
	6.				
12.55			tapped.	23.	fair.
	1.				
12.56		plugged.			
	6.				
1.02			tapped.	24.	very liquid.
	1.				
1.03		plugged.			
	5.				
1.08			tapped.	25.	very good.
	½				
1.08½		plugged.			
	8½				
1.17.			tapped.	26.	very good.
	1.				
1.18.		plugged			
	8.				
1.26.			tapped	27.	very lively.
	½				
1.26½		plugged			
	8½				
1.35			tapped	28.	very good.
	½				
1.35½		plugged			
	8				
1.43½			tapped.	29.	Good; as usual.

omit

Blast Furnace. Run.

Tap Hole Time.	Interval.	Plugged	Tapped.	No. of Kettle	Remarks
	1/2				
1.44	5	plugged.			Good.
1.49			tapped	30.	
	1/2				
1.49 1/2		plugged.			All these runs
	8 1/2				
1.58			tapped.	31.	were good as
	1/2				
1.58 1/2		plugged.			could be wished.
	8 1/2				
2.07			tapped	32.	
	1/2				
2.07 1/2		plugged.			
	7 1/2				
2.15			tapped	33.	
	1/4				
2.15 1/4		plugged.			
	9 1/4				
2.24 1/2			tapped.	34.	
	1.				
2.25 1/2		plugged.			
	7 3/4				
2.33 1/4			tapped	35	
	1/2				
2.33 3/4		plugged.			
	7 3/4				
2.41 1/2			tapped	36.	
	1/2				
2.42		plugged			
	8.				
2.50			tapped	37.	
	1/2				
2.50 1/2		plugged.			
	8.				
2.58 1/2			tapped.	38	
	1/2				
2.59	8	plugged.		39.	
3.07			tapped.	39.	

omit

Blast Furnace.
Tap Hole.

Time	Interval.	Plugged	Tapped.	No. of Kettle.	Remarks.
	3/4				
3.07 3/4	9 1/4	plugged			
3.17	1/2		tapped	40	
3.17 1/2	7 1/4	plugged			Good as usual,
3.24 3/4	3/4		tapped	41.	
3.25 1/4	8.	plugged			
3.33 1/4	1/2		tapped	42.	
3.33 3/4	8.	plugged			
3.41 3/4	3/4		tapped	43.	
3.42 1/4	10.	plugged			
3.52 1/4	3/4		tapped	44.	
3.53	11 3/4	plugged			
4.04 3/4	1/2		tapped.	45.	
4.04 1/4.	9 1/4	plugged			
4.14	1/2		tapped	46.	
4.14 1/2.	13 3/4	plugged			
4.28 1/4	3/4		tapped	47.	
4.29	12	plugged			
4.41	1/2		tapped	48	
4.41 1/2	7 3/4	plugged			
4.49 1/4	1/4		tapped	49.	
4.49 1/2		plugged.			

omit

Blast Furnace.
TAP HOLE.

Time.	Interval.	Plugged.	Tapped.	No. of Kettle.	Remarks.
4.57½	8		tapped	50	
4.58	½	plugged			
5.06	8		tapped	51.	
5.06½	½	plugged			Good.
5.13½	7.		tapped	52.	
5.14½	1.	plugged			
5.23½	9.		tapped	53.	
5.24	½	plugged			
5.33½	9½		tapped.	54	
5.34¾	1¼	plugged			
5.44¾	10.		tapped.	55	Good.
5.45¾	1.	plugged			
5.53¾	8.		tapped	56	
5.54¾	1.	plugged			
6.03¾	7.		tapped	57	
6.05	1¼	plugged			
6.12	7		tapped	58	
6.12¾	¾	plugged			
6.21¼	8½		tapped.	59	
6.22.	¾	plugged.			

omit

BLAST FURNACE.

TAP HOLE.

Time	Interval	Plugged.	Tapped.	No. of Kettles	Remarks.
	8.				
6.30			tapped	60	Good.
	2				
6.32		plugged			
	8				
6.40			tapped	61	do do.
	1				
6.41		plugged			
	7½				
6.48½			tapped	62	do. do.
	1				
6.49½		plugged			
	7				
6.56½			tapped	63	Very good.
	1.				
6.57½		plugged			
	8.				
7.05½			tapped	64	Good: after this all have soda.
	1.				
7.06¾		plugged			
	7½				
7.14¼			tapped	65	Good
	¾				
7.15		plugged			
	9½				
7.24½			tapped	66	Good.
	3.				
7.27		plugged			
	9.				
7.36			tapped	67	Good
	2½				
7.43½		plugged			
	5.				
7.38½			tapped.	68	Good Blast shut off and furnace cleaned as much as possible

Print

Lucy

The furnace was started with Revere Slag, and the first twelve kettles consisted of this and were laid aside. The remaining products were slag and lead and a small quantity of iron matte. The lead and slag were separated, and gave 220.42 k. slag, 82.54 kilos lead. There was also 90.71 kilos foul slag i.e., slag containing too much metallic lead to be thrown away, consisting of nozzles, furnace bottom and bits of the total slag with adhering lead. This slag was crushed, and sifted; the part remaining on the sieve, jigged; the portion going through the jig treated on the Spitz Lütte. The products from these operations were smelted, and 8.16 kilos of lead obtained; this was put with the cake lead, increasing its weight to 90.70 kilos. From each cake as it was broken up a piece of pure slag was picked out; these bits were crushed and sampled

Long and an analysis made of the sample. It is to this analysis that I have reference when speaking of slag, page 20. An analysis was also made of the total slag - it gave 2.90% Pb, i.e., 6.39 kilos. Now the amount of lead that should have been obtained is 143.95 the amount that was obtained is 90.71 k.; this with the amount in the slag = 97.10 k. or 67.45%, thus showing a loss in the operations of smelting and roasting of 32.55%.

I am not able to state how much of this loss is due to roasting nor how much is due to smelting. The loss is probably caused by fumes, and it must be caused by volatilization because all the products have been assayed for lead and none has been found, save as stated.

The cake lead was now refined. It was attempted to do this in a crucible by skimming, but this was found

Print

Lowry

impracticable on account of the great impurity of the lead. The lead was, for the foregoing reasons, ~~ran~~ run into ingots and sweated.

The resultant lead, 62.5 kilos, was quite pure and was all ready to be zinced. This lead was assayed for silver and gave .448% or 280.31 grms Ag.

There was formed during this process of refining quite an amount of skimmings. There was also quite an amount of dross left on the sweating furnace. Both of these compounds contained a large amount of copper. They were smelted and from the smelting was obtained, 7.7 kilos. lead containing .341% Ag or 26.26 grammes; and 12.70 kilos. lead with .332% Ag or 42.16 grms. The 7.7 kilos was from the skimmings, the 12.70k from the dross. The loss of lead by this operation was 7.74 kilos; the loss of silver cannot be told.

Print.

Leary

The lead obtained from the smelting of the skimmings and dross was very rich in copper; now it was desired to remove the silver from this lead by zincing. Copper is a detriment to this process and therefore it was advisable to remove it (i.e. the copper). The method devised for doing this depends on the greater affinity of sulphur for copper than for lead. If galena and copper are heated in a reducing atmosphere lead is set free and copper matte (sulphide of copper) is formed. Taking advantage of this fact the copper was removed from the lead. The maximum amount of copper that could be present being known, enough galena was added to change this to sulphide, also an sufficient amount of phyrrhotine magnetic pyrites was added ~~enough~~ to make a matte of the formula ($Cu_2S\ FeS$) and the fusions were made. The results were very good; a matte rich in copper and

Omit all after 3d line on this p. Close with (C).

Leavy a lead very free from copper were obtained. Weight of lead. 15.25. kilos.

.. .. Matte. 5.90 kilos.

End here

This lead was zinced and refined.

The weight of the refined lead is 12.24 it contains .02 % Ag or 2.45 grammes.

The refined lead 62.5 kilos. was now zinced by the addition of 3%. This operation was performed by melting the lead in a crucible, raising it to a red heat and then putting in the zinc. The zinc is well mixed and the contents of the crucible dipped out and ran in to ingots. These ingots were then sweated and the argentiferous zinc remained behind.

The weight of the desilverized lead was after purifying 48.98 k., the 13.52 kilos being in the skimmings and in the argentiferous zinc; this lead assayed .043 o/o Ag; this being too rich it was again zinced and sweated as before.

The refining of this lead took a very long time and its weight was reduced by skimming to 30.39 kilos. This lead assayed .0065 o/o Ag. and was put one side as being pure enough to leave. This lead contains 1.97 grammes Ag.

The skimmings of the first refined, i.e the refining of the 62.5 kilos. lead were now smelted and gave; 4.99 kilos lead assaying .038 and containing 1.90 grms Ag.

The skimmings from the refining of the 48.98 kilos were smelted and gave 14.97 kilos lead this when refined yielded 12.15 k. having .014 Ag and containing 1.70 grms Ag.

A fusion was made of the various residues obtained during the working of the lead and the result was 5.78 kilos of a very impure lead containing .25 o/o Ag. or 14.45 grammes.

The argentiferous zinc was now de-zinced this was readily accomplished, by placing it in a crucible, filling the crucible with lumps of charcoal and heating at a low red heat, all the zinc is driven off and only argentiferous lead remains behind. The weight of this argentiferous lead was 12.47 k. it assayed 2.31 o/o thus containing 288. grms of silver.

The following table shows the amounts of lead and silver before cupellation.

Wt of Lead	% of Ag.	Wt of Ag. gramme.
30.39	.0065	1.97
4.99	.038	1.90
14.97	.014	1.70
12.24	.02	2.45
5.78	.25	14.45
12.47 Ag Pb.	2.31	288.0
80.84		310.47

The lead is 89.12 o/o of the cake lead thus showing a loss of 10.88 o/o or 9.87 kilos. The silver is 50.52 o/o of the silver in the ore, what o/o it is of that in the cake lead, I do not know having no assay of this.

The lead is 56.16 of the lead in the ore thus showing a loss during all the operations of 43.84 o/o

~~Bonas~~

The next process was to cupel the argentif erous lead in order to get the silver.

In this operation a great deal of diffi culty was met with and the final re sult was not as satisfactory as could be wished. The cupels used in this opera tion consisted of bone ash 6.70 litharge .15 fire clay .07 kilos. that is 100 pts. bone ash 2 parts litharge and 1 part fire clay.

The record of the first cupellation is as follows.

Fire at 9. a.m.
Blast at 12. m.
Cupel cracked at 12.10.
Charged at 12.20
Wt charge 1.58 kilos.
Button blicked 3.40
Wt button 11.24 grms.
Coal 1111.

The crack in the cupel was at first

thought to be harmless, and the intention was to cupel all the lead but this idea was given up and only the amount stated was run through. The amount of silver that should have been obtained was 36.50 grms. thus showing a loss of 25.26 grammes this loss was caused by the cupel becoming too hot; part of the silver was absorbed by the cupel and the remainder was volatilized.

The remainder of argentiferous lead was treated on a second cupel of the same composition as the first.

The record of this cupellation is as follows.

Coal = 7 hods

Fire lit 7.45. Blast at 10. A.M.

The weight of	1st Charge	at	10.22
lead cupelled = 10.89 k.	2nd "	at	10.45
Each charge was	3d "	at	11.30
the same as near-	4th "	at	12.10
as could be without	Button blicked.		1.10.
weighing.	wt button = 87.86 grammes.		

The amount of silver that should have been obtained from the cupellation is 251.56 grams. The reason for so great a difference in these amounts may be found in the fact that a large portion the silver-lead ran through the bottom of the cupel. This was caused by the too great heat under the cupel and also by the cupel cracking. The lead that came through was run into ingots and part of it was re-cupelled. The total amount of lead was 1987 grammes, 287 grammes of this were cupelled yielding 24.40 grms of silver The remaining 1700 grammes contains 8.35 % Ag. This is 141.95 grammes of Ag. The total amount of silver obtained from the ore is 11.24 + 87.86 + 24.40 grammes = 123.50 grammes or 19.47 %. The table on the next page gives the amounts of lead and silver on hand.

Lead.				Ag.
3039 kilogram.	containing	.0065 %	or Ag	1.97 grams.
4.99 "	"	.038 %	" Ag	1.90 "
14.97 "	"	.014 %	" Ag	1.70 "
12.24 "	"	.02 %	" Ag	2.45 "
5.78 "	"	.25 %	" Ag	14.45 "
1.70 "	"	8.35 %	" "	141.95 "
70.07 total lead.		Also Ag.		123.50 "
				287.92

The lead obtained is 48.47 % of that in the ore. There is also a quantity in the cupel bottoms that can be easily recovered. The silver accounted for is 45.39 % of the silver in the ore. Assuming that in the operations of roasting and smelting the amount of silver lost was the same as the lead, viz 32.55 % we should have for the loss in these operations 206.46 grammes. This leaves 139.92 grammes, or 22.06 % unaccounted for. Probably a great portion of this was carried off by the antimony in roasting.

The value of the products obtained are.
lead; 70.07 kilos at 6¢ @ lb. $9.26.
silver; 287.92 grammes at 4.13¢ @ gramme
$11.94 (gold.) or $13.73 (currency) giving the
total value of the lead and silver
$22.99 The value of the lead and silver
in the original ore $49.37; and thus by
a simple calculation it is seen that
the value of the lead and silver obtained;
is 46.56% of what was started with.

FINIS.

Report on Burleigh Tunnel Ore (Colorado.)
By
B. A. Oxnard.

The ore which was given me to work up, is one which comes from one of the veins discovered in running the Burleigh tunnel through the mountain, near Georgetown, Colorado. This enterprise started by Eastern capitalists was under the direction of Mr. Charles Burleigh, the inventor of the steam drill of the same name. These drills were exclusively used in the boring of the tunnel. At Georgetown were worked a number of rich silver mines in this very mountain and the tunnel was pierced with the double intention of 1st. Discovering, if possible new lodes. (As before stated it was from one of this latter kind of lodes, the ore under consideration came.)

2nd. Ventilating, draining and otherwise assisting in the working of the then existing mines.

The sample given me was supposed to be a fair one, indicating the relation between the mineral part & the gangue in the ore also showing the average richness of the mineral part. It came to me in lumps and powder. The examination was begun by making a search for the minerals contained in it. The greater part was galena and zinc blende; in noticeable quantities iron pyrites and some copper pyrites. The gangue was mostly siliceous with little specks of mica here and there, and some feldspar and clay. There was also observed a little decomposed siderite, but not enough to give any trace of CO_2 in the analysis of the ore.

The work assigned to me was the extraction of the lead and silver from the galena and that of the silver from the zinc blende. The blende for this did not have to be reduced to the metallic state. The w't of the ore sample was 214 lbs. 12 oz. I went all over this lot, breaking it into small pieces and putting aside a specimen of all the different minerals, found, for blowpipe analysis. Their names have been given above. Another object of the picking and breaking up of the ore, was to sort out as much as possible of the galena which was free from blende, as this would save the trouble of separating it by jigging. The ore was thus divided into two lots, one consisting of pure hand-picked galena, the other, of mixed galena, blende, gangue, etc.

These lots by weight* then were:—

Blende, galena, etc portion	176 lbs. 6 oz
Galena (hand picked)	33 " 6 "
Samples of minerals	4 " 4 "
	214 " 0 "

Each portion of the ore was then crushed first in a Blake's crusher and then between rolls, the galena to 1/40 in. the blende-galena to 1/4 in.

After crushing the portions weighed:—

Blende-galena portion	176 lbs. 8 oz
Galena (hand picked)	33 " 2 "
	209 " 10 "

I explain the increase in weight of the galena-blende, by the fact that the galena being crushed first, some of the dust was not removed, but was taken up after with the blende-galena.

After the crushing each part was well mixed and a sample was taken from each, of which a complete analysis was made.

The two samples being analysed first qualitatively and then quantitatively were found to contain:—

Analysis of I blende-galena part.

Lead	38.00%
Zinc	7.84
Iron	9.81
Copper	0.33
Sulphur	21.22
Silver	0.06
Gangue	16.43
Water	0.75
Undetermined	5.56
Total	100.00

Analysis of galena part.

Lead	73.60 %
Zinc	0.45
Iron	4.19
Copper	0.39
Sulphur	16.33
Silver	0.128
Gangue	2.62
Water	0.35
Undetermined	1.942
Total	100.00

The ore after sampling weighed:—

Blende-galena portion	171 lbs 0 oz
Sample ditto.	5 " 2 "
Galena portion	31 " 6 "
Sample ditto.	0 " 10 "
	208 " 2 "

Dust lost in blende-galena portion	0 lbs. 6 oz
" " " galena "	1 " 2 "

The galena portion was now all ready to roast, having been picked so as to require no ore dressing. It was therefore put aside for the time. The blende-galena portion had to be dressed so as to separate the galena, blende & gangue from each other. The specific gravity of the two minerals, differing very greatly, that of galena being from 7.25 to 7.70 and that of blende from 3.5 to 4.0, a very good separation could be effected by jigging. As a preliminary step the ore was separated by sieves into 5 sizes, viz:—

1st. size. that which remained on 1/4 in mesh

2nd. " " " passed 1/4 in. but remained on 1/8 in.

3rd. " " " " 1/8 " " " " 1/20 "

4th. " " " " 1/20 " " " " 1/40 "

5th. " " " " 1/40.

The first 3 sizes were put each separately on the jig. In this case the operation was made easier by the iron pyrites in the ore, which being intermediate in sp. gr. between galena and blende, formed a bright thin layer between the other two minerals. It was of great importance that as little blende as possible should be mixed with the galena, as it would be in the way in the future work, but it did not make as much difference, whether the blende was mixed with a little galena or not. The iron pyrites did not harm either part. The course I followed was this: after having skimmed off the gangue, I skimmed off the blende & the layer of pyrites leaving only the galena. Some of the galena was taken with the blende to make sure that no blende remained in the galena. This accounts for the rather high percentage of lead found in the blende part. The ore was put in so as to be about 3 in deep for the larger sizes & about 2 in. for the smaller. I gave about 150 strokes for each portion & the separation appeared good in each case. The jig was a 1/30 in. sieve placed in a large tub of water, and suspended remd to a spring board above. A good deal of ore which had not passed in the dry sifting went through the 1/30 in sieve. This was collected and treated on the shaking table, as was also the 1/40 in. product and that which had passed the 1/40 in. sieve. The principle of separation in the shaking table is to throw the ore up an inclined plane by means of a jerk at regular intervals, while water between the jerks, causes it to go down. By regulating the stream and the slope of the table, the water may be made to carry off the blende and gangue, leaving the galena on the jerking table.

There were 3 parts from the jig, viz:— galena, blende & gangue portions. The gangue portion having been jigged several times to get all the mineral part out, was then put aside and not worked any more. The ore from the shaking table was also divided into 3 parts viz:— a galena portion, a mixture of gangue and blende and a slime which was not worked.

The w'ts from the jig were:—

Gangue portion	20 lbs	7 oz
Blende "	48 "	0 "
Galena "	63 "	6 "
	131 "	13 "

The w'ts from the shaking table were:—

Slime	2 lbs	14 oz
Blende & gangue portion	13 "	0 "
Galena "	29 "	4 "
	45 "	2 "

Samples were taken of the 3 portions from the jigs and 2 portions from the shaking table and they were analysed for lead & zinc and assayed for silver with the following results:—

Galena from jig

Lead	70.36 %
Zinc	0.75
Silver	0.103

Zinc blende from jig

Lead	14.27
Zinc	20.75
Silver	0.035

Gangue from jig

Lead	6.28
Zinc	0.88
Silver	0.013

Galena from shaking table

Lead	62.65
Zinc	1.90
Silver	0.087

Blende & gangue from shaking table

Lead	14.15
Zinc	10.89
Silver	0.035

The galena had now been separated from the rest of the ore and after drying & crushing to 1/40 in. the different parts weighed:—

Galena, hand picked	31 lbs. 6 oz
" from jig	63 " 6 "
" " shaking table	29 " 4 "
Total Galena	124 " 0 "

An analysis of this gave:—

Lead	69.35 %
Silver	0.106

(The percent. in this case were calculated from the aliquot parts)

This galena was roasted in a reverberatory furnace at a heat just below the fusing point of galena. The object of the roasting is to drive off the sulphur & this would be retarded by the caking of the galena. The ore is stirred constantly to expose every part of the ore. In this case I failed to avoid caking by fusion, but the lumps thus formed were broken up as much as possible by the stirrer. The roasting went on for 3½ hours, at the end of which

Lamr

time no more sulphur came off from the ore. A sample was taken at the end of 2 hours roasting, also at the end of the roast and they were analysed for sulphur with the following results:—

% Sulphur after 2 hrs. roast	8.61%
" " " 3½ " "	8.15 "
Before roasting, the hand picked galena contained	16.33 "

The roasted galena was then analysed with the following results:—

Lead	59.28%
Silver	0.11
Wt Roasted Galena	113 lbs. 11 oz

Sup

The great loss of lead was in lead fumes which went up the chimney during the roasting. The theoretical am't of lead to be got from the roasted ore was then 67 lbs. 6 oz and the silver 0.125 lbs.

A flux was calculated for the roasted galena according to the % of sulphur + lead in it, which consisted of 52.4 lbs magnetite

Smy

and 4.9 lbs charcoal dust to 100 lbs of ore.
60 lbs. of ore were mixed with the fluxes in the above proportions; but, one charge of ore of 15 lbs. having been tried it was found by its action on the crucible that there was too much iron for the amount of coal, & consequently 4 lbs. charcoal dust were added to the remaining 45 lbs. The result proved more satisfactory. The iron is put in to reduce the sulphide of lead to lead and to take up the siliceous impurities. The charcoal reduces the oxides of lead and iron that are formed. When the 45 lbs. ore gave out, each charge of 15 lbs. of ore was mixed with 2 lbs charcoal and 2/3 as much ~~ma~~magnetite in proportion as before. The estimate of magnetite was at first too large on account of an error in the estimation of the per cent. of sulphur in the roasted ore which was caused by an impurity in one of the laboratory reagents.
The number of charges fused was 8, out of

Carry which, one charge went through the crucible and about 4 lbs of metallic lead were lost. 7 of the 8 charges had 15 lbs ore and the 8th had 8 lbs. 10 oz. The charges were smelted by being first well mixed with the fluxes and then put in to a black lead crucible and fused at a bright red heat from 2 to 3 hours until the whole mass was perfectly liquid. The charge was then poured into a mould. Great care had to be taken to heat the black lead crucible very gradually, as they are liable to crack if this is not done.

The result of the smelting consisted of the slag, matt and 51 lbs. 6 oz crude lead. If the 4 lbs. lost by poor crucible were added to this it would give:—

Total lead obtained (about.)	55 lbs 6 oz
many Theoretical am't of lead in ore	67 " 6 "
Loss in slag + matt	12 " 0 "

Lewy

This would show that between 17 and 18% of the lead had gone into the slag ~~and~~ matt. and fume

The crude lead was refined by fusing it to drive off the sulphur and by sweating it. This gave 47 lbs. 14 oz of refined lead which was assayed for silver with following result.

% Silver obtained in lead	0.21
Weight silver	0.101 lbs
Loss on theoretical am't	18 to 19%.

Lewy The lead was melted in a cast iron pot and heated to driving. 2 lbs. 6 oz of ~~zinc~~ (5% of the am't of lead) were melted in a black lead crucible and just as oxide of zinc was beginning to be formed and to burn, it was poured into the molten lead and the bath having been thoroughly stirred, the zinced lead was poured into ingots.
The process of extracting silver out of lead by means of zinc is called "Parkes' process" after the name of its inventor.

It depends on the fact that when zinc and silver lead are mixed at a strong red heat nearly all the silver goes with the zinc.

It consists of 5 stages:—

1st. Zincing the lead. 2nd. Sweating off the pure lead from the argentiferous zinc lead, owing to the lower fusing point of pure lead. 3rd. Refining the lead, thus obtained. 4th. Removing the zinc from the argentiferous zinc lead by distillation. 5th. Cupelling the argentiferous lead.

The first stage I have already described. The ingots were then put on the sweating furnace and the pure lead melted from the zinced lead. The lead thus melted was refined by driving off the zinc and it was then assayed for silver giving 0.0038 per cent % Ag. Before refining the weight of the lead was 38 lbs. After refining it was 34 lbs. 5 oz. The loss was due to the escape of lead fumes as well as to the driving off of the zinc. The argentiferous zinc lead was then refined by distilling off the zinc at a bright red heat in a black lead crucible. Charcoal was put on top of the molten liquid to reduce the litharge formed, while the oxide of zinc burned off. The time taken to distil the zinc was 2 hrs. The argentiferous lead was then poured into ingots and weighed 9 lbs 1 oz. A sample of this was taken and assayed 0.815% Ag. The argentiferous lead was then cupelled in a cupel furnace. The furnace was first heated to a bright redness and the ingots of lead were put in one by one and as they oxidized the litharge was poured off.

The ingots were put in 3 hrs 30 min after the fire was started & the button blicked 2 hrs. 23 min. after the lead had been put in. The silver button weighed 22.7782 grs or .0502 lbs instead of the theoretical .0782 lbs which had been put on the cupel

The cupel was made by mixing 15 lbs. of bone ash, 2 oz fire clay and 6 oz litharge

		% Ag	wt Ag in lbs.	loss Ag since last operation	% Pb.	wt Pb.	loss Pb since last operation
124lbs	Galena	0.106	.1314		69.35	86lbs.	
113lbs 11oz	Roasted Galena	0.11	.1252	.0062	59.28	67lbs 6oz	18lbs. 10oz
51lbs. 6oz	Crude lead	—	—	—	100.0	51lbs. 6oz	16lbs. 0oz
4lbs	Loss by accident	—	—	—			
47lbs 14oz	Refined lead	0.21	.1005	.0247	99.8	47lbs 14oz	3lbs. 8oz
9lbs. 1oz	Argentif. lead	0.815	.0782 } .0795	.0210	99.2	9lbs. 1oz } 43lbs 6oz	4lbs. 8oz
34lbs. 5oz	Commercial (refined)	0.0038	.0013		100.0	34lbs. 5oz	
.0502 lbs.	Silver button	100.0	.0502	.0293	0	0	9lbs. 1oz

Lead obtained 34lbs 5oz

Silver obtained .0502 — 86lbs. 0oz Total lead in ore.

.1314 Total silver in ore

		Loss of lead	Loss of silver
Silver in the ore	0.1314 lbs.		
Lead in the ore	86 lbs 0 oz		
Loss from roasting		18 lbs. 10 oz	0.0062 lbs
" " smelting		12 " 0 "	0.0247 "
" by accident		4 " 0 "	
" from refining		3 " 8 "	
" from Zincing Sweating & distilling Loss from refining lead with silver taken out		4 " 8 "	0.0223 "
Loss from cupelling		9 " 1 "	0.0280 "
Lead obtained		34 " 5 "	
Silver obtained	~~Total lead = 86 " 0 "~~		0.0502 "
		~~Total silver = 0.1314~~ "	

See 1st page for end.

The portion of the ore containing the zinc was now treated, this consisted of:—

Zinc blende from jig	48 lbs. 0 oz
" " + gangue from shaking table	13 " 0 "
Total blende	61 " 0 "

This was roasted for 7½ hours at the highest heat which could be attained in a reverberatory furnace and nearly all the sulphur was removed by this means. After the roast the ore weighed 52 lbs. 4 oz. This was now crushed for an hour with 4 lbs. of salt in a Chilian mill. The salt and ore by this process being thoroughly mixed were put again in the reverberatory furnace and a chlorination roasting of 3½ hrs. was given to it, which changed all the silver to chloride. (I will here say that an ore is never put into the reverberatory furnace, until the furnace is heated to the required temperature for the operation.) The wt of the ore was now 60 lbs 4 oz. This was put into an amalgamating tub with iron lined paddles which rotated by steam power. Just enough water to make the whole mass liquid was added and some iron filings were put in. The iron in the tub decomposed the chloride of silver forming chloride of iron and leaving metallic silver. The ore was mixed for 3 hours with iron only and then mercury was put in it little by little in the form of a fine spray, made by pressing the metal through chamois skin. This was done every 10 or 15 minutes. The wt of mercury put in was 2 lbs. 11 oz. This amalgamated with the silver. The ore was mixed for 3 hrs. with mercury and then the paddles were stopped and the mercury and silver amalgam was separated from the ore by panning.

Towards the end, this process being rather slow, an attempt was made to separate the ore by a modification of the spitz lutte, but this had to be abandoned as a part of the mercury was in so finely divided a state, that it floated off before the heavier pieces of rock did. The whole had to be panned. The amalgam obtained weighed 2 lbs. 1oz. In attempting to distil off the mercury an accident happened to the apparatus whereby all the mercury went up the chimney. The silver which remained was scorified ~~&~~ cupelled. The silver button weighed .00305 lbs.