

The Newburyport 18
Silver Lead Mine.

By
J. H. Fushman.
176 Boston

Thesis. 1876.

The Newburyport Mines of
Silver and Lead

Julius H. Sessmann
Mining Department.
Mass. Inst. Technology.

The Newburyport Mines of Silver and Lead -

I purpose to give in this paper, in addition to a general account of the history, geology, and mineralogy of the newly discovered mining district in Newburyport, the description of experiments conducted by the writer in conjunction with Mr Townsend in concentrating and smelting ores from the Chipman lode. - I regret to have been forced to omit many proposed investigations in connection with this subject on account of the lack of time to perform the necessary analytical work. I have endeavored to make the subjects presented as complete as possible.

Analyses, which I have not verified personally.

I have marked by a star. (*)

Section I.

Discovery and Development of the Mines —

There is, in the north-eastern part of Massachusetts in the town of Newbury, extending into Byfield, Georgetown, & Howley, and northward along the Merrimac into New Hampshire, a range of barren, rocky hills intersected by a large number of metallic veins of greater or less importance.

The existence of these veins, although in a country which has been settled for nearly three hundred years, and in the vicinity of large towns, was practically unknown until within a few years; when the accidental finding of a piece of galena by a purposeless wanderer over ^{the} pastures of

Highfield led to their discovery; and this was the beginning of that prospecting fever which soon spread over the whole of New England, causing the farmers to leave their fields and wander among the rocks, with which they are so abundantly blessed, in search of the precious metals, and feeling only too happy, if they were rewarded by the discovery of a small vein of pyrite or of galena; but such a discovery was often the beginning of their ruin; for close on their heels followed the speculators, who often succeeded in robbing them of their money by imposing on their ill founded hopes. The excitement has now nearly subsided except in the immediate vicinity of Newburyport. It has served to show how the education of people has been neglected; they should have been taught to observe as well as to read and write; hence more attention should be devoted to the natural

sciences in our public schools.

To return to our subject, Rogers, the first discoverer, imparted the information to a Mr Adams, who bought the land of the owner to be used ostensibly as a cow pasture. On this land the first pit was sunk in May 1874 and float specimens carried by drift taken out. In October the vein was found and a shaft sunk under direction of Mr Chipman. This vein (The Chipman Lode) has proved so far the most valuable and will be principally treated in this section. It has passed through various hands, and is now owned by the Merrimac Mining Co., who also possess the Boynton property over the continuation of the same lode. The lode has been systematically mined under the superintendence of Mr Paterson. In order to understand the work done, it will be necessary to give some details of this vein, which will be treated more particularly

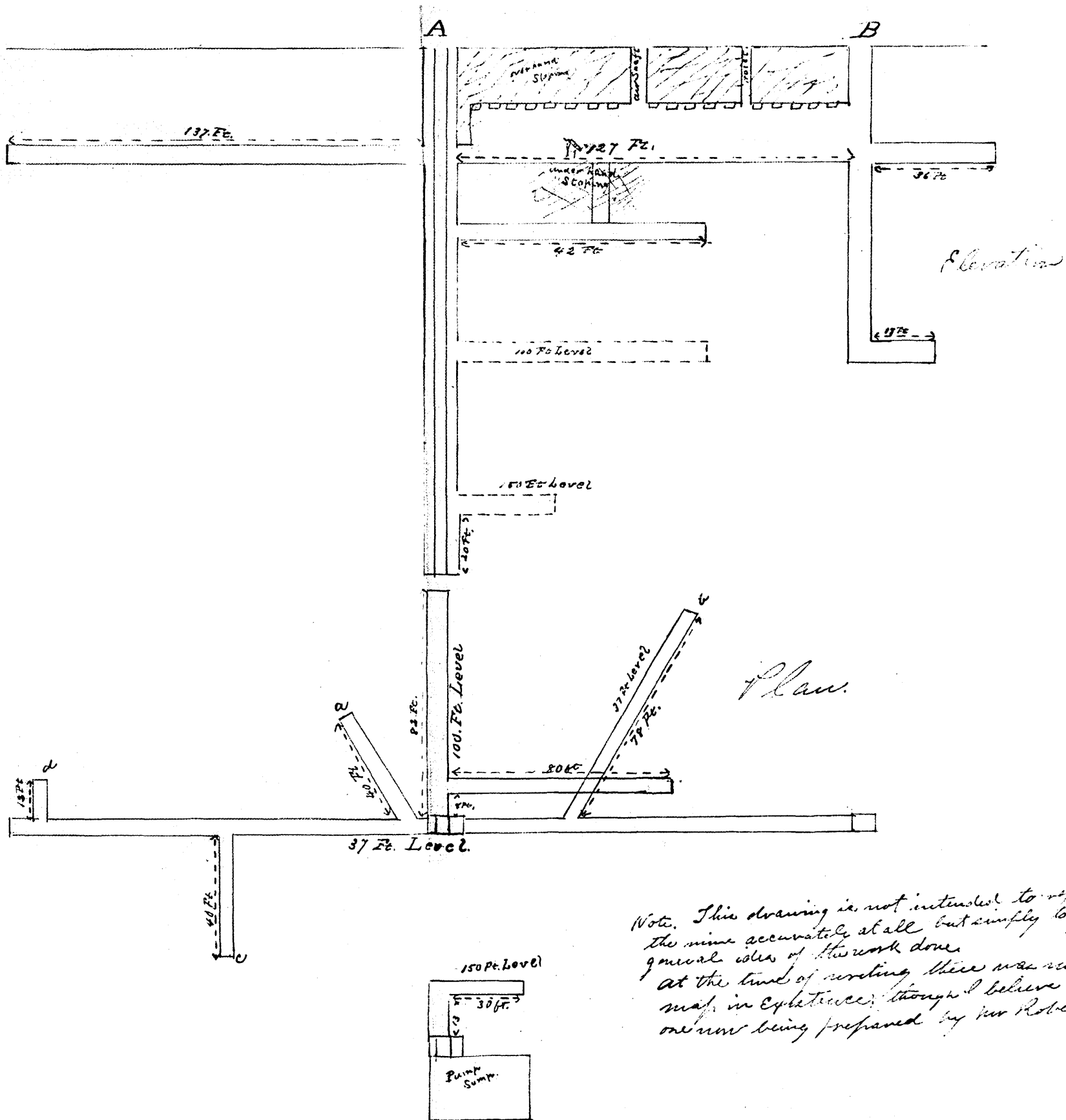
from a geological standpoint in the next section.

The strike is North 80° East.

The dip varies. It starts nearly vertical but swerves to the S.E. at the depth of 50 or 60 ft. The vein is on an average three feet thick but in places it widens to six ft.

Besides the main vein there are rich streaks, sometimes over a foot thick, often indicated by crosscuts on the surface, many of which have been worked.

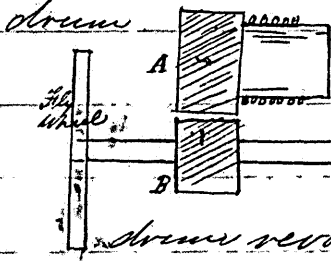
The accompanying drawing, which I owe to the kindness of Mr Robertson, represents the underground work as it had progressed by April 15th 1876. The elevation is represented on a vertical plane parallel to the strike, the observer looking southward. A is the main shaft, which has reached the depth of 170 ft. It is well-timbered throughout and divided into three parts. The shaft B is 127 ft



S. 80.^{W?} E is the old Boynton shaft and is now only used for the purpose of ventilation. The first level has been driven at a depth of 37 ft., and the ore stoped out to the surface. Numerous headings (a. v. c.) have been made, some striking rich streaks, others proving barren. The second level, driven at 100 ft depth, shows the vein departed 8 ft from the vertical shaft. The 150 ft level shows a departure of 13 ft. At this point the pump is placed. The work is now progressing in the 100 & 150 ft levels, with a good show of ore; though the relative proportion of first class ore is said to diminish with the depth. Blasting is done with Nitro-Glycerine.

Pumping and hoisting are done with a single engine of about ten Horse power. Hoisting is done by means of two drums working independently. A peculiarity in the hoisting machinery, worthy of notice, is the transmission of the power

from the shaft to the drum, by means of the friction of drums of soft wood, placed on the shaft and drum respectively, when they are pressed against each other by means of long levers. (See Figure & Plan of shaft &

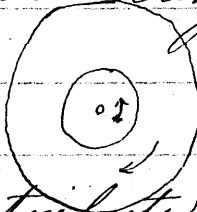


When A is pressed against the drum B on the revolving shaft the friction is sufficient to make the hoisting drum revolve.) The advantages of this form of hoist are said to be, cheapness of original construction, economy of power, greater security, and the ease with which it can be repaired; the breaking of a single tooth rendering an iron gearing useless.

Machinery for the concentration of the third class ore is now being constructed under the direction of Mr Robertson.

The ore is to be crushed by rolls 14 inches in diameter, made of soft iron, which is preferred to chilled iron, because it is said to wear more evenly; it being packed firm by the hard rock. — The sand is to pass from the rolls to a large revolving cylinder like

those used in California; This separates the slime from the coarse sand. The latter is concentrated on large jigs, rectangular in shape, with side pistons. The slime is concentrated both on inclined frames and on large circular buddles. The latter

re- consists of an inner circle revolving 3 times per minute to distribute the water and an outer ring revolving in the opposite direction once per minute.

The necessary water is to be derived from a four acre pond made by damming up a small stream.

The above work is being executed in a systematic and economical manner, and is predictive of the success of the company, if the supply of ore does not fail. The company are daily hoisting 5 tons of first class ore valued at \$100 per ton, in addition to the second class ore valued at \$52 per ton. The profits which are now very large will diminish as the mine increases in depth. The ore

is smelted in New York. Lately, however, two smelting works have been erected here (1888), which might have proved of great importance to the region had the managers been satisfied with established methods of smelting. They prefer to experiment with new inventions which seem to be in some respects in violation of rules established in all previous experience; and in any case ^{it} will require much time and capital to perfect them.

The Four Rocks mine is next in importance to the Chipman. It is superintended by Mr. Kempton, who has sunk a shaft 40 ft deep and taken out considerable ore, most of which requires concentration. A great many shafts have been sunk in the region; some on very rich looking veins; but most of them will have to be abandoned on account of the narrowness of the veins.

Section II. —
 The Geology and Mineralogy of
 Newburyport —

In the geology of this region, we have presented some very complicated problems, the solution of which will require much more study. Before discussing the theoretical questions, I shall briefly give the succession of rocks, as they appear in the town. — Just south of the town, we find eruptive diorites and granites; on crossing the line we come first to a bed of Petro-silex, having an easterly strike; then north of the Parker river are beds of talcose & chloritic slates; and beyond these we have banded porphyries gradually changing to ordinary Petro-silex as we go north. These beds form a small synclinal, undoubtedly of Huronian age. But as we go a little farther to the north-west, the character of the rocks changes entirely; we see ^{little} ~~nothing~~ but granite & diorite, which occupy the greater portion of the mining lands. Cropping out.

from these granites, are beds of an impure
 limestone, containing much quartz and
 a very beautiful variety of serpentine, the source
 in fact of the fine serpentine marble, ~~for~~
 for which Newbury has long been noted by
 mineralogists. With these appear beds of a green
 rock which I am not able to name and whose
 character has never been well made out. It
 varies in color from very light to ^{quite} dark green;
 in texture from very fine grained specimens,
 looking like and often mistaken for a quartzite,
 to those having the appearance of a coarse sand-
 stone. It varies as much in composition and
 hardness.
 A splinter of the light variety examined with
 the microscope showed grains of serpentine
 enveloped by a white translucent quartz
 and specks of mineral sulphides invisible to
 the naked eye. In other specimens the quartz
 was partially replaced by calcite. It seems to
 pass gradually into a limestone, and it is
 often impossible to find the dividing line.
 Scattered all through these beds are crystals
 of galena and pyrite. The decomposition

- of the latter giving the yellow color to the soil
 on the outcrops of the beds. It is in this
 formation only that the metallic veins exist -
 But it is the granite which renders the geology
 of this region so complex; and this complication
 is best explained by regarding them as eruptive
 masses, which broke up through the beds last
 described, dislocating and fracturing them. In
 the fissures thus produced the metals diffused
 through the beds were ^{re-}deposited forming the
 numerous veins, whose banded structure forbids
 any doubt of their accretionary origin.
 It is true these granites (?) have been regarded as
 bedded gneisses; but the absence of visible lines
 of bedding, the existence of the granite in wedge
 shaped masses, as is shown by the diamond drill
 borings of Mr Keupton, and its relation to
 the eruptive diorites, tend to verify the first
 hypothesis.

Regarding the age of the metal bearing
 beds, my classmate Mr Jaques, who has
 devoted considerable time to the geology of
 this section, suggests that they extend

under and form the basement, as it were, upon which were deposited the ^{beds of the} synclinal before described. This would make them as old or older than the Huronian age. There is little doubt, that they belong to the crystalline rocks however.

— The Mineralogy of the Veins —

In the Chipman lode, which is mineralogically typical of the majority of the veins in the section, the vein rock is principally quartz. In this we have bands of galena, with more or less tetrahedrite, together with siderite, pyrite, chalcopyrite, zinc-blende, etc. — The galena which, is highly argentiferous, ^{occurs} both in ^{the} coarse and in extremely fine varieties. Specimens of the coarse galena assay \$56 in silver to the ton. While the fine galena sometimes assays as high as \$75, and even higher. Blow pipe reactions show this difference to be due to intermingled grains of tetrahedrite in much of the fine galena. The tetrahedrite

is extremely rich in silver, often containing as much as 13%. Whether the pyrite, which is very abundant, contains gold and silver is not known. I got tests for nickel in both pyrite and chalcopyrite; and Shaw told me one vein of pyrite contains 5% of nickel.

The siderite occurs in an almost black and in very light varieties; it often contains considerable manganese. Blende forms but a small proportion in the Chipinaw lode but is more abundant in other veins.

A soft mineral of a dull black color assaying high in silver was found to be tetrahedrite mixed intimately with silica.

Besides the above mentioned minerals, there are various silicates, and minerals, produced by the alteration of the above by the action of the weather or by sea-water.

I saw a film of native silver in a float specimen; probably from the decomposition of tetrahedrite. Native Gold (?) is also reported.

Section III.

Concentration of Third Class Ore.

For the following experiments in re-dressing, 16.62 pounds of third class ore from the Chipman lode was taken. This was first cribbed and specimens taken for blow-pipe analysis; it was then passed through a Blake's crusher, and crushed by rolls, till it would all pass through a $\frac{1}{2}$ in sieve. Rolls were preferred to stamps, because it was desirable to keep the ore as uniform in size as possible. $7\frac{1}{2}$ lbs, i.e. $\frac{1}{2}$ of the whole, was lost in the shape of dust; a portion of this loss is due to the cracks in the floor.

The crushed ore was sampled and the following partial analysis made from which the approximate mineral composition was calculated. Some of the Cu. copper, accredited to Chalcopyrite, probably exists as tetrahedrite.

— Composition 3rd grade ore —

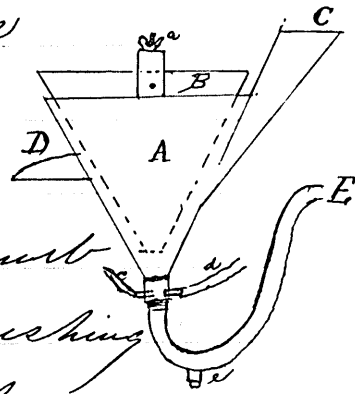
		Percent		
Lead	-----	9.00	Galena	----- 10.40
Copper	-----	.40	Pyrite	----- 6.33
Iron	-----	13.20	Chalcopyrite	----- 1.45
Zinc	-----	.05	Siderite	----- 22.73
Silver	-----	.03	Blende	----- .07
Sulphur	-----	6.05	Silver	----- .033
Silica	-----	50.45	Quartz	} ----- 59.08
Alumina	} Qualitative Tests		and Siderite	
Magnesia				
Arsenic				
Antimony				
			100.00	

The problem was to separate the galena from the other minerals; it was soon found that an attempt to separate all the pyrite would entail a great loss of silver; the greater portion was therefore retained, and this explains the small percentage of lead in the concentration.

As a preliminary to concentrating, the ore was graded or classified; that is, the fine dust was separated first, and it was ^{then} arranged in ^{portions} products, not with particles of the same size, but with a known relation between the sizes of the particles of different Sp. Gravity.

— The Spitz-Lutte —

This method is preferred to a sizing by sieves, wherever the mineral to be processed forms but a small proportion of the whole ore. The classifying machine used was a modification of the German machine, the Spitz-Lutte. This consists of a V box, A, into which is placed a second V box B which can be moved up or down by a thumb screw *a*, increasing or diminishing the space between the walls.



The ore is fed in, ^{with water} at *c* and discharged at *D* and *E*. An upward rotating current is produced by admitting water under pressure at *c, d*. The force of the current is varied by varying the space between the walls of the V^2 . A system of three Spitz-Lutte, discharging ^{from D into c,} into each other, each with a weaker current than the preceding one, were used.

The principle upon which the classifying is done is as follows. The particles of all sizes enter

Lutte I and fall by gravity till they reach ^{action} the upward current, which has a two-fold _n; by partially counteracting gravity, it only permits the heavier particles (large particles gangue large & medium galena to pass out at E; but large particles present more surface to the current, hence there is a second tendency to drive up the large particles. The two combined causes a separation into two products; a product composed of large quartz, large galena, (and Pyrite), and medium galena, discharged at E. (Prod. Lutte I.); and a product of large quartz and fine quartz with fine galena discharged into Lutte II. Lutte II weeds out more coarse quartz and fine galena making a well graded product discharged at E' (Prod. Lutte II). Lutte III takes the discharge at D' and gives us two products Prod. I Lutte III at E", consisting of fine galena and coarse quartz, and Prod. (2) Lutte III at O" having only very fine galena with both fine and coarse quartz. The following table will show the ~~pro~~ proportion of the four products with the amts of silver and lead in each -

Products Spitz-Lutter.

	Weight of Product.	Weight Galena.	Oz. Silver	% of Entire Galena in ore.	% of all silver in the ore.
Lutte I.	255 ¹ / ₄	61.08	1606	36.30	20.39
" II.	218 ³ / ₄	26.73	906	16.88	11.53
" III Pr.(1.)	188	14.84	1.105	8.94	* 13.40
" III Pr.(2.)	909 ¹ / ₂	54.	35156	32.08	* 44.70!
Slime	-	-	-	-	-

1628 lbs of the ore was taken and was fed at the rate of 180 lbs per hour, which was the full capacity of the machines.

The experiment showed that the above system, though working on the right principle, is susceptible to many improvements. In the first place, the amount which can be fed in a given time is too limited; secondly the work was being stopped constantly by large pieces getting into the bent discharge tube; also by ^{the} sand packing in the bend and tube.

Again, on acct of the large proportion of the very fine galena, it would take a very large number of lutes to make a perfect separation in the shape of well-graded products, as is shown by the similar comparisons of Products 1 & 2 of Lutte III. Such a

* Note that the silver predominates in the finer products —

- Jigs -

system would take up much space, and would require a great quantity of water; it would be better, therefore, ^(with this ore.) not to attempt a complete separation into graded products, but to remove the coarse galena & from the fines by means of a single Lute of sufficient capacity, so modified, as to allow the discharge of a much larger first product by diminishing the force of the upward current. This product could then be treated on jigs, and the discharge, on the End and side-bump tables successively, as will be explained later. Since these experiments were made, Prof. Richards has invented a modification of the Dolly tub which seems to come up to the stated requirements -

- Jigs -

The products from Lute I and Lute II were concentrated on two small jigs with side pistons and rounded bottoms. Each jig was double, that is had two compartments with separate sieves and pistons; the overflow from the first sieve ^(A) passing into the second ^(B) and ^{being} again concentrated, before being rejected as tailing. The ore was fed on continuously till

galena was seen escaping in the tailings. ^{were cleaned out} The jigs, about every half hour. Prod. sent I weighing 255 1/4 lbs was concent on jig I in 6 hours 7 minutes. Prod sent II, weighing only 218 lbs and poorer in lead was concentrated on Jig II in 6 hours 29 minutes. Each jig produced four products and a tailing, as is shown by the subjoined table. The B products were all reconcentrated, as the fear of loss caused them to be poor at first. The large proportion of lead in the tailing is due to small specks of galena adhering to large particles of ^{quartz} galena. Their recrushing and concentration is only a question of economy.

Products	Weight	- Jig I -		Cu.	Fe	Sulphur	SiO ₂	n.c. Silver
		40 Pk.	40 Pk.					
A. Prod. ---	53 1/2	54.60	.86	16.25	19.70	5.00	.085	
A Sifting ---	49 5/8	33.60	.60	12.10	12.39	21.62	.085	
B Prod. ---	25 1/4	14.41	*1.13	26.28	18.38	16.32	.03	
B Sifting ---	7 5/8	3.13	.54	13.98	5.14	50.12	.033	
Tailings	119 1/4	2.50	.40	12.80	3.45	57.97	.01	
		255 1/4						

* Due to much Chalcocite.

	n.c. galena	n.c. Pyrite	n.c. Entire Silver
A.P. -	63.05	17.81	8.46
A.S. -	38.80	11.40	7.83
B.P. -	16.70	25.46	1.80
B.S. -	3.62	7.37	4.9
Tail. -	2.88	4.86	2.21

Rate - 40 lbs per hour.

Fig. II.

	Weight	Pb.	Cu	Fe	S	SiO ₂	Ag.	Pyrite	Galena	% whole silver.
A. Prod.	24 3/4	42.20	.66	16.75	16.17	8.00	.09	18.43	48.73	4.40
A. Sift	26	24.58	.40	13.01	13.98	23.26	.065	16.01	28.38	3.15
B. Prod	29 1/8	9.37	.58	25.53	16.41	12.74	.02	23.75	10.98	1.08
B. Sift	5	6.99	.64	19.30	8.60	36.03	.043	11.89	8.06	.40
Tailings	134 1/4	2.50	.50	16.01	4.27	54.92	.01	6.16	2.88	2.50
	218 3/4									

* Mr. Townsend's analyses.

— Product (1) Lenth III. —

This product was intended to be run on the Rittinger table; but on acct, either of insufficient grading or, what is more probable, poor feeding arrangements, it did not succeed; it was therefore first analyzed, then partially concnt. on the End bump table, and added to the concentration from the same table of Prod. 2 Lenth III.

Weight	% Pb.	Cu	Fe	S	SiO ₂	Ag.	% Galena	% Pyrite
188.	6.84	46	12.48	4.91	52.01	.04	7.90	5.93

From End. Bump.

Wt Concentration = 78 lbs. Tailings = 110 lbs.

— End Bump Table —

Prod 2 Lutte III, which is the largest product, contains all the galena in a fine state of division; hence it can be best, ^{partially,} concentrated on some sizing machine, though sifting alone would answer the purpose. The machine chosen was the End bump table so common to require explanation, which separates the fine from the coarse particles.

The lattes were poor enough to be thrown away as tailings. To do good work this table requires a perfectly uniform, regular, feed; if feeding stops for any cause, a layer of coarse material collects on the fine concentration, ^{which} is covered up when feeding begins again, and the product is injured. This was the cause of the unevenness and large proportion of our concentration to the whole.

	Weight.	Pb.	Cu.	Fe	S	SiO ₂	Ag.	Calc.	Pyrite	% total galena	% total silver
* End Bump Concent.	333 7/8	12.03	0.36	16.52	6.52	41.92	0.53	13.81	7.41	27.23	32.93
* End Bump Tailings	575 5/8	1.15	.35	9.04	2.00	68.71	0.11	1.33	2.95	4.47	11.77
Total	909 1/2										

The whole was run in three hours not including time for cleaning up —

The concentration was added to the End Bump Concentration from Prod. 1 Lutte III and the whole passed through a Dolly test. This was only an experiment to see

— The Dolly —

to what extent the galena would be removed.

We only succeeded in removing a portion of the galena in the Dolly Product, which consisted of Coarse Quartz and medium galena (the coarse galena all being previously removed). This was concentrated on the jig's —

The Dolly Discharge was a nicely graded product (the dolly acting like a Sphero-balls) of medium quartz, and both fine galena and quartz. This was concentrated by removing all the medium quartz, by passing the product through a 120 sieve, giving us an extremely poor tailings, and a product of fine quartz & fine galena (with Pyrite) which went on to the Rittinger. Instead of sifting the product, ^{it} might have been treated equally well on the end-bump table.

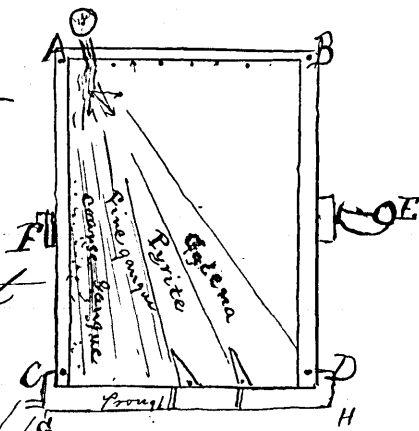
— The Rittinger Table —

The Rittinger table is a device used some in Germany, which I believe has never before been introduced into this country.

It is a perfectly smooth inclined table A, B, C, D suspended by ropes so arranged as to vary the inclination of the table for different sized product. —

— Rittinger Table. —

By means of small jets along A B a uniform sheet of water flows over the table; the amount of water being regulated by cocks. The table receives a sharp bump from E, and is brought back into position by the spring F which impart a gradual motion. The well sorted ore is fed on with water from a Dolly or Lute at A.



The theory of the separation is as follows. — All the particles have a downward motion imparted by the current and a transverse motion due to the table being pushed forward under them, as it were, by the sudden bump — These combined give them a diagonal movement; but the larger particles presenting more surface to the current will tend to move downward faster than the small ones; again the fine particles of high Sp. Gr. sink to the bottom and are more retarded in their downward motion by the friction of the table, than those of low Sp. Gr. Friction

does not retard the particles in their transverse course, on acct of the sharpness of the blow. In a product, therefore, where the heavy minerals are comparatively fine, we will have, beginning at the right, successive bands of minerals in the order of their Sp. Gr., and further to the left comes the band of coarse material hurried down by the current - By a proper arrangement of pointers we can intercept any one of these bands. The diagram shows how our product distributed itself on the table. It was found advisable to retain much pyrite with the galena - A product of 50 lbs assaying 0.137% of silver i.e. 13% of all the silver in the ore was obtained, and proved to be the richest product ^{in silver} obtained, owing to its fineness probably.

The advantages of this machine are 1. that it is continuous in its action. 2. That, with a well graded ore, the different minerals can in most cases be separated in a considerable degree of purity 3. that it can easily be adapted

to the character of the ore to be concentrated. 7

The greatest objection to it is the small amount of ore treated in a given time, though its capacity may be increased somewhat by increasing the size — In lieu of any results regarding the amount concentrated in a given time, which the irregularity of the feed prevented us from getting; the following data from Kettiger may be of interest.

"On a table 4 ft. by 8 ft., inclined 6° for "meble" ($\frac{1}{2}$ to $\frac{3}{4}$ in. diameter) 3° for Slimes, 316 lbs of the former or 43 lbs of the latter can be fed per hour, from a feed trough 8 inches wide.

A cubic ft of water should be fed with every 15 lbs of the former or 6 lbs of the latter.

The No of bumps per minute should be 70 or 80 in the first case and 90 to 100 in the latter, moving the table $2\frac{1}{2}$ inches⁽¹⁾ or $\frac{1}{2}$ to $\frac{3}{4}$ inches." So a proper table will treat $2\frac{1}{2}$ ton ore in 24 hours with very little power expended —

Slime

All the water used in the preceding process was allowed to settle in a tank for several days, it was then drawn off and the slime dried, weighed sampled and analysed with the following results —

Wt.	% of ore treated	per. per.	Slime				Ag.	Percent entire Silver
			Cu.	Fe	S.	SiO ₂		
56 1/2.	3.48.	6.88	82!	12.50	3.91	50.50	.10!	10.50 %.

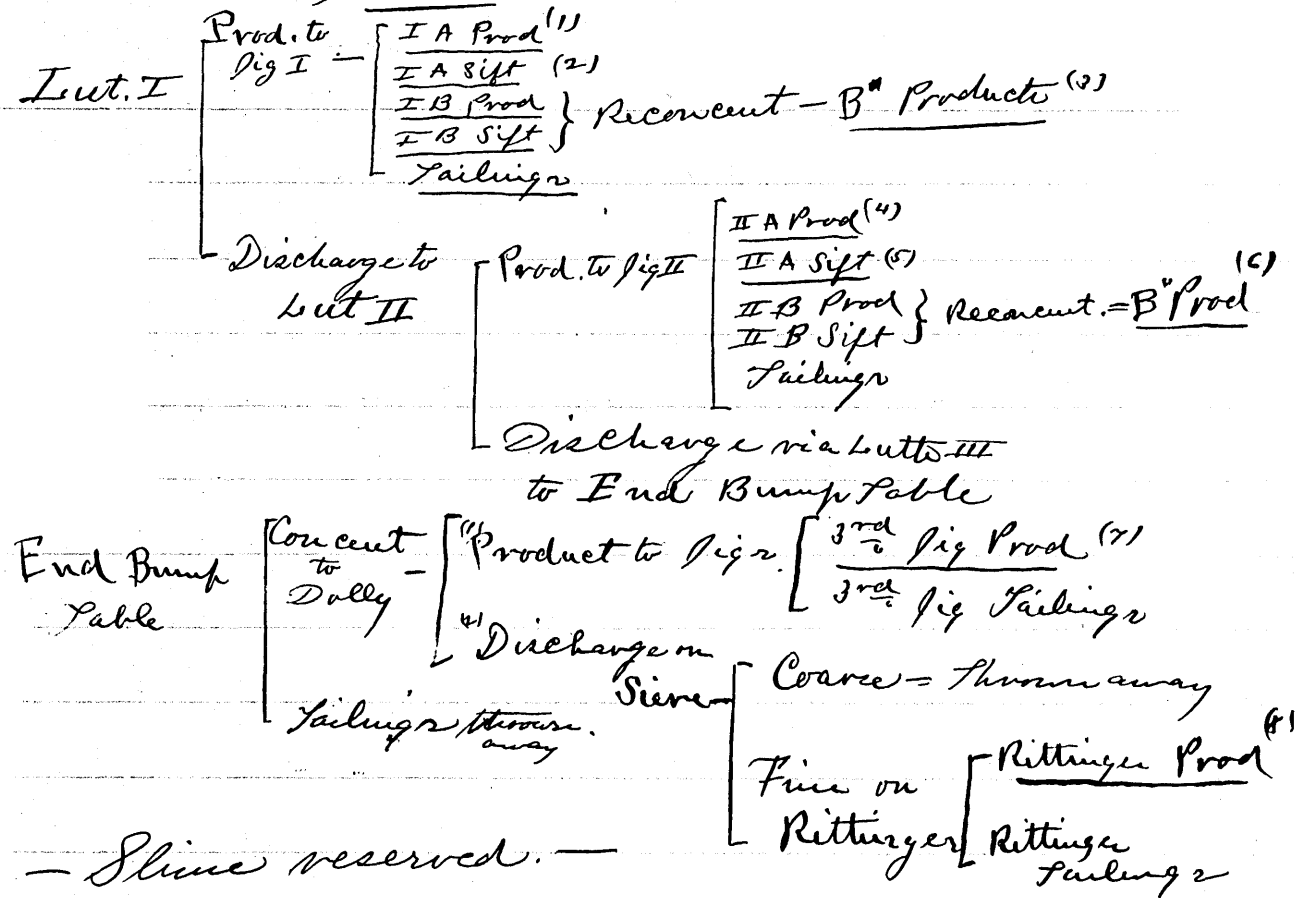
Here we have a product though only 3.50% of the ore treated, containing over 10 percent of the entire silver. The large percentage of Copper ^{or Ag} proves that the fine tetrahedrite holds this silver as we have had occasion to notice in previous fine products. Its fineness makes it a very difficult product to treat; and the only method I could suggest would be to mix it with milk of lime, or some other similar material to make it coher, previous to roasting —

Final Concentration

The following scheme is a brief recapitulation of the ore-dressing experiments; The products under a cover were added together.

and constitute the final concentration

— Table. —



— Final Concentration —

Weight -	2.33 lbs	Concentration contains	73.42
p.c. lead	46.02	p.c. of entire lead	54.71
p.c. Silver	0.115	Concent contains	23.85
lbs. Lead per ton	920	p.c. of entire Silver	34.79
oz. Silver per ton	33.6	Lead lost in tailings	2.73
lbs. Copper	0.56	entire Silver " " "	10.50
p.c. of ore taken	13.71	entire Lead in Slime	

Nearly 1/4 of the lead and over 1/3 of the silver is practically lost in treating this ore

- Section IV. -

- The Metallurgical Treatment of Newburyport Ore -

For the metallurgical experiments about ~~80~~⁷⁶⁸ lbs of Float ore from the Chipman lode was taken. Being in pieces of considerable size, it was first broken by hand; and, without any attempt at selecting the best pieces, passed through the Blake's crusher and rolls, which reduced it to a size allowing it to pass through a $\frac{1}{2}$ sieve. It was then sampled and the sample analysed; Being a weathered ore, this analysis does not enable us to give the mineral composition with any degree of accuracy. The most important minerals are, - Galena, pyrite, chalcopyrite, siderite, zinc-blende, tetrahedrite, quartz, etc -

— Analysis Float Ore —

	pc.		no.
Lead	44.29	Zinc	0.73
Copper	0.70	Silica	12.33
Silver	1.503	Alumina	.84
Arsenic	.78	Sulphur	16.88
Antimony	1.14	Calcium	Qualitative tests. —
Iron	12.02	Magnesium	

The ore therefore contains ~~88~~^{340.25 = actual amt lead} 5.8 lbs lead, 14 lbs Copper and 44 ounces silver to the ton. The first question to occupy our attention was the choice of the best method for extracting the above named metals from the ore. In this choice we were governed by composition of the ore. The large percentage of silica renders smelting by the "roast-reaction" process impossible. It might be smelted in a reverberatory furnace with metallic iron, as is still done in Chicago with ore still poorer than this. But the high cost of iron, and the production of a large quantity of rich matter requiring subsequent treatment, has led to the almost universal abandonment of this method. The same can be said of the method of smelting raw ore.

— Roasting —

with metallic iron in blast-furnaces.

The remaining method of roasting off most of the sulphur in a reverberatory furnace, and subsequently smelting in a blast-furnace with proper fluxes was the one chosen. One of the ores smelted in Freiberg, by this method is very similar in composition to ours; I shall therefore take occasion^{to} frequently compare our results with those reached there. —

— Roasting —

In practice the roasting and agglomeration of the ore immediately follow each other in the same furnace. For the sake of convenience and also to get a sample, we treated them as separate processes.

The roasting was simultaneously conducted in two small reverberatory furnaces. The temperature was raised gradually and afterwards raised to just below the fusing point; at which temperature the ore was kept for some time —

In this way the greatest possible amount of Sulphur was driven off, with a small formation of sulphates. — The charge was heated four hours and then sampled and

— Agglomeration —

analyzed for sulphur. (Results under agglomeration)

During roasting there was some antimony & arsenic driven off but apparently more of the former than of the latter —

— Roasting Record —

Furnace.	No. Charges	Wt. Roasted lbs.	Time of Roasting hours	Fuel Consumed pounds	Pounds fuel per hour	Fuel per lb. ore.
Long.	4	340	15 1/4	207	13.6	.609
Hollow Bed.	5	435	19 1/4	311	16.7	.73 lb.

— Agglomeration —

This process was conducted in the Hollow Bed furnace which is a Fletcher furnace on a small scale. In this case the charges were very much increased; — The temperature is greatly raised and access of air prevented; a smoky flame was also produced; this caused a reduction of the sulphate to sulphide consequently a further elimination of sulphur. At the end of an hour and a quarter, no more sulphur appeared to come off and the ore was in a pasty condition; It was withdrawn, and on cooling proved to be a black glassy slag (Silicate of lead & iron) with little free silica. It was then sampled and

analysed for lead, Sulphur & silver -

— Agglomeration Record State —

Hollow bed furnace

Time	Event	Wt. Charge lbs	Fuel	Notes
4. A.M.	Fire lit			
7.30	Simmering, heat.		280 lbs	Between 7.30 and 11 - Attached slag from former run was being removed - It will be seen after furnace is heated up it consumes 60 lbs per hour or 1/3 lb of fuel for every lb ore -
11-	1 st charge in	194		
12-30	2 nd " "	194		
2-00	3 rd	194	238	
3-30	4 th	132		
5-	Drawn			
		714	518	

The following table gives in a condensed form the results of Roasting and Agglomerating -

— Table Roasting & Agglomeration —

	Wt.	h.c. dead.	% Sulphur	h.c. Silver	Weight lead	oz silver
Original ore.	774 3/4	44.30	16.88	0.1525	338.6	17.063
Roasted "	714 5/8	—	5.65	—	—	—
Agglom. "	650 1/2	49.10	3.00	0.16.25	319.4	15.47*

	Reduction wt. round.	Loss weight h.c.	Loss entire Sulphur %	Loss lead h.c.	Loss silver h.c.
Original Ore	—	—	—	—	—
Roasted	59 3/4	7.71%	11.23	5.68	8.03!
Agglom.	54 1/8	6.98	12.63		
Total*	113	14.69	13.88	5.68	8.03

Allowances were made in calculation for a sample of 10 lbs taken from the ^{Roasted ore}

- Roasting -

From the data in the preceding tables, we see, first, that the removal of sulphur was as complete as practicable with an ore containing pyrite. In Freiberg the Agglomerated ore contains from 3 to 5 per cent of Sulphur, while at Stollberg, where it is free from pyrite, it contains only 1 per cent. We arrived at this result by roasting less than $5\frac{1}{2}$ hours while at these other places each charge remains in the furnace 16 to ¹⁸ hours.

This is explained by the smallness of our charges and the ore being in a smaller state of division.

Losses — The loss of lead (5.64%) in fume is rather high, 2 to 3% being the usual loss in good works; but the loss of silver (8%) is excessive. I believe it to be due principally to two causes; first and foremost the ore was ground too fine causing the softer, silver bearing, tetrahedrite to be reduced to a very fine powder, much of which was carried off mechanically; and in the second place the large amt of arsenic and especially of Antimony caused considerable loss. The first cause of loss can be avoided by leaving the ore coarser, at the expense of a longer roasting.

- Smelting -

period. The loss can be to a great extent recovered by erecting fume condensation chambers, which are an almost indispensable adjunct to a furnace roasting such impure ores as *Carroll*.

- Smelting the Agglomerated ore in a Blast Furnace -

In the agglomerated ore, we have a product consisting of the oxides and silicates of Fe, Pb, Cu, Zn, etc., together with a small percentage of sulphides; and our problem is to reduce the compounds of lead and silver to the metallic ^{state}, and separate them from the other materials. This is done by the reducing power of heated carbonic oxide from the fuel, upon the oxides reducing them to the metallic state; and also by the reduction of FeO to Fe, which then replaces lead and silver in the sulphides and allows them to separate.

This last reaction gives us a matte, and in this matte all the copper in the ore will be found. But we must, in order to separate our metal and matte from the other material

— Slag calculation —

completely fuse this latter, and this is done by adding to the charge fluxes in such proportion, as will form with the ore a proper fusible slag. Preliminary to smelting came therefore the

— Calculation for Slag. —

The properties aimed at in the slag were, 1st sufficient fusibility to allow complete separation of lead & matte, 2nd the retention of as small as possible an amount of lead as silicate, and 3rd the smallest tendency to attack the bricks lining the furnace. —

Experience has shown the proper slag to be a proto-silicate, in which FeO is the principal base.

No arbitrary rule could be followed in calculating the slag, but it had to be adapted to the nature of the ore and fluxes.

The latter consisted of "puddle cinder" a highly basic silicate of iron and limestone; the latter to diminish the quantity of slag, it being permissible to replace FeO by CaO to the extent

of 12 or 15 per cent. Higher percentages tend to make the slag less fusible, though otherwise it is a better agent than iron even, on acct of its action being quicker in reducing any sulphide. ~~But the~~ Caustic lime is preferable to the carbonate however, and is alone used at Freiberg.

Below are the analyses of the ore and fluxes. That of puddle Ciicles is only approximate being taken from Mr James's analysis of a different sample.

<u>Aggr. Oliv. Ore.</u>	<u>Puddle Ciicles.</u>	<u>Limestone</u>
SiO ₂ - 14.45	SiO ₂ - 17°	CaO. - 55.60
FeO - 18.28	FeO - 73	CO ₂ - 43.80
Al ₂ O ₃ - 1 -	Al ₂ O ₃ 6.2	SiO ₂ 6.0
S. - 3 -	CaO 1.9	
P ₂ O ₅ - 52.88		
CaO -		
MnO ₂ }		
SiO ₂ }		
And ₂ }		

The composition of the slag calculated for is shown in the next table; and from this after making allowance of a sufficient amount of FeO to remove the sulphur as iron matte (FeS)

the percentage of materials of the blast furnace charge exclusive of coke is calculated. —

Proposed Slag.		Charge calculated	
FeO	— 55.01	Ore.	— 66.20 lbs.
SiO ₂	— 29.51	Lime	— 8.40 "
Al ₂ O ₃	— 4.68	Fiddler Cinder	— <u>25.40 "</u>
CaO	— 10.86		100 lbs.

Calculated weight = 50.62 lbs.

The value of these preliminary calculations was very much diminished by the addition of variable amounts of raw galna to each charge. — This was done to prevent the formation of "bears" in the furnace. I fail to see how adding more sulphur to each charge (from 1 to 27, weights added) should have this effect, and it seems to be contrary to the principle laid down by Grimes & Percy, that in smelting argenteif iron galna the smallest excess of sulphur tends to produce losses, which the use of additional iron can not remedy. In Freiberg, bears are prevented by adding a constant percentage of slags from previous runs to each charge. —

— Blast Furnace Run —

The charge as calculated, with the addition of coke and galena, was smelted in the small 3-tiered blast furnace which resembles the slag hearth used to smelt grey slags from the Flutchie furnaces. The rate of charging was regulated by the decrease in height of the material in the furnace as the slag and lead flowed off. The escaping gases were kept as cool as possible to diminish loss by flame. The proportion of coke was determined by a previous experiment to see how much the tierces would burn. It was varied through the run to suit the working of the furnace. A valuable addition to the furnace, was the forehearth, into which the lead and slag flowed continuously as they formed, and were kept in a molten state by means of a small coal fire, allowing a complete separation of metal, which was tapped into moulds from the lower part of the hearth, while the slag overflowed into buggies producing no foul slag at all.

— Charges —

The smelting campaign was very successful; the slag being liquid and nothing interfering with the downward course of the charges. The record slate will best show the details. The smelting was accompanied by strong fumes of arsenic sulphur and lead which were blown out by the blast. At the beginning and end of the campaign Reverse slag was charged to start the furnace and also to wash out all products of the smelting of ore at the end; this is the source of the "beginning & End" slag which is simply a mixture of Reverse and the Run slag. The record shows that any change in the character of the charge manifested itself at the tap hole from 10 to 12 minutes later; hence time for charge to descend is about 10 minutes. —

— Charges —

	I	II	III	IV	V	VI	VII
Ore	---	---	30	40	40	16 $\frac{5}{8}$	---
Cinder	---	---	11 $\frac{1}{2}$	15 $\frac{3}{4}$	15 $\frac{3}{4}$	6 $\frac{1}{4}$	---
Limestone	---	---	3 $\frac{3}{4}$	5	5	2	---
Galena	---	---	3	3	3	1 $\frac{1}{5}$	---
Coke	10	8	8	10	8	3.15	.8
Ratio	1:3	1:4	1:6	1:6	1:8	1:8	1:5 (Coke to charge)
Reverse Slag	30	30	---	---	---	---	40

Record Slate
— Feed Door — — Tap Hole —

Time	Time interval	Charge	Height of charge	Notes	Time	Notes	Buggy
8:40				Coke changed	8:34	Blast on.	
8:58			2'	3 holes of coke	9:35	Slag appears.	
9:08				Stopped 3 minutes	9:55	Slag of reflow into Buggy.	
9:15		no coke	2' 2"	Oil g. minor.	10-10	" running freely	1
9:24	9'	I	2' 6"		10:15	Court taken off pot.	
					10:25	Slag hardening and raising in middle	2
9:37	13	I	2' 9"	Flame at mouth.	10:35	Took off court. Put chimney over foreblast	
9:43	6	I	2' 9"	" " "	10:40	Swell of arsenic	3
9:50	7	I	2' 10"	" " "	10:50	Slag well settled	4
10:02	12	II	2' 3"	" " "	10-	Frames of lead at tap hole	5
10:09	7	II	2' 3"	" " "	11-13	Slag well settled	6
10:15	6	II	2' 6"	" " "	11-16	Slag running very liquid	
10-19	4	II	2' 6"	" " "	11-20	Lead flame	
10-30	11	III	2' 3"	" " "	11-25	Lead appeared. Slag well settled	7
10-35	5	III	2' 4"	" " "	11-30	Slag of fine texture	8
10:40	5	III	2' 6"	No flame.	11-33	Tapped lead	
10:43	3	III	2' 6"	" " "	11-35	Slag of fine texture	9
10:48	5	III	3'	" " "	11-37	Slag running very liquid	
10:55	7	III	3' 3"	" " "	12	" well settled	10
11-	5	V	3' 9"	" " "	10:42	Lead fumes	11
11:09	9	V	4'	" " "	10:44	Slag fine & rained	12
11-27	18	IV	3' 6"	Settling 3/16" each charge	10:45	Tapped Lead.	13
11-35	8	IV	3' 9"	Escaping gases cool enough to bear hand in	12-	Stopped Blast. Took court off foreblast	14
11:41	6	IV	3' 9"	Rise on charging 6"	12:10	} Slag liquid. Well settled and free from metallic matter	15
11:52	11	IV	3' 9"	Temp. frame 80° Cent	12:30		16
12-	8	IV	3' 9"	Rise on charging 6"	12:35		17
12:11	11	IV	4'	" " " 6"	12:50	Blast preserve = 1 ft water	18
12:23	12	IV	4'	" " " 9"	1-	Tapped lead — Slag fine.	19
12:37	14	IV	3' 9"	" " " 6"	1:05	Slag appears again	20
12:47	10	V	3' 9"	Flame before charging	1:15	} Slag fine well settled	21
12:59	12	V	3' 6"	" " " 6"	1:20		Tapped lead
1-10	11	VI	3' 3"	" " " 6"	1:33	Slag coarser a little	23
1-20	10	VII	3' 3"	" " " 6"	1:35		24
1-37	17	VII	2' 6"	" " " 6"	1:50	Slag coarse & rained	25
					2-04	Slag has little matter	26
					2-12	Blast stopped	27
					2-33		28
							29
							30
							31
							32
							33

Buggies 9 to 29 inclusive were regarded as Run Slag
The others as Beginning and End Slag.

Table I shows the materials that entered the furnace. Table II the products which were sampled analysed and assayed. The proportion lead lost in steep etc. is great on acct of the small amount of ore run. The slag from ore and fluxes is about 80% of the Agglom. ore. found by subtracting the Revere from the total slag. I.

	Weight	Weight dead	Oz silver	% Copper	Products weight
Ore.	650.5	319	15.47	5.5	Run Slag - 456
Burlingh Galena	55 7/8	34	.86	.5	B&E Slag - 372 1/8
Limestone	85				Rough Lead (matt etc) - 193
Tap Cinder	253 1/2				Steep etc - 195
Revere Slag	320				Total - 1216 1/8
Total	1164 7/8	353 1/2	16.33	6-	

Note - Burlingh Galena contains 6% of zinc.

— Run Slag —

Physically, the run slag is fine grained hard, tough nearly black in color, of quite high Sp. Gravity and containing occasional bubbles filled with very fine brilliant crystals — To get a sample the whole was broken up, quartered, and this portion passed through the rolls and sampled. Of the final sample I made nearly a complete analysis. Surprisingly

— Run Slag —

The analysis adds up to over 100, because I have represented all the metals as oxides, being unable to make proper allowance for sulphur. Too much oxygen has been given to the iron, a portion being in the metallic state. —

Analysis —	
Silver —	.03
SiO ₂	31.23
Al ₂ O ₃	8.91
CaO	8.48
MgO	1.10
FeO	43.44
PbO	5.40
ZnO	1.88
P ₂ O ₅ —	35
Cu	Trace
As	
Sb.	
S. —	1.22
Total	102.06
— Oxygen —	—
total 100	

The composition of this slag presents some curious anomalies; — We have iron in the shape of metallic globules, and at the same time a large percentage of lead and considerable sulphur. — Another curious fact is that though 56 lbs of matte were calculated for only about 9 lbs separated —

This leads to the questions. — How is the sulphur combined? Why did it not combine with the iron to form a matte? Why does a slag with such an active agent as metallic iron in it, contain so much unoxidized lead?

I can give no satisfactory solution of these questions, but will offer the following theories, the third

- Slag -

of which seems, to my mind, to be the most plausible.

First, - The sulphide may be in the shape of microscopic particles of matte (retaining lead) diffused through the slag. Such was proved, by Puzoszek, to be the case with a slag from Przibraw. The objection to this theory is that nearly all the copper in the ore is to be found in the lead and ⁱⁿ the small amount of matte that did separate; were there a diffused matte however, it would necessarily retain most of the copper, as a copper matte. - Again Przibraw slag was infusible on acct of the large amount of zinc oxide, while this slag is sufficiently liquid to permit complete separation of the matte. -

Secondly the sulphur and lead may be due to the incomplete reduction of the raw galena on account of the rapid descent of the charge.

My third theory is that the S. is in combination with calcium, except the small portion which combined with copper and some iron in the true copper matte. The lead is

- Slag -

in the shape of PbO from the original one which was not reduced by the iron, on acct of the rapid descent of the charge. - This is more probable as the agglomerated one was mostly completely fused, and Grüner says it requires much more time to reduce the lead in a fused ore than ⁱⁿ one in which complete fusion was avoided -

Much of the zinc comes from the added galena. - The discrepancy between the actual and the calculated composition of the slag is due to later changes in the charge composition, to the fact that bricks were very much attacked, and that the slag was calculated on the basis that no lead or sulphur would enter it; also to probable variations in the composition of the Puddler Cinders -

It retains the form of a proto silicate however.

- Utilization of lead and slag in the Slag -
 Though containing so much lead, this is not an unusually bad slag, as is shown by the analyses of average slags given on the next page. These slags are not thrown away; but the

- Rough Lead treated -

lead and silver in them is saved, and their basic character taken advantage of, in Freiberg by smelting them in a reverberatory furnace with poor siliceous silver ores, producing a matte containing lead and silver, which is roasted, and smelted in the blast-furnace

The slag thrown away has only 1.38% Pb. -

	Freiberg ⁽¹⁾	Freiberg ⁽²⁾	Portigband	Ministubuck
SiO ₂ -	28	35.6	38	33
Al ₂ O ₃ -	6	1		3
FeO -	37	38	19.2	54.93
CaO -	7.6	2.4	2.4	0.41
PbO -	7.3	7	6	2.04

- Refining the Lead -

The rough lead from the blast furnace was mingled with matte and slag; it also contained much S, As, Sb, etc. - It was purified by melting in black-lead-crucibles, skimming, and keeping it at a high temperature for two hours; this operation should have been continued much longer, as we found our cupel lead to be very impure, as the process of zincing seems to concentrate other impurities, besides silver, and it involves greater loss of silver to refine this rich argentiferous lead, than the original lead.

The skimmings from the above refining were melted in a crucible at a very high temperature, causing a separation of more ingot lead and all the furnace matte containing Cu.

The copper and impurities were further removed from the ingots of the above two operations, by melting them at a temperature just above the fusion point of lead in an inclined furnace. — The products of this liquation were 1. refined lead and cupriferous skimmings.

The copper was removed from these skimmings and the lead sold, by taking advantage of the greater affinity of sulphur for Cu than for Pb. The above matte and the last skimmings were melted with raw galena in a crucible.

This gave us a lead but slightly cupriferous and a copper matte. — The lead was liquated and added to the refined lead ready for zincing.

The liquation skimmings were analysed and set aside; no further account is taken of the lead in them, because it equals the lead in the added galena. The following tables show the order of these operations.

I. Refining Rough lead = 293 lbs.

Products	Weight	No. of charge	Time per charge	Fuel
A Cuprif Lead	236	3.	2 hours	6 load Blastfurnace
B Skimmings	57			1 load Charcoal

II. Recovering Skimmings B

Products.	Weights
A' Cuprif Lead	12 7/8
C. Furnace Matte	9 7/8

III. *Liquating the Cuprif Lead A + A' = 248 7/8

Products	Weights
D. Refined Lead	201 7/8
F. Cuprif Skimmings	46 3/4

*Rate of Furnace = 55 lbs per hour

IV. Fusion of Skimmings E and matte C with galena.

Products.	Weights	Charge
F. Lead.	42 7/8	Skimmings E matte C
G Cu. matte	20 3/8	Galena (117 lbs)

V. Liquation of poor Lead F

Products	Weights
D'. Refined Lead	32 3/4
H Skimmings	9 7/8

We have as the available products from the blast furnace, - Lead (D + D') from which the silver and gold may be subsequently extracted. - And a Copper matte (G) and skimmings (H) from which we can extract copper and silver by wet processes. -

In the following tables, table 1 shows the amount of valuable metals in these products, Table 2, the relation of these amounts to the amounts entering the furnace, and the

final losses. Table 3 shows how these losses are distributed and gives the loss as fumes.

Table (1)

Products.	Weight in	% Silver	% Gold	% Copper	Pounds Lead.	oz Silver	oz Gold	Pounds Copper
Lead.	23 1/4%	.427	.0075	-	23 1/4%	14.66	0.256	
Matte	20 3/4%	.20	-	2.46	2 1/4%	.59		4.37
Skimmings	9 7/8%	.60	-	6.20	-	.87		.62
Total					2.37	16.12	0.256	5

Products and Table 2 losses in Smelting -

	Silver	Lead	Copper
Percent of Amt changed.	98.71	67.13	83.33
Percentage lost.	1.29	32.87	16.66
	100	100	100.

Table 3 Losses in

	Weights	% Lead.	Wt Lead	% Entire lead ^{lost} changed
Rem Slag.	456	5.00%	22.8	6.10
B. & E "	372	2.04	7.5	2.05
Steep.	19.5	10.60	20.67	6
Fume -			66.03	18.7

The loss of lead as fume is our greatest loss (18.7) and this can probably not be ^{reduced} without adding fume

Chambers to the furnaces. The slag is certainly open to improvements. The loss in steel is accidental, most of it can be recovered; and the quantity would have been the same had the amount smelted been much greater. I therefore come to the conclusion that with the facilities at the Institute and the use of proper care, four ounces of lead can be smelted with about 25% loss, ^(yield) And I think that were we to carefully analyse the products of the smelting works in this country treating similar grade ores very few would show better results. -

At Freiberg the final losses ^{of lead} (including loss in desilverising) are 19% of the assay results. The real loss is said to be even larger.

At Pontigband the loss with similar grade ore is 16% (not counting loss in Desilverising)

The loss in silver is of course nowhere so large. This may be remedied by taking more care in roasting. -

This concludes the account of our work on smelting proper. The silver

was afterwards concentrated by Parkes's process
and removed by cupellation. For the
details of this work I will refer to
Mr Townsend's thesis.

Julius H. Sussman

— May 1876. —