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The Newburyport
Silver Lead Mine
By
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Thesis,

Working of two Classes of
Silver Lead Ore
from the
Merrimac Mining Co's Lode
at
Newbury Mass.

by
W. D. Townsend
Class '76

History and Description of The Newburyport Silver Mine.

It was found by experience in the Mining Labratory of the Institute last year; that, in order to work a Silver Lead ore with any considerable degree of thoroughness, two men must work together on the same ore. On account of the large number of chemical analyses which must be carried on at the same time to check the work. Accordingly when the subjects for Theses were decided upon at the beginning of the year; Mr Susmann and myself fixed on the Newburyport Silver Mine and the treatment of some of its ores, as one likely to give us the most varied experience in the working of Silver Lead ores. Starting with this idea, we visited the mine about the beginning of the school year and were presented with some 3^{d} Grade ore, the treatment of which will

be described hereafter.
We visited the mine again in the spring and got a good idea of what their surface works were to be, at this mine, for dressing the ore, and at the same time inspecting the other lodes in the vicinity and noting their progress.

The history of the discovery and development of this lode, is as follows.
A man named Rogers from the town of Byfield while wandering over the fields in this vicinity, picked up a piece of the float ore and thinking from its weight that it might be valuable carried it to a man named Adams, who, studying up Mineralogy, found it to be a lead ore. Whereupon he bought the land, where it was found, of the owner, Mr Jacques, for $350. After his discovery became known Mr Adams bonded the property to Messrs Kelley & Chipman for $100 000; these gentlemen with Mr Boynton also owned the adjoining lot.
After the mine was proved to be of some value, Mr Chipman sold his

share to a Mr Shaw of Newburyport, who with Dr Kelley bought the mine at its bonded value.
Since then the Boynton & Chipman lodes have been consolodated, and a stock Company formed called the Merrimac Mining Co.
The first piece of float ore was found in 1868 and the first pit opened in May 1874. The first vein discovered was only a few inches wide but it widened rapidly as it went down. It is said to be a limestone vein, surrounded by gneiss, by Mr Patterson, the superintendent. The whole vein formation is stated by him to be 200ft wide, the pay streak, he is working; varying from 18 in to 3 ft in thickness. The limestone forms a very small portion of the vein, but its presence in large quantities in many other lodes in the neighborhood would seem to indicate that this is like the others; about which their is no doubt.
It has been placed geologically in the

Huronian & in the Potzdam series, while it does not seem to be at all clear, that it is in either. The strike of the vein is North 72° East and the dip is vertical to a depth of 50 ft below the surface and then the vein inclines to the South East.
The strike of the outcrop of the country rock is the same as the vein and the dip is North 30° West Some of the ore obtained near the surface was brought to the Institute; and Mr Shockley smelted some of it last year.
In addition to our treatment of the 3rd Grade ore Mr Susman & myself have smelted this year about 750 lbs. of this surface ore, which is about the quality of the 1st Class ore now being brought out of the mine from a depth of 170 ft. The minerals present in the ore are; Galena Pyrite Chalcopyrite Quartz Limonite Hematite Bornite, Blende & Siderite, with traces of other minerals.
The difference in the specimens of Siderite are mentioned in connection with the 3rd Grade ore. Gray Copper should also be named as occurring in considerable quantities in some spots in the vein.

The different assays made of specimens of galena and gray copper give the following results, some of which are from assays made in the Institute last year.

Sample from lot smelted last year by Mr Shockley, 58.32 oz Ag per ton & .58 oz Au.

Picked specimens:	Coarse Galena	102 oz Ag per ton
	Medium Galena	29.16 " "
	Fine Galena	65.9 " "
	Gray Copper	465 " "

Other specimens of gray copper assayed while the mine was being opened, gave values of \$1422 per ton of Ag & \$145 of Au & 4583.93 " " " " & 26.69 of Au but these are exceptionaly fine pieces.

Analyses of the walls of the veins made ~~the~~ in the chemical labratory of the Institute last year gave for the,

South Wall

A Pale yellowish green rock with a S. G. of 2.766, Hardness 2.5, slightly fusible and but little affected by acids. Composition; $SiO_2 = 66.53$, $Al_2O_3 = 25.09$ $Na_2O = .39$, $K_2O = 4.67$, $H_2O = 2.64$, total 99.32

This rock is supposed to be a mixture of quartz and the mineral Agalmatolite

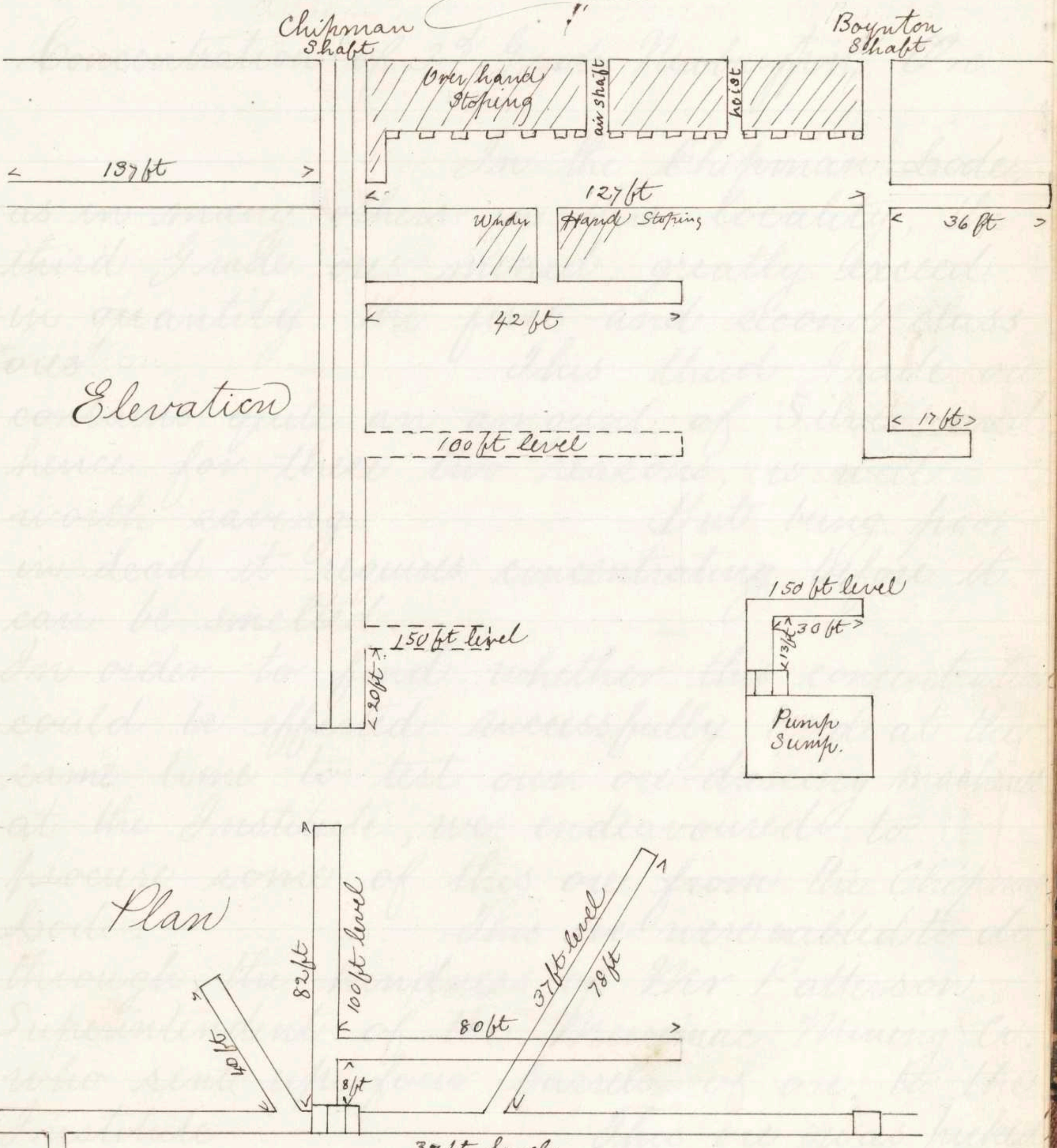
North Wall

A compact fine grained grayish green rock fusible to a black slag. Its S. G. is 2.71 and it seems to consist of three different greenish Minerals not very well determined, but resembling the south wall rock.

The discussion of the other lodes near the Chipman is best deferred until the working of the 3rd Grade Chipman ore is described, as this ore resembles that from the neighboring lodes.

A sketch of the mine, as accurate as could be obtained is given on the next page. The vein leaves the shaft about the 100 ft level, owing to its dip which departs slightly from the vertical. The vein at the 150 ft level is about 3 ft wide and not quite so rich as it was, this condition of things however is liable to change at any time as they go deeper, but whether it will change for the better or worse, no one can tell.

Plan of the Chipman Lode



This is a copy of the best plan of the mine yet made, but is not accurate.

Concentration of 3^{d} Grade Newburyport Ore

In the Chipman Lode, as in many others in this locality, the third Grade ores mined, greatly exceed in quantity the first and second class ores. This third Grade ore contains quite an amount of Silver, and hence for these two reasons, is well worth saving. But being poor in Lead it requires concentrating before it can be smelted.

In order to find whether this concentration could be effected successfully and at the same time to test our ore-dressing machines at the Institute, we endeavoured to procure some of this ore from the Chipman Lode. This we were enabled to do through the kindness of Mr Patterson, Superintendent of the Merrimac Mining Co. who sent up four barrels of ore to the Institute This ore was picked from the dump-heaps near the mine and is a very fair sample of this class of ore, possibly a little richer.

The first process to be gone through was that of cobbing, or breaking up the ore into pieces small enough to go into the Crusher, and at the same time looking for the different minerals.
The principal minerals in the ore were as follows: Galena, Pyrite, Chalcopyrite, Siderite & Quartz; while ~~Zincite~~ Blende & Gray Copper were present in small quantities. The other minerals were in such small quantities that they are not worth mentioning. The Galena occurs in large and small crystals disseminated through the Quartz, often in such small quantities and such fine crystals as to present great difficulties in concentration. Pyrite seems to be present in small quantities compared with the Chalcopyrite which is very abundant.
Siderite occurs in large quantities and varies in color all the way from black to light brown and white.
When black, it has a very dull lustre and is very tough & slightly malleable. It will be brought into notice again when the crushing operation is spoken of.
The Gray Copper occurs partly with a bright lustre and resembling that, ~~to be~~ already

spoken of, in connection with the first class ore; and partly with a dull lustre, being an impure variety, which in some specimens would be likely to be mistaken for the dark siderite.
Siderite and the different pyrites having Specific Gravities so near to that of Galena; the separation of this ore was much more difficult than one consisting principally of quartz and Galena.
The first step in the process was to crush the ore to a suitable size for separation, that is, so small that each mineral exists in particles by itself and not joined onto pieces of another mineral. If the minerals were joined together the specific gravity of the combined mass would become changed and some small particles of rich ore sticking to the gangue will be carried into the tailings. A small loss on this account is unavoidable in ore-dressing.
The ore was crushed by a "Blake Crusher", and then passed through chilled iron rolls onto a twelveth inch sieve.

The rolls are about 8 in. in diameter and 10 in wide. They revolve at the rate of 60 times a minute and their maximum capacity is about one ton in eight hours.
With a higher rate of speed their capacity would undoubtably be greatly increased At present the Blake Crusher works far ahead of them. (Blake Crusher = 3in by 5in) The amount of ore taken was 1662 3/4 lbs, which was passed through the Crusher & Rolls in six hours. The rolls were stopped quite frequently by the dark variety of the carbonate of iron.
This was probably owing to the slipping of the belts in some degree, and then the rolls losing their inertia were stopped by one of these tough pieces coming directly afterwards from the hopper.
The faces of the rolls are of chilled iron which wear away quite regularly if the feed is well regulated.
The rolls are kept in position and the width of the opening regulated by set-screws. Between these set-screws and the rolls are placed iron springs, which allow the rolls a little play when pieces of ore larger than usual are feed in.

The sand (crushed ore) from the rolls, after passing through the seive, was weighed and found to have lost 5½ lbs in weight; probably due to loss by dust and error in the scales.
After an ore has been got into this condition of sand, two courses can be taken with it.
It can be "sorted" first and then "sized" or vice versa
Sorting then sizing consists in first sorting the ore into lots in which grains of equal weight are together that is so that there shall be large particles of light gangue rock and smaller particles of galena, etc. in each lot; the size of both kinds of particles diminishing in each successive lot.
Secondly in seperating the Galena from the gangue by reason of its smaller size by seives or other sizing machines.
Sizing then sorting consists in first sizing by means of seives into lots of equal sized particles of gangue and useful mineral and secondly, by reason of the greatly dissimilar weights of the bodies,

to seperate them by jigs, spitzlutter, and other sorting machines.
The method adopted in our case was the the first mentioned, Sorting then Sizing. The sand was first passed through a series of Spitzlutter one discharging into another.
A Spitzlutte consists of two V shaped boxes, one inside of the other, the space between them being widened or narrowed at will. Water is fed in at the bottom of the V in two jets opposite each other and so placed as to give a rotary effect to the water. Below these can be another jet of water, part of which flows upwards and part downwards and out through a siphon shaped tube, the long end of the siphon being attached to the bottom of the V.
The sand is fed in at the top of one arm of the V in the shape of quick-sand, passes down until it reaches the whirlpool, where the heavier particles sink and the lighter particles pass up the other arm of the V and are discharged into another Spitzlutte to be retreated in the same manner.

The heavier particles on sinking through the whirlpool meet the uprising stream from the single jet of water and all light particles are again weeded out and carried upwards. This single jet is so regulated that a small steady stream of heavy particles are allowed to pass down and out through the siphon.
In a series of Spitzluttes the product from the first one will be large Galena, large quartz & smaller galena. In all the succeeding ones the products will be quartz of a certain size and the next size smaller of Galena, both diminishing in size as the number of Spitzluttes increase.
Hence from these machines, no perfect product will ever be made, as small Galena will always go off with the next larger size of Quartz.
The work these machines are capable of doing is to seperate the slime from the rest of the sand, also clearing out most of the lighter particles.

The first product obtained is well adapted for jigging
One of the chief disadvantages of the Spitzlutte is the frequency with which the outlet pipe for the product is clogged. The sand before it is fed in must be carefully sifted, and even then the pipe will clog. On this account a set of Spitzluttes require constant watching
Now the greatest speed at which these Spitzluttes will work is about 2 lbs a minute, not counting stoppages
In our first trial we found, that when running very carefully and getting a very pure product, we could run only 33 lbs in 32 minutes or about one pound a minute. At our second trial we ran 413 lbs in 3 hrs 57 minutes not counting delays, but taking 5 hrs 45 min. including delays.
These figures show how irregular the working of a Spitzlutte is.
In the system of Machines now in use at the Institute, the set of Spitzluttes have been replaced since our trials by a ~~dolly~~ Cone which has also points of resemblance to the Spitzlutte

This ~~dolly~~ machine consists of a cylinder 4 in in diameter terminating in a cone. from the end of the cone a glass tube leads into a closely stopped bottle At short intervals along the cone and cylinder are set in, pairs of jets, which create a whirlpool from the top to the bottom. The sand is fed in to the middle of the cylinder through a large tube which passes down to about the junction of the cylinder & cone. The heavier particles sink and the lighter ones rise to the top and flow away. The heavier particles are held up by the jet of water introduced into the glass tube as in the principle of the Spitzlutte, When the concentration is good enough the heavier particles are allowed to fall continuously into the bottle. This machine does the same work as the Spitzlutte in weeding out the finest sand & slime, is much more simple, rarely ever clogs and will work 200 lbs an hour if not more.

The feed for both these machines was arranged as follows, A long trough with a hole one inch in diam at one end is put up so that sand coming from this opening goes directly into the Spitzlutte or ~~Dolly~~ Cone. Over the trough is placed lengthwise a rod with a thread cut on it. This rod is revolved by a small leather belt from the main shaft, running over a small wheel on the rod.
On this rod a nut travels up & down to which is attached a stop-cock and rubber tubing. The sand is put on the bottom of the trough to the depth of an inch or more and the nut placed at the end of the trough nearest the hole; the nut & stopcock gradually travels back washing ~~away~~ down the sand as it goes, giving a very perfect automatic feed. Sand has to be fed into the trough about once in 20 minutes.
The product from the Spitzlutte or ~~dolly~~ cone is usually fed onto the jigs. The product from the latest form of ~~dolly~~ cone used in the labratory, seems however to need no retreatment, being in splendid condition for smelting.

In our run with three Spitzluttes, the product from the first one was fed onto the first set of jigs, the second one onto the second set of jigs, the third one and tailings from the third were put together onto the End Bump Table; showing that the third Spitzlutte was unnecessary.
The product fed onto the first set of jigs was much richer than that on the second; as it contained two classes of Galena, the large and the next size smaller, with large quartz; while the second Spitzlutte only contained one size of Galena & the next size larger of quartz.
On this account the first jig products and much richer than the second jig, as will be seen by analyses given hereafter.
These jigs are made like the ordinary form of round bottom jigs.
Each jig is divided lengthwise by a partition extending part way down to the bottom, and divided across by ~~two~~ a partition extending all the way down.
Hence there are four compartments, each left hand one communication with the

opposite right hand one.
In the left hand compartments are plungers and in the right hand ones are sieves. The arrangement for communicating power to this machine was as follows. A cam on the shaft in revolving struck against an iron bow, fastened to the floor so that the top of the bow could move to & fro. To the top of the bow was attached an iron rod which was attached by the other end to a hammer with two ends. The bow being struck first on one side and then on the other, imparted a forward and back motion to the rod and an upward & downward motion to the hammer.
These ends striking the tops of the plungers forced them down suddenly, the plungers being lifted again quietly by a small spring attached to them. The machines being filled with water, the bows are thrown into gear, and the plungers work up & down.
The sand is fed on to the sieves in a steady stream, a strip of iron around the sieves prevents the sand from flowing away until it

has attained a depth of 3/4 in. The tailings from the first sieve flow over the second one, giving the heavier particles a second chance to separate out. The principle of the separation is as follows. When the plunger is forced down suddenly it forces the water up in the adjoining box; this lifts the sand up suddenly. The plunger rising slowly by the force of the spring, allows the sand on the sieve to settle quietly and the particles to arrange themselves according to their Specific Gravity. When the sieve gets full the lighter particles of gangue rock flow over, first onto another sieve then into buckets and are generally thrown away. In working our ore we found that the Jigs worked much faster than the Spitzlutte hence their capacities could not be determined. The rate however would be much increased if there were three sieves consecutively instead of two. The ore could be fed on much faster in proportion

The end sieve cannot be expected to make much of a concentration but merely to act as a check on the others; hence with three sieves, two of them effect good concentrations while with two sieves only one can do much work, giving about two times as much work for 3 sieves as for two.
Our set of jigs in the Labratory consists of two jigs with two sieves each.
In the analyses given farther on the Jigs are numbered I & II; the first sieve in each is A and the second B.
Sand is generally fed onto the Jigs until particles of Galena are seen coming over from the tailings of the second sieve.
The boxes are then thrown out of gear and the upper layer of sand taken from the sieves and retreated.
The sand on the bottom of the sieves is the product. The siftings which pass through the sieves are generally very rich, especially from the two A sieves.
In our ore, owing to the presence of so much siderite and pyrite, the galena could not be obtained as pure by jigging as in an ore of galena & quartz only.
The matters of length of stroke of plungers,

depth of sand on sieves, feed & water supply, vary with every sample of ore & must be found out by actual trial. The quality of the jig products will be given farther on in the table of analyses.

To return now to the product & tailings from Spitzlutte 3.
These were fed onto the End Bump Table, which is merely a large trough suspended by 4 cords and swung to & fro by a cam acting against a bumper fastened to the table at the feed end. The table is moved out gently by the cam and comes back with a bump by its own weight.
The ore is fed on in a steady stream and those particles which are heavy and present the least surface for the action of water stay near the head of the table, the bump balancing the flow downwards of the water.
The feed of sand must be constant or the flow of water will wash down the heavy minerals. The feed of sand is kept up until there is a

layer of it on the table 4 or 5 in thick. The water is let off at the foot of the table through a number of holes which are plugged up as the layer of sand grows deeper. The machine is then stopped and cleaned out. The first 4 or 5 in at the head of the table are quite rich and are saved to be retreated on the side bump table.
The next foot or two is retreated on the end bump and the rest is generally thrown away. This table does excellent work in concentrating the useful mineral, out of a large mass of poor sand, in a speedy manner. These concentrations have to be retreated to render them fit for smelting.
This retreatment we tried to effect in different forms of dolly-tubs with an upward stream of water similar to the cone described previously, but with no success. This failure was owing to the fact that large quartz will always accompany small galena in machines of this character.
An attempt was made to get the sand into shape for these dolly-tubs by sizing through sieves, but the sifting could not be carried far enough to get suitable lots

for these machines. The only result was to get rid of some coarse quartz which did not pass through the sieves. This product was then tried on the side bump table and after two or three failures we succeeded in getting splendid products. The side bump table is a table with a small rim around three sides of it & suspended by four ropes. The open end is lower than the other and the elevation can be changed at will. A Bumper is attached to the middle of the table and is struck by a cam in the same manner as in the end bump table. The two lower end ropes are attached to the ends of a whiffle-tree which is suspended by a single rope from the middle, this enables one to raise and lower the end of the table very easily. Across the head of the table are placed jets of water, which distribute an even layer of water over the whole table. If the table bumps to the left the ore is fed on the

upper right hand corner, just in front of the jets of water, and on the left of the table bumps to the right.
The particles after being fed onto the table would naturally be bumped towards the left hand side of the table and also be carried down by the flow of the water.
But the particles of galena being heavier than the quartz roll down the table more slowly on account of the greater friction. Hence the galena being longer on the table than the quartz would get bumper over farther to the left; and arriving at the bottom of the table would drop into a different division of the trough set to catch the overflow. The quartz going down at quite a slight angle with the sides of the table while the galena makes a large angle.
With an ore consisting of only galena and quartz, and one of this kind has been tried in the Labratory, the line of seperation is very marked When however, pyrite, siderite & similar minerals are present, as was the case ~~which~~ with our ore, one mineral blends into

another, and the lines of seperation are drawn approximately by means of two movable pointers fastened at the bottom of the table. All the sand passing to the right of these pointers is very poor and is thrown away; all passing between them is poor sand containing some galena and must be retreated; that passing to the left of the pointers is very rich galena.
The cause of our first failures was the irregularity of the feed.
When the feed is interrupted even for a few seconds the lighter particles being washed away and the full force of the water acts on the galena, some of which is washed into the tailings before it has a chance to get across the table.
When the feed is perfectly regular the galena is covered by a layer of quartz and is protected from the full force of the water until it gets some little ways across the table. The most regular feed that we could find was a cylinder terminating in a cone, at the

bottom of which was inserted of pair of rotation jets, which kept the mass liquid, the sand was fed onto the table through a rubber tube from the bottom of the cone.
The slimes from all these machines were run into a large settling tub, and by after experiments on other ores we found that they could be be treated very nicely on the side bump table.
Now in any well-regulated Ore dressing Establishment, the sand would pass continuously from one machine to another not stopping until the separation was complete. To do this the size of the machines would have to be proportioned to each other.
This of course we were unable to do in the Labratory, but what we did do was to obtain results as to the kind of work done by each machine and the order in which they should be used in working this ore.
The quality of work done by the machines will best be given by analyses of the different products from the machines.

3rd Class Ore = 9.00 o/o Pb, .65 = Cu, .03 = Ag

Name of Products	Weights	Ag %	Pb %	Cu	Fe	SiO_2	S	Remarks
Jig I A Product.	53 1/2	.085	54.64	.86	16.25	5.00	19.70	Finished Prod.
" I B Product.	49 5/8	.030	14.41	1.13	26.28	16.30	18.38	Retreated
" I A Siftings	25 1/4	.052	33.80	.60	12.10	21.62	12.39	Finished Product
" I B Siftings	7 5/8	.033	3.13	.54	14.00	50.13	5.14	Retreated
" I Tailings	119 1/4	.0265	2.75	.40	12.80	57.97	3.45	Thrown away
Jig II A Product	24 3/8	.040	42.20	.66	16.75	8.00	18.17	Finished Prod.
" II B Product	26	.020	9.51	.58	25.53	12.74	16.47	Retreated
" II A Siftings	29 1/8	.0355	24.58	.40	13.01	23.26	13.98	Finished Prod.
" II B Siftings	5	.043	6.99	.64	19.30	36.03	8.60	Retreated
" II Tailings	134 1/4	.010	2.54	.50	16.01	54.92	4.27	Thrown away
Spitzlutte 3 Product	188	.040	6.84	.58	12.40	52.00	4.91	Retreated
Product End Bump Table	333 7/8	.039	12.03	.33	15.22	41.94	6.52	Retreated
Tailings " "	575 5/8	.285	1.15	.35	9.04	68.71	2.00	Thrown away
Slime	56 1/2	.46	6.88	.46	12.50	50.50	3.91	Kept for future treatment.

total = 1628 1/8 lbs
Sample of original ore taken = 37 lbs
1655 1/8 lbs

Weight started with = 1657 5/8

This weight obtained would account for nearly all of the original ore started with.

These analyses show the comparative results given by the different machines.

The Silver was determined by assay.

The lead was weighed as sulphide & sulphate; the chief objection to the use of the sulphate

method seems to be that iron is apt to come down with the lead.
Apart from that objection, the sulphate is the more accurate method of weighing.
The iron can be seperated by precipitating the lead as sulphide, dissolving in dilute HNO_3 and reprecipitating as sulphate.
The copper is best precipitated on the positive pole of a battery. In this ore the Sb & As are apt to come down at the same time. In case they do, the best method of procedure is to dissolve the Cu, Sb & As off of the platinum electrode by HNO_3, neutralize by Ammonia-water and let the solution stand in a warm place. The As & Sb will precipitate as hydrates & can be filtered off; the Cu can then be thrown down again by the battery. To get all the Sb & As down this way & determine them requires too much time but I think it could be done. This method of seperating Cu & Sb & As I found out while experimenting on the color method for determining Cu.
The iron in the ore was determined volumetrically and we got very accurate results where Cu As & Sb were first carefully separated.
All the analyses in the wet processes and smelting were done conjointly by Mr Susmann & myself.
The following is a table of wet processes gone though by us.

Table of Processes
Concentration of 3rd Class Ore.

- Original Ore 1657 5/8
  - Sample 37 lbs
  - Ore 1628 lbs Run through Spitzlutte 1
    - Product Run over Jig 1
      - Jig I A Product (K)
      - Jig I B Product Rejigged
        - Tailings (T)
        - Product (K)
      - Jig I A Siftings (K)
      - Jig I B Siftings Side Bump Table
        - Tailings (T)
        - Product (K)
    - Tailings Spitzlutte 2.
      - Product Over Jig II
        - Jig II A Product (K)
        - Jig II B Product rejigged
          - Tailings (T)
          - Product (K)
        - Jig II A Siftings (K)
        - Jig II B Siftings over side bump table
          - Tailings (T)
          - Product (K)
      - Tailings Spitzlutte 3
        - Tailings End Bump Table
          - Tailings (T)
          - Product Side Bump Table
            - Tailings (T)
            - Middlings
              - Tailings (T)
              - Product (K)
        - Product Side Bump Table
          - Tailings (T)
          - Product Side Bump Table
            - Tailings (T)
            - Middlings Side Bump Table
              - Tailings (T)
              - Product (K)
            - Product (K)

Slime from all the machines (K)

The rich products marked (K), except the slime, were put together and gave a general product weighing 233 lbs and analyzing 46.02 % Pb and .115 % Ag. Those marked (T) were too poor to save & were thrown away. The rich concentration contained 73 % of the original Lead in the ore and 54 % of the Silver. The slime contained 10 % of

the Silver in the ore, showing how important it is to have settling tubs in treating this ore
After the ore had been run over the machines once, analyses of the products were made.
Some products could be thrown away at once & some were rich enough for smelting while others had to be retreated as being neither one or the other
Jig I B Product, II B Product and the coarse part of Spitzlutte 3 product were rejigged.
Jig I B. Siftings, II B. Siftings, the fine part of Spitzlutte 3 Product & Product End Bump table were retreated with success on the side bump table.
The tailings in each case were thrown away & the products put into the total concentration.
The works being put up at the mine are jigs, drums with spirals to separate the fine from the coarse ore, and slime tables.
Their efficiency remains yet to be proved.
Most of the mines near the Chipman lode are taking out ore, corresponding to this 3^d Class Chipman, hence these ore-dressing machines must have an extensive use in this locality.
With many of the mines, the success or failure of the mine, will depend on the economy with which the Galena can be separated from limestone and siderite

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Smelting of Float Ore from Newburyport.

The ore was some obtained from Newburyport last year when the mine was first opened. The minerals in it are given in the general description of the mine. The total amount taken was 768¼ lbs; a sample was taken from the crushed ore and gave the results given in the table. The large quantity of As & Sb contained in the ore as shown by the analysis is a great obstacle to the successful treatment of the ore. The Silver being mostly chemically combined with the As & Sb; when in roasting or smelting the ore the As & Sb are volatilized and carry off Silver with them.

Pb =	44.29
Cu =	.70
As =	.78
Sb =	1.14
Fe =	12.02
Zn =	.73
S =	16.88
Al_2O_3 =	.84
SiO_2 =	12.33
	89.71
C & O etc	10.29
	100.00

Again in refining the Zinciferous Lead the As & Sb require four or five days heating before they volatize, which causes a loss of Silver. A Summary of operations is given on the next page

Summary of the Float Ore Treatment.

Float Ore - roasted 768 1/4 lbs | Roast Ore 702 1/4 lbs - agglomerated | aggl.'d Ore - 650 3/4 lbs - Smelted in blast furnace with fluxes gave

Crude Lead = 293 lbs Pb
Steep, containing = 21 " "
Run Slag " = 22 " "
Beginning & End Slag = 9.7 " "

Most of the Pb in the steep was in the form of Galena which was added to the ore as a flux.

Crude Pb 293 lbs Refined in crucibles

- Skimmings recovered in crucibles
  - ~~Cu~~ Lead Matte 9 7/8 lbs (added to a.)
  - Skimmings (K.)
  - Lead 12 5/8 lbs added to d
- (a) Skimmings - 46 3/4 Pb recovered by smelting with 11 5/8 lbs galena & the Cu matte added to get all the Cu together.
  - Copper Matte (K) 10 5/8 lbs.
  - Pb - Liquated 42 7/8 lbs
    - Skimmings (K)
    - Lead 32 1/4 lbs added to b
- (d) Pb + Pb 236 lbs + 12 5/8 lbs = 248 5/8 lbs Liquated to remove Cu etc
- Pb + Pb 206 3/8 + 32 1/4 = 238 5/8 Zinced & Liquated
  - Pb. Zinced again & liquated
    - Pb. Zinced again & liquated
      - Zinciferous (K) lead 153 5/8 lbs
      - (Z) Skimmings 10 5/8
    - (Y) Skimmings 15 1/8 lbs
  - (X) Skimmings 70 1/2 lbs

Skimmings (X) (Y) (Z) - Refined from As Sb & Zn in crucibles

- Scum (K) As Sb & Pb
- Argentiferous Lead - cupelled
  - Ag (K) 230 grms.
  - Litharge (K)

(K) = Kept for future treatment

This reaction will not take place in the presence of so much gangue as we had in our ore. The method of treatment of this ore was to agglomerate the ore after roasting; thus forming a rich lead slag which is run down in the blast furnace. The object of roasting the ore in this case was to diminish the amount of fluxes to be used in the blast furnace. As the sulphur combined with the lead has to be provided with some other element to take the place of the lead, in order that the lead may be set free.

The roasting was carried on in two reverbratory furnaces at the same time. Three different ores were roasted in these furnaces one after the other. This was done to economise time and fuel. The furnaces requiring a long time & much fuel before they become heated enough to receive the first charge.

The amount of charge, the time & the students name who attended each charge will be found in the table on the next page.

Table of details of roasting Ore

Ore	Fuel	Ch'ge in	Ch'ge out	Time Interval	Furnace	Name
85 lbs	58 5/8 lbs	10.40	2-40	4-00	Long Furnace	Mr Townsend
85 "	64 7/8 "	10.30	2-35	4-05	Hollow Bed "	" Susman
85 "	29 1/4 "	2.45	6-00	3-15	Long "	" Schwarz
85 "	50 1/4 "	2.55	6-00	3-05	Hollow Bed "	" Robinson
85 "	43 1/4 "	6.05	9-55	2-50	Hollow Bed "	" Townsend
85 "	46 1/4 "	6.07	9-55	2-48	Long "	" Susman
85 "	39 3/8 "	10.10	1-55	3-45	Long "	" Gould
85 "	73 3/4 "	10.07	2-03	3-56	Hollow Bed "	" Allen
88 1/4 "	46 1/4	2.15	6.00	3-45	Hollow Bed "	" Fletcher
768 1/4 "	451 7/8			31 hrs 29 m		

The furnace getting hotter as the roasting continued, less coal was used.
The difference in the amounts of coal used in each charge, is accounted for by the fact that at the end of some charges the fire was left very low & in others very good.
The Sulphur in the ore was reduced from 16.88 %o to 5.65 %o.
The agglomeration was done in the hollow-bed furnace. Much larger charges could be put in, than in roasting, and as the heat is raised as quickly as possible was much more quickly done.

Details of the Treatment of Float Ore

The ore was crushed for smelting by a Blake Crusher and chilled iron rolls, the same that were used for the 3rd Grade ore. The object of crushing the ore up fine is to enable the heat and air to get at each particle while the ore is being roasted to drive off the Sulphur. This ore having only 44 o/o Pb in it, has to be run down in a blast furnace. Ores containing 70 o/o Pb can be treated almost entirely in the reverberatory furnace by what is known as the Flintshire Process. In this process the ore is crushed and roasted; after it has been roasted until no more sulphur fumes appear, the heat is raised considerably; the sulphates and oxides which have been formed bye the roasting and sulphides which have been added react on each other and form metallic lead. Reactions are partly these.

$PbSO_4 + PbS = 2Pb + 2SO_2$ & $2PbO + 2PbS = SO_2 + Pb_4S$

& $PbSO_4 + C_2 = Pb + SO_2 + 2CO$ and others similar.

The agglomeration of the ore is for the purpose of making the ore solid & able to resist the weight put upon it in the blast furnace.

Table of Details of Agglomeration.

Time		Weight	Time Interval	Remarks
3-55	Fire lit			
4-30				Flame strong.
5-40	Crucible Furnace fire started			This makes a great increase in the draught
7-15				Strong Roasting Heat.
7-35		lbs.		Slag covering parts of hearth begin to melt & simmer.
11-05	Charge in	181 1/4	hrs min	Stirred every 10 m. & found it too often it chilled the furnace down.
12-40	Charge out		1-35	
12-45	Charge in	181 1/8		Stirred every 15 m. Fusion
2-00	Charge out		1-15	good better than last.
2-20	Charge in	181		
3-40	Charge out		1-20	Fusion good.
3-50	Charge in	158 7/8		
5-00	Charge out		1-10	No free Silica visible,
5.40	Furnace cleaned			fusion good.
	total	702 1/4	5 - 20m	Agglomeration good.

A Partial analysis of the ore after agglomeration gave Pb = 49.10, Cu = .85, S = 3.00

Its weight was 650 3/4 lbs.

The ore was then ready for the blast furnace. This blast furnace is really a Spanish hearth furnace. It has three tuyers 3/4 in in diam, these were supplied with air from a small Sturtevant blower and gave a pressure of 8 oz. The bottom of the furnace was rammed with steep, and was hollowed out a little to give the lead a place to collect in. From this depression a small channel was cut in the steep leading to large basin in front of the furnace. This basine or pot was made of bricks and lined with steep. It was so arranged than the lead & slag flowing into it, being kept liquid by a covering of charcoal, seperated from each other. The lead could be tapped off from the bottom and the slag overflowed into buggies at the top. This makes a very good seperation.

The charge put into the furnace was calculated for a tri-basic slag; and was of the following composition

Ore ---- 66.20

Limestone --- 8.40

Puddle Cinder 25.40 .... 100.00 %

Partial analyses of the Ore, limestone & puddle used gave

	Aggl'd Ore	Puddle Cinder	Limestone
SiO_2	14.45	17.00	.60
FeO	18.28	73.00	
Al_2O_3	1.00	6.2	
S	3.00	1.9	
CaO			55.60

The charge thus calculated should have given a slag of:

$FeO = 55$

$SiO_2 = 29.50$

$Al_2O_3 = 4.68$

$CaO = 10.86$

And in addition a matte of 50 lbs of sulphide of Pb Cu etc.

The actual slag obtained had the this composition →

Slag obtained

$SiO_2 = 31.23$

$Al_2O_3 = 8.91$

$CaO = 8.48$

$MgO = 1.10$

$FeO = 43.44$

$PbO = 5.40$

$ZnO = 1.88$

$P_2O_5 = .35$

$S = 1.22$

And only 9 lbs of matte were obtained, the lead going into the slag. This ~~slag~~ composition was probably owing to the action of the melted slag & metal on the sides of the furnace & to the Galena which was added to each charge (3 lbs) to prevent the formation of "bears" in the furnace

This addition of so much Sulphur seems to have changed greatly the composition of the slag. The analysis of the slag given the page before, adds up above 100 op owing to the impossibility of knowing how the Sulphur & Oxygen are combined with the other elements. The slag was very liquid and flowed easily and was bad only by reason of its containing so much Lead. The blast furnace run was begun by the charging of 240 lbs of Revere Copper Slag to get the furnace running well. After the ore was all run through the furnace was washed out with 80 lbs of the same. The Details of the blast furnace run are given in the following pages, and explain themselves.

Blast Furnace
Charging Door.

Time	Time Interval	Charge	Height of charge in Furnace	Remarks.
11.41	6	Charge IV	3 3/4 ft	Rise of 1/2 ft in furnace
11.52	11	" IV	3 3/4 "	" " " " " "
12.00	8	" IV	3 3/4 "	Rise = 3/4 ft.
12.11	11	" IV	4 "	" = 3/4 " .
12.23	12	" IV	4 "	Slight flame before charging
12.37	14	" IV	3 3/4 "	
12.47	10	Charge V	3 3/4 "	
12.59	12	" V	3 1/2 "	Coke began to build
1.10	11	Charge VI	3 1/4 "	up in furnace so the
1.20	10	Charge VII	3 1/4 "	charge was changed
1.37	17	" VII	2 1/2 "	gradually diminishing
1.58	11	" VII	1 3/4 "	the amount of coke
2.12	14	" VII	1 1/4 "	Blast off.

Ratio of Ore & Fluxes in Charges.

Charge	Ch'ge I lbs	II lbs	III lbs	IV lbs	V lbs	VI lbs	VII lbs	Remarks
Ore			30	40	40	16 5/8		
Puddle Cinder			11 1/2	15 3/8	15 3/8	6 1/4		The Galena
Limestone			3 3/4	5	5	2		used was
Galena			3	3	3	1 1/5		some from
Coke	10	8	8	10	8	3 1/5	8	Burleigh
Revere Slag	30	30					40	Tunnel, being
Ratio of Coke to Charge	1:4	1:5	1:7	1:7	1:9	1:8	1:6	worked by Mr James.

Blast Furnace
Charging Door

Time	Time Interval	Charge	Height of charge in Furnace	Remarks
March 15 5 P.M		2 hods Charcoal 4 " coke	1 ft	Started to warm up furnace
March 16 8.40		2 hods Coke	1½ ft	Blast on.
8.58		3 " "	2 "	
9.08		1 " "	2 "	Stopped blast 3 m. to oil governor.
9.24	m	Charge I	2½"	For composition of charges
9.37	13	" I	2¾"	see page
9.43	6	" I	2¾"	Flame still appears at chg door.
9.50	7	" I	2⅞"	
10.02	12	Charge II	2¼"	
10.09	7	" II	2¼ "	Gas takes fire shortly
10.15	6	" II	2½ "	after charging.
10.19	4	" II	2½"	"
10.30	11	Charge III	2¼"	Amount of coke
10.35	5	" III	2⅓ "	diminished once more
10.40	5	" III	2½ "	as it was building up
10.43	3	" III	2½"	in the furnace.
10.48	5	" III	3"	No Flame
10.55	7	" III	3¼"	"
11.00	5	Charge V	3¾"	"
11.09	9	" V	4"	" "
11.27	18	Charge IV	3½"	
11.35	8	" IV	3¾"	Rise, when the charge was fed in, of 1/2 ft in height.

Blast Furnace
Tap Hole

Time	Number of Buggie	Remarks.
8.35		Slag appears
9.55		Slag filled pot & partly chilled on top
10.05		Slag flowing freely
10.10	I	" " ", took crust off of top.
10.25	II	Sheet iron chimney put over top of pot to prevent it from chilling
10.40	III	Mostly Revere Slag.
10.50	IV	Ore beginning to report by a smell of Arsenic.
11.00	V	Well settled no lead or matte. Pb fumes
11.10		Blast stopped 2 min. owing to an accident.
11.13	VI	Slag from Ore, well settled, no Pb or matte.
11.25	VII	Pb appeared running into the pot, Slag very liquid
11.30	VIII	Transition between fine & coarse slag
11.33		Tapped Pb. from bottom of pot into ingot mould.
11.35	IX	Fine slag, well settled.
11.42	X	" " very liquid.
11.45	XI	Heavy Pb, As & Sb fumes gathering on the charcoal.
11.50	XII	Slag flowing freely, fine grained.
12.00	XIII	Tapped lead from pot.
12.03	XIV	Shut off blast & raked coal off of the pot.
12.08		Blast started.
12.11		Slag flowing again.
12.14	XV	Slag, without matte or metal, heavy & fine grained.

Blast Furnace
Tap Hole

Time	Numbers of Buggie	Remarks.
12.19	XVI	Slag fine grained, without matte or metal
12.23	XVII	" " " " "
12.30	XVIII	" " " " "
12.35		Tapped Lead from pot.
12.36	XIX	Same as XVI
12.45	XX	" " "
12.50	XXI	" " Blast = 8 oz.
12.55	XXII	
1.00	XXIII	Tapped Lead from pot.
1.05	XXIV	Slag a little coarser grained.
1.15	XXV	Slag same as XVI.
1.20	XXVI	" " "
1.25	XXVII	" " "
1.30	XXVIII	Tapped Lead from pot.
1.33	XXIX	Slag a little coarser grained
1.35	XXX	Slag coarse grained.
1.43	XXXI	" " .
1.50	XXXII	" "
2.04		Slag almost stopped flowing
2.12	XXXIII	Blast off, Slag coarse grained & some matte.
2.13	XXXIV	Tapped metal. Slag coarse grained, no matte or metal.

The metallic lead obtained by the blast furnace run was 293 lbs which was the lot subsequently treated.
The rest of the lead went into the slag and into the steep and among the bricks in the sides of the furnace.
That which went into the steep was chiefly galena, probably the same as was added in the charges.
A good deal of the lead which crept in between the bricks of the furnace was changed to silicate of lead.
All this lead in the slag & steep was thrown away as it would cost too much to extract it.
To get a sample of the run slag for analysis, a number of buggies were taken from the middle of the run; after the Revere slag had been washed out by the regular charges, crushed and sampled, the analyses has been stated before. The rest of the slag, called Beginning & End Slag, was analyzed for lead alone, to find how much lead was lost in it.

The crude lead was melted in crucibles under charcoal to refine it. It was kept for a few hours at a dull red heat, the scum taken off & the lead poured into ingots moulds. We found afterwards that the lead was not refined enough and should have been heated much longer at a higher heat to drive off the As & Sb in the lead.
The skimmings contained much of the copper which came from the ore. They were remelted in a crucible and skimmed, and then poured into a buggie and allowed to settle.
Three products were obtained, the second skimmings, which have not been retreated, and in this connection it may be well to say, that many of the skimmings and secondary products could not be retreated.
Both on account of lack of time and because of their small quantities These objections would be obviated in large works running continuously.
The second product from the skimmings was matte containing lead & copper. This was added to the liquation skimmings

as will be seen by refering to the general table of Processes, to be recovered.
The third product was ingot lead which was added to the original ingot lead
The ingot lead was liquated to seperate the copper; this consists in heating the ingots on a piece of sheet iron and letting them gradually melt, the lead melting more easily than the copper runs off and leaves the copper behind as a scum containing some lead.
These skimmings are smelted in a crucible to recover the lead, as follows.
The maximum amount of copper present in the lot is calculated and enough Galena is added to give Sulphur enough to combine with all the Copper.
This it will do, as Cu has a greater affinity for Sulphur than lead.
The other lot of Cu matte formed was also added to bring it all together.
After the crucible had been brought to a bright red heat, it was poured into a buggie and gave as products metallic lead & copper matte which was stored

away for future treatment.
The lead was liquated, and gave skimmings which were also stored away, and lead which was added to the original lot of liquated lead. (see table).
This liquated lead was now ready for treatment to extract the silver.
The two best processes for extracting Silver from lead, stated briefly are as follows.
Pattinson's Process. The lead & silver alloy is heated up to melting and allowed to cool slowly; pure lead crystallizes before the alloy of Ag + Pb does; these crystals are skimmed out of the liquid mass and transferred to another pot where the process is repeated, the same is done with the melted alloy left.
Till gradually one lot is so impoverished that the Silver can be disregarded and the other becomes so rich that it will pay to cupel directly.
Park's Process. When Zinc is added to a melted alloy of Pb + Ag it combines with the Ag, and the zinc alloy can be seperated from the Pb by skimming or by liquation; the last was the way we used. The zinc is then distilled off and the lead & silver left is cupelled.

The Park's Process was chosen by us, because it is much more easily worked with small lots than the Pattenson.
The amount of Zn to be added differs according to the different writers consulted. We added 13 o/o of the amount of lead, but since then I have been brought to believe by experiment etc that 3 o/o would do as well. Our lot of lead was melted and about 1/2 the whole amount of Zinc was added and stirred in well. The alloy was then ladled into ingot moulds and then liquated giving rich skimmings and comparatively poor lead. This was again zinced & liquated giving a second lot of skimmings and again a third time. Leaving a lead containing only .001 o/o Ag and 1.30 o/o Zn & otherwise pure Lead, this was stored for future treatment. In large works the third skimmings contain much free Zinc and are added to the next lot of lead to be zinced instead of pure Zn, they answer the purpose very well.
In our case by adding such an excess of

Zinc, the Silver was contained in a much larger amount of alloy of Zn Pb & Ag than was necessary. This alloy in large works is heated in retorts and the Zinc is saved and used again, but as we had no suitable apparatus we were compelled to drive the Zn up the chimney. The silver-lead alloy left was now considered ready to cupel. This operation consists in heating the alloy in an oxidizing flame and changing ~~the~~ the Pb to Lithage and either absorbing this lithage or pouring it off so as to leave the Ag in the metallic state.
This is done by placing the alloy on a cupel or hollowed mould made of boneash, limestone & fireclay or some such mixture which will gradually absorb lithage, and heating it in a small reverbiatory furnace. The lead is oxidized by a blast of air blown directly upon the melted alloy; when a little lithage has collected it is poured off through a little channel cut in the lead. The lithage is then saved to be recovered again.
The cupels we tried were made of 68 parts limestone and 32 of fireclay.

Thinking that our argentiferous lead was was pure enough to cupel, we started cupelling with the first run given below

1st Cupel Furnace Run

Time	Charge	Remarks
6.15		Fire lit.
11.20	2 3/8 lbs	Shockleys Lead to test the cupel
11.21		Driving & spattering a little.
11.33		Blast on.
11.59	2 pigs	about 6 lbs. Drive at once
12.35	" "	Drive in 3 minutes.
1.03	" "	Drive very slowly
2.00	" "	Drive after 3/4 hours strong heat
2.15		Stopped working.

But we found the alloy was very impure & coated over with scum the minute the blast touched it. After a few pigs had been put on it was found useless to process. The next day the (ag) lead was put into crucibles and refined at a high heat. Arsenic & Antimony fumes came off in great abundance. We refined until these fumes ceased coming off. This took us four days using five hods of anthracite coal per day. In practice the Ag lead would be refined in a covered furnace by poling with green poles, where nothing would be lost by spattering. But even then it would be a long process. After the lead was refined we again tryed to cupel it. See tables next page

2nd Cupel Furnace Run

Time	Charge	Remarks
7.40		Fire lit for draught in small furnace
8.05		Fire lit in cupel furnace
11.30		Intense heat in " " .
1.10		Skockleys lead. Blast on.
1.40	5 pigs	2 small & 3 large drive at once.
2.25	2 "	Large ones, Drive at Once
3.00	" "	Drive at once.
3.35	" "	" " "
4.00	3 "	1 Small & two large.
4.30	3 "	Drive in two min.
4.57	3 "	Drive in 10 min. Furnace cooled down
5.17		Hole appeared in cupel the lead which remained was blicked

3rd Cupel Furnace Run.

Time	Charge	Remarks
5.55		Small fire lit for draught
6.00		Fire lit in Cupel Furnace
11.00	1 lb Pb	Shockleys lead. Blast on
11.05	2 pigs	Drive at once.
11.35	" "	" " " .
12.00	" "	" " " .
12.25	1 "	" " " .
1.30		The lithage ate a hole lengthwise into the tap hole and some lead ran into the pot, the rest was blicked
		The remaining Pb(Ag) weighed 7 1/2 lbs.

These cupels giving out this way showed that something was the matter with composition. It seems to me that we used to much limestone. When heated it loses 44 % of its weight. This loss probably makes the cupel so porous that the lead gradually eats its way through. The best way of making cupels seems to be known only at the large smelting works.
The remaining Ag. lead was cupelled in a muffle furnace in small bone-ash cupels, which worked very well.

The Silver obtained weighed 250 grms, but part of it was not entirely free from lead. One button ate so deeply in the cupel that it was impossible to pour off the lithage from it and it did not quite blick.
The gold was not seperated as the term had ended. For further figures as to the amounts and per cents lost in the different processes I refer to Mr Susmann's Thesis. All our work in the metallurgical labratory was checked by analyses and assays, which showed us where our losses had been.
We suffered from not having dust chambers to catch the lead coming from the furnaces. We also laboured under the disadvantage of working with small quantities and in having such small amounts of matte & skimmings that it would not pay to work. But though our results were not so good as might be desired, we learned by our mistakes what to avoid in the future

Finis.