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Report  
On the working for Silver and Gold  
of a

Middle Grade Product  
from

One of the Merrimac Mine; Newburyport.

by.

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The Merrimac Ore has been worked heretofore at the Institute, as elsewhere, by separating it into two products, a smelting ore and a waste product, and the latter has been found to contain a good deal of Galena, Zinc Blende, Tetrahedrite &c, which <sup>carry</sup> ~~contains~~ more or less silver, & the object aimed at this year was to concentrate the ore in such a way as to make a second or middle grade product which should contain the Blende, Tetrahedrite, Pyrite, Siderite &c, and to work that product for the silver by several different methods and compare the results, to see whether it would be possible to work such a product on a large scale, & if so, which of the methods experimented upon would be the most advantageous to use. It was thought that the product would contain a large proportion of Zinc Blende, and based



on that supposition, it was intended to try the following general methods, and such others as we might be able to devise after finding out exactly what the ore did contain.

1. Raw Amalgamation.
2. Chlorination and Amalgamation.
3. Chlorination and solution.
4. Zievogel's process.
5. Distillation and solution.
6. Distillation and Amalgamation.

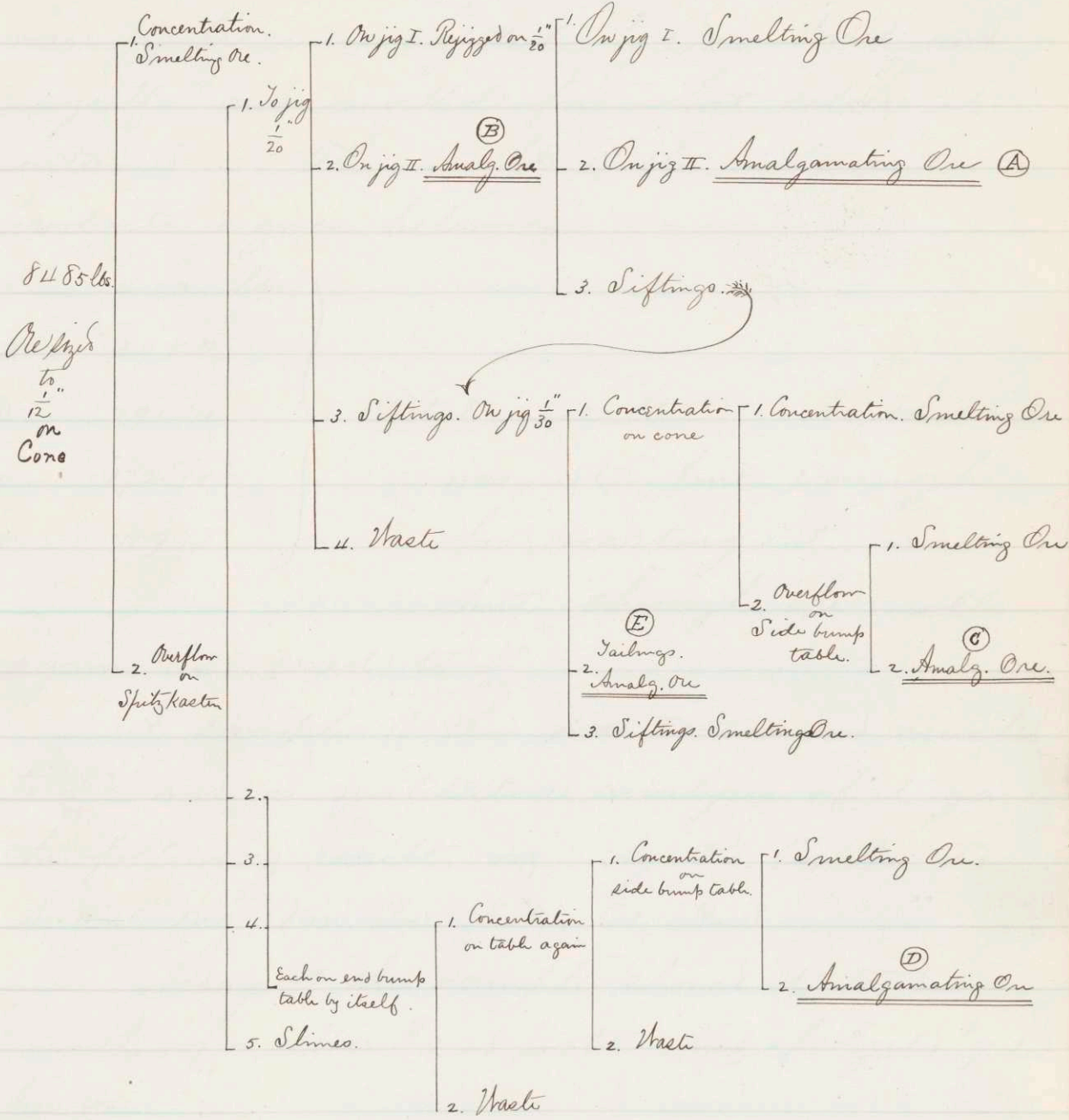
When the ore had been analyzed several of these methods were found to be impracticable. Instead of a large amount of Blende, it was found to contain only a small per cent. of that mineral, and that the greater part of it was Siderite ( $\text{FeCO}_3$ ) which had perhaps been mistaken, at first sight, for Blende in the products formerly obtained.

The Ore as it came to us from the Mine, did not present a very promising

appearance, and if it were a good sample of the material that they were getting from the mine it was not very flattering to their prospects. But after visiting the Mine, I can say that the ore sent to us was rather a poor specimen of what the Mine affords.

The weight of ore that we started with is not known, but after crushing and sampling it weighed 8485 lbs. This was broken up and sampled for minerals and the following species were recognized - Siderite, Quartz, Chalcopyrite, Pyrite, Galena, Dolomite, Chlorite, Blende (much of which was "Black Jack"), Serpentine, Arsenopyrite, and Tetrahedrite, but this last was rather scarce. The ore was then put through a Blake Crusher and afterwards through rolls and sized through a sieve of  $\frac{1}{16}$ ". It was then ready for concentration, and the method used in washing was as follows:-





The products A, B, C, D & E were called "Amalgamating Ore" for want of a better name, and they

constituted the middle grade product which was to be investigated by Mr. Wood and myself, and worked for what silver it contained. The weight of the different products is given below:—

A — 413 lbs.  
 B — 375 <sup>3</sup>/<sub>4</sub>  
 C — 191 <sup>1</sup>/<sub>8</sub>  
 D — 172 <sup>5</sup>/<sub>8</sub>  
 E — 119 <sup>1</sup>/<sub>4</sub>

Total 1272 lbs. As this was not fine enough for roasting it was put through the rolls

again and sized to  $\frac{1}{40}$ "

A sample of this was taken and crushed to  $\frac{1}{60}$ " and a qualitative analysis of it gave the following result viz. Ag. Pb. Cu. Au. Fe. As. Ca. Mg. Mn. Zn. SiO<sub>2</sub>. CO<sub>2</sub>. & S.

Assays of this sample showed it to be worth for silver \$11.28 per ton, and for gold \$9.23 per ton.

Quantitative Analysis gave the percentage composition as shown in the table:—

|  |       |       | Duplicatus |       |
|--|-------|-------|------------|-------|
| CO <sub>2</sub>                              | 14.98 | 14.74 | 15.12      | 15.06 |
| Fe   | 24.84 | 24.84 | 24.82      |       |
| O (combined with Fe in FeO.CO <sub>2</sub> ) | 3.92  |       |            |       |
| SiO <sub>2</sub>                             | 21.09 | 21.08 | 21.11      |       |
| Cu   | .68   | .68   | .62        |       |
| As   | 1.33  | 1.29  | 1.37       |       |
| Ag   | .03   | .03   | .032       |       |
| Au   | .0016 | .0016 | .0016      |       |
| Zn   | 1.83  | 1.59  | 1.83       |       |
| MnO  | 1.04  | .82   | 1.04       |       |
| CaO  | .83   | .88   | .78        |       |
| MgO  | 3.13  | 3.11  | 3.15       |       |
| Pb   | 10.01 | 11.19 | 10.01      | 8.82  |
| S  | 14.56 | 14.56 |            |       |
| H <sub>2</sub> O                             | .52   |       |            |       |

Total 98.7916

The calculation of the per cent of minerals was based on all the As being there as Arsenopyrite - the Cu as Chalcopyrite - the Zn as Blende, and the Pb as Galena.

for calculations see table on next page:-



Arsenopyrite As 1.33 = Fe .994 = S .566  
 Chalcopyrite Cu .68 = Fe .599 = S .686  
 Blende Zn 1.83 = S .91  
 Galena Pb 10.01 = S 1.549

S 3.711

Pyrite =  $\frac{Fe\ 9.527}{14.56\ \text{total S}} = 10.85\ S\ \text{remaining.}$

Fe 11.12

Siderite =  $\frac{CO_2\ 10.72}{14.98\ \text{total } CO_2} = 13.72\ Fe\ \text{remaining.}$

4.26 CO<sub>2</sub> remaining.

1.04 MnO = .645 CO<sub>2</sub>  
 .83 CaO = .65 CO<sub>2</sub>

2.975 CO<sub>2</sub> remaining = 2.704 MgO  
 3.13 total MgO

Serpentine = .13 H<sub>2</sub>O = .436 SiO<sub>2</sub> = .426 MgO remaining  
 21.09 total SiO<sub>2</sub>  
 20.654 SiO<sub>2</sub> remaining = Quartz.

|              | As   | Fe    | S     | CO <sub>2</sub> | O    | MnO  | MgO   | CaO | SiO <sub>2</sub> | H <sub>2</sub> O | Cu  | Zn   | Pb    | Ag  | Total % |
|--------------|------|-------|-------|-----------------|------|------|-------|-----|------------------|------------------|-----|------|-------|-----|---------|
| Mispickel    | 1.33 | .994  | .566  |                 |      |      |       |     |                  |                  |     |      |       |     | 2.89    |
| Chalcopyrite |      | .599  | .686  |                 |      |      |       |     |                  |                  | .68 |      |       |     | 1.965   |
| Blende       |      |       | .91   |                 |      |      |       |     |                  |                  |     | 1.83 |       |     | 2.74    |
| Galena       |      |       |       |                 |      |      |       |     |                  |                  |     |      | 10.01 | ?   | 11.589  |
| Pyrite       |      | 9.527 | 10.85 |                 |      |      |       |     |                  |                  |     |      |       |     | 20.377  |
|              |      |       |       | .645            |      | 1.04 |       |     |                  |                  |     |      |       |     |         |
|              |      |       |       | .65             |      |      |       | .83 |                  |                  |     |      |       |     |         |
| Siderite     |      | 13.72 |       | 10.72           | 3.92 |      | 2.704 |     |                  |                  |     |      |       |     | 37.204  |
|              |      |       |       | 2.975           |      |      |       |     |                  |                  |     |      |       |     |         |
| Serpentine   |      |       |       |                 |      |      | .426  |     | .436             | .13              |     |      |       |     |         |
| Quartz       |      |       |       |                 |      |      |       |     | 20.654           |                  |     |      |       |     |         |
| Total        | 1.33 | 24.84 | 14.56 | 14.99           | 3.92 | 1.04 | 3.13  | .83 | 21.09            | .13              | .68 | 1.83 | 10.01 | .03 | 98.411  |

H<sub>2</sub>O unaccounted for .39  
 98.801



Before the analysis of the ore had been made, a number of methods had been proposed, by which to work it, and the following were assigned to me.

- I. Chlorinating roasting, followed by Amalgamation & distillation of the mercury and amalgam.
- II. Chlorinating roasting, and lixiviation with brine saturated with Cl. & precipitation of the silver from the solution. Roeszner.
- III. A method by Phillips. Roasting with salt and leaching with water.
- IV. Roasting to sulphate - leaching with hot (acid) water & prec. (by copper).

On analyzing the ore, arsenic to the amount of 1.33% was found to be present; this made it necessary to omit the Zivorgel method; all the silver would be changed to arseniate and none go into the solution. There were left, to be tried, the Amalgamation, and Roeszner methods. (Phillips' was not practicable for several reasons). Up to this time

the calculations had been made for getting out silver only, as the ore was not supposed to carry much gold: but on making assays, and finding it to contain almost as much gold as silver (by value), it seemed advisable to alter the methods somewhat, so to extract both the gold and silver, if possible.

The methods that I decided to try were I A modification of Roeszner's process, & II Amalgamation: and the first trial was made on a sample of the ore by method I

Gas Chlorination Method: #1.

Fifty lbs. of the ore crushed to  $\frac{1}{40}$ " was put into a reverberatory furnace, (previously heated to low redness), and roasted at a good red heat for  $2\frac{1}{2}$  hours, & then exposed to a stronger heat for another hour when the charge was drawn. It was then put into a Chilean Mill and ground for 45 minutes with 5% of rock salt.



This mixture of roasted ore and salt was then put into the furnace again, and exposed to a moderate red heat for  $1\frac{3}{4}$  hours and then drawn out. Enough sulphates were left on roasting the first time, so that when heated with NaCl decomposition would take place with the formation of  $\text{Na}_2\text{SO}_4$  and chlorides of Zn, Pb &c. At the same time most (theoretically all) of the Ag is changed to AgCl, and some of the Au to chloride though most of it remains as metallic Au. Some free Cl goes off at the same time.

AgCl being quite volatile it is advisable not to heat above a moderate redness. nevertheless the loss of some of it cannot be prevented, as it is carried away by the other volatile chlorides which are formed, and which cause a constant fuming of the charge.

If the first roasting were carried on long enough, all the sulphur would either be burned

to sulphurous acid ( $\text{SO}_2$ ) or the sulphides would be changed to sulphates; and samples were taken at different stages of the process to determine the amt. of  $\text{SO}_2$  present after different times of roasting. These samples were analyzed and gave results shown in the table below. The analysis was made by digesting for some time with a warm solution of  $\text{Na}_2\text{CO}_3$ , which brings all the  $\text{SO}_2$  into a soluble form as  $\text{Na}_2\text{SO}_3$ , and determining the  $\text{SO}_2$  in the liquid.

| No. of hours roasted. | % of $\text{SO}_2$ present. | % increase of $\text{SO}_2$ |
|-----------------------|-----------------------------|-----------------------------|
| Raw ore               | 0                           | 0                           |
| 1                     | 2.56                        | 2.56                        |
| 2                     | 5.68                        | 3.07                        |
| 3                     | 7.86                        | 2.23                        |
| 3½                    | 8.14                        | .28                         |

By this table it will be seen that the increase during the last half hour was quite small, and at the time the charge was drawn there was none or only very little of the S present as sulphides. (probably none). Assuming that all the Sulphur existed as sulphates (5.42% S)



then 63% of the total  $\text{SO}_2$  in the oil went up the chimney in this operation. When the charge was drawn it gave no smell of  $\text{SO}_2$ .

The table shows the loss of wt.

———— First roast. ————

Charged 50 lb. Dren out  $42\frac{1}{8}$ . Loss  $7\frac{7}{8}$  lb = 15.75%

Coal used =  $65\frac{3}{8}$  lb.

———— Grinding in Chilian Mill. ————

Charged  $42\frac{1}{8}$  lb. roasted oil + 2 lb. salt.

Wt. after grinding =  $41\frac{7}{8}$ . Mechanical loss =  $2\frac{1}{4}$  lb.

= 5.1%. This large loss was caused by the mill being run too fast.

———— Chlorinating Roasting. ————

Charged  $41\frac{7}{8}$  lb. Wt. after roasting =  $39\frac{7}{8}$  lb.

Loss =  $\frac{1}{4}$  lb = .6%. Coal used  $26\frac{1}{2}$  lb.

Total loss in the operations =  $10\frac{3}{8}$  lb = 20.75%

One half of this chlorinated oil was then made damp + put into an inverted bottle with the bottom cut off, and Chlorine gas was passed up through it till the vessel was full: this took about  $3\frac{3}{4}$  hours.

It was then left over night to give

sufficient time to change all the gold to chloride, and was then leached, first with warm + then with hot saturated solution of salt, + finally with hot water.

The liquid as it passes through, takes up the chlorine, in which respect the process is similar to Roessner's, though the liquid is not saturated with Cl. except the first portions of it; but it is much more convenient.

The liquor, which contained Ag Cl, Pb Cl, Au Cl<sub>3</sub> + perhaps other chlorides, was made slightly acid with HCl and the Au, Ag, Pb + (Cu?) precipitated as sulphides.

The precipitate was filtered, dried + run down in a crucible (after partially washing it) with 150 grms. of proof lead; the lead button scorified and cupelled + yielded a button of Ag + Au weighing 1.4045 grammes. This, parted by HNO<sub>3</sub>, gave Au = .0449 + Ag 1.3596

The assay value of this amount of ore, is Au \$ .115 + Ag \$ .141 - the value of A. + S. extracted = Au .029 + Ag .05



From this it appears that about  $\frac{1}{3}$  the silver &  $\frac{1}{4}$  the gold was extracted, but there was no way of knowing whether the rest was lost in roasting or whether it was in the tailings, as no assay was made of the chlorinated ore as it came from the furnace. The tailings assayed for gold \$1.81 per ton & for silver \$2.55 per ton.

The principal reason why this trial was not more successful, appeared to be that the leaching was not carried on long enough, & it seemed probable that if properly leached, a large portion of both the silver and gold might be extracted: and the process was afterwards worked more satisfactorily; an account of which will be found farther on.

575 lb. of the raw ore was then roasted in the same manner, as the sample had been, with a few modifications.

The time of the first roasting of the ore varied from  $2\frac{3}{4}$  hours to 4 hours, making

it a rule not to draw the charge until all the odor of  $\text{SO}_2$  had gone. The grinding with salt in the Chilean Mill was continued for an hour at first, and finally reduced to 30 minutes. The mill was run more slowly and there was not the mechanical loss that there was before.

The chlorinating roasting was kept up 2 hours.

The losses &c will be seen below.

— First Roasting —

Charged 575 lbs. ore. Drew out 468 $\frac{1}{4}$  lbs.

Loss = 106 $\frac{3}{4}$  lbs. = 18.57%

— Grinding in Chilean Mill —

Charged 468 $\frac{1}{4}$  lbs. ore + 24 salt. = 492 $\frac{1}{4}$  lbs.

Took out 486 $\frac{1}{8}$ . Loss = 6 $\frac{1}{8}$  lbs. = 1.25%

— Chlorinating Roasting —

Charged 486 $\frac{1}{8}$  lbs. Wt. after roast = 472 $\frac{1}{4}$

Loss = 13 $\frac{1}{8}$  lbs. = 2.8%

Total loss = 126 $\frac{3}{4}$  = 22%

When roasting the raw ore, after the charge had been heated for about 30 minutes it became remarkably liquid, running about



almost like water - probably due to the escaping of the  $\text{CO}_2$  from the Siderite ore.

The weight of ore to each charge for roasting was from 50 @ 55 lbs.: and for chlorinating, 40 lbs. is about the right amount: we ran two charges of 67 lbs. but it is rather too much. It does not matter so much in the salt roasting as in the first roast, where contact with the air is more necessary.

Coal (Cumberland) used in roasting the raw ore =  $697\frac{3}{4}$  lbs. or 1.2 lbs coal for every pound of ore. Coal used in the salt roasting =  $330\frac{1}{2}$  lbs. or .67 lb. for each pound of ore chlorinated.

Assay of the chlorinated ore gave its value as Ag  $\$11.06$  + Au 7.23 per ton. On account of the loss of weight in roasting, 78 lbs. of the chlorinated ore should contain all the gold + silver that was in 100 lbs. of raw ore: the roasting being a sort of concentration: its value should have been Au 11.83 + Ag 14.46

in other words, about 38% of the gold and 23% of the silver was lost by volatilization or otherwise during the processes of roasting.

Amalgamation.

100 lbs. of the Chlorinated oil were put into the Washoe Pan with water enough to make ~~at~~ a thick cream, + the pan was run for two hours to grind the mass.

Then 5 lbs. of Hg. was added and the mixture kept at about 125° Fahrs. by blowing steam into it. After running one hour and a half longer, 5 lbs. more of mercury was added, and also a little K<sub>2</sub>Cy to keep the gold + mercury clean. At the end of another 1/4 hour, 2 1/2 lbs. Hg was put in and after 3/4 hour more, another 2 1/2 lbs. making 15 lbs. in all. The reason for adding the last 5 lbs. was that the charge was getting too liquid from the condensation of steam used in heating it. It should be kept stiff enough to prevent the Hg from settling out too fast.



Four hours after the mercury was first added, the pan was filled up with water, and the engine slowed down to allow the Hg and Amalgam to settle out. The liquid was then drawn off at the top and the mercury cleaned off out of the bottom of the pan.

The mercury was cleaned from dirt by a glass separator



The mercury was distilled in an iron retort over a forge fire, and it was found that no appreciable quantity had been lost in the operations - if any had been lost, the scales used would not show it. The residue in the retort consisted of about 65 gms. of lead containing Ag & Au. This was run down in a crucible with 150 gms. of proof lead, and as that did not work very well, it was run down again with 100 gms. litharge and some borax (there was some carbonaceous matter from the lining of the retort). The lead button was scorified and cupelled and yielded

Silver 3.4672 grms. + gold .0938 grms. = Ag #.129  
 and Au #.0623      The assay value of  
 100 lbs. chlorinated ore is Ag #.553 + Au #.361 ;  
 therefore only 17% of the Au + 23% of the  
 silver was extracted by this method + it was  
 evident that this was not the method by  
 which to work that ore, especially as a  
 rough trial of the Gas Chlorination process  
 gave far better results. That process  
 was therefore the one by which the rest of the  
 ore was to be worked, or as much of it as  
 time would allow.

#### Gas Chlorination Method #2

88 lbs. of the chlorinated ore was taken for this  
 trial. The method used was about the  
 same as in the first case, except that the  
 operations were done more thoroughly.

The ore was saturated in a tub, with  
 chlorine (being first moistened and all the  
 lumps broken up) and left 15 hours: it was  
~~then~~ then saturated again with chlorine and  
 allowed to stand 36 hours, when it was



thought that all the gold would certainly be chlorinated. Two pailfuls of boiling hot water was then passed through it: then 4 pailfuls of hot saturated brine & finally 2 more of hot water. The sulphides were precipitated as in the first case, allowed to settle over night and the top liquid was drawn off by a siphon. The remaining liquid and precipitate was then filtered and the precip. dried. An attempt was made to roast the precipitate, but either from the presence of an excess of S, or from some other cause, it fused at a very low heat and made roasting by ordinary means impossible. So it was run down in the same manner as an ore for a lead assay, using the following flux:- The PbS weighed 760 grms. Soda 380 Borax 20. Coal 15, + iron about 225 grms. This was exposed in a #8 black-lead crucible, to a bright red heat for 30 minutes & then the charge poured into a cooling mould. A pig of lead was

obtained weighing 490 grms. This was cupelled  
and yielded Ag 9.2334 grms. Au .1916

Value of Ag. obtained \$ .344 Au. \$ .127

Assay value Ag. .486 Au .31

By this operation 70% of the Ag. and  
41% of the gold was recovered from the chlorinated  
ore.

#### Gas Chlorination Process #3

Another lot of the ore was then treated  
by the same process, for the purpose of ascer-  
taining at what stage of the leaching the  
gold + silver was taken out, and the amt.  
of each at different periods. 92 1/2 lbs.

of ore was chlorinated by Cl. gas as before,  
and leached, first with 3 pails of hot water,  
then 6 of brine + finally one of water. The  
liquid after passing through, was kept in bottles  
(holding 2 gallons @) and precipitated separately.

The leachings amounted to eight bottles  
full. Nos 1 + 2 contained the water solution  
(No. 2 was only 2/3 full) and the others brine solution,  
(No. 8 only 1/2 full). The following table will



give a description of the contents of each bottle. When the solutions cooled, a lot of white salts crystallized out, probably composed of  $PbCl_2 + AgCl$ . Part of the sulphides were run down with lead alone, and the others by the fluxes as in the lead assay.

| No. of bottle                      | 1           | <sup>2/3 full</sup> 2 | 3            | 4      | 5         | 6      | 7                     | <sup>1/2 full</sup> 8 | Total  |
|------------------------------------|-------------|-----------------------|--------------|--------|-----------|--------|-----------------------|-----------------------|--------|
| Smell of Cl.                       | strong      | strong                | quite strong | weak   | very weak | none   | none                  | none                  |        |
| Amt. of salts crys. out on cooling | very little | little                | 1/8"         | 1/2"   | 1/2"      | 1/4"   | 1/4"                  | 1/4"                  |        |
| Solution in?                       | water       | water                 | weak brine   | brine  | brine     | brine  | brine                 | weak brine            |        |
| Wt. of sulphides                   | 10.         | 10                    | 23           | 130    | 135       | 140    | 130                   | 56                    | 634    |
| Run down with                      | 60 Pb       | 60 Pb                 | 100 Pb       | 100 Pb | 130 Pb    | 90 Pb  | Flux as in lead assay | same as 7             |        |
| Wt. lead button                    | 67          | 56                    | 100          | 110    | 152       | 96     | 134                   | 133                   | 848    |
| Wt. silver obtained                | .1155       | .055                  | .6924        | 1.5787 | 1.6818    | 1.1683 | 2.371                 | .6184                 | 8.2811 |
| Wt. gold obtained                  | .1390       | .068                  | .0996        | .0038  | .0022     | .0027  | .039                  | .0026                 | .3579  |

Value of gold obtained = \$.24 silver = \$.31

Assay value gold = \$.33 1/4 silver = .51

% of gold extracted from Chlorinated ore = 72 %

" silver " " " = 61 %

It seems from this experiment that there was considerable silver being dissolved when the operation was stopped after the eighth bottle

in fact the amt. of silver in #8 was only a little less than in the preceding ones. The quantity of gold, on the other hand, was getting quite small, and probably not much more could have been obtained by longer leaching. There is no doubt but that if the leaching had been continued till we had 12 bottles filled, a very large per. cent of the silver would have been obtained. The amt. of gold would not have been increased much and the most probable state of affairs is that the gold was not all chlorinated by the gas, & though it was allowed to stand saturated for 24 hours, it should have been left longer.

58% of the gold extracted came in the first two bottles, - the first 3 gallons of leachings - & 86% of it came in the first 3 bottles - the first 5 gallons of liquid; the amt. in the others was small. The silver that came in the water solution of bottles 1+2 was probably dissolved by the salt that was left in the ore after the chlorinating roast.



The buttons obtained from cupelling the prec. in # 1 + 2 had such a large proportion of gold that they would not part without recupelling with more silver.

The results obtained from the different trials are shown in the following table.

| Method               | Chl. Ore taken | Equivalent in raw ore | % of Ag. + Ag. extracted from chlorinated ore |      | % extracted from raw ore |        |
|----------------------|----------------|-----------------------|---|------|--------------------------|--------|
|                      |                |                       | Ag  | Au   | Ag                       | Au     |
| Amalgamation         | 100 lbs        | 128.2 lbs             | 23 %  | 17 % | 17.8 %                   | 10.½ % |
| Gas. Chlorination #2 | 88             | 112.8                 | 70  | 41   | 54                       | 24½    |
| " #3                 | 92½            | 118.1                 | 61  | 72   | 46½                      | 44     |

\*

\* The small percentage here is owing to the loss in roasting. Both the processes tried are preceded by the chlorinating roast and it is found that the greatest loss is in this part of the operation.

#### Assays

|                 | Au     | Ag      |                         | Au     | Ag     |
|-----------------|--------|---------|-------------------------|--------|--------|
| Raw Ore         | # 9.23 | # 11.28 | Tailing. Amalg. process | # 6.03 | # 6.80 |
| Roasted Ore     |        | 11.91   | " Gas. Chl. #2          | 4.82   | 1.70   |
| Chlorinated Ore | 7.23   | 11.06   | " " #3                  | 3.61   | 1.70   |

The efficiency of the Gas Chlorination process is far ahead of the Amalgamation process for

this ore, and I think that with a product such as would be obtained from a richer ore. e.g. the ordinary ore of the Merrimac Mine, the process, or something very much like it, might be made to pay.

But several changes should be made in the method of working. There seems to be no way to prevent the large loss of Ag. and Au. during the roasting, but the heat in the salt roasting should be just sufficient to make the mass fume, not too strongly. The ore should be moistened just enough to hold the Cl. well, but not enough to make it stick together in lumps or run into a mud or paste. After it is saturated, the tub should be covered tightly on top and allowed to stand about 12 hours; and then be saturated again & this process repeated 3 or 4 times. In this way all the gold would be chlorinated and none of it escape the action of the gas as I think it did in the experiments made.



The leaching should be done about as follows, for 100 lb. charges. First 4 pails of hot water, which will heat up the mass and dissolve 85% or more of the gold. Then leach with 12 pails of hot saturated brine. I think that this would take out nearly all the silver & gold that were present as chlorides in the mass.

If it were to be done on a larger scale, a number of tubs (perhaps 6) should be used, and set one above the other, the first one at the bottom. The hot brine should be poured into the upper one and allowed to drain through the whole of them.

In this way a complete leaching could be accomplished without the loss of salt and time which would be the case if each tub were leached separately, when the last few pails of brine would take up only very little silver. It would take considerable time to heat up the ore

in all the tubs when the apparatus was first started, but it should be done with water which is not so expensive as brine. Cold brine dissolves only very little  $\text{AgCl}$  and therefore it would be a waste of material to heat up the mass by pouring hot brine through it. After the tubs were once well heated up, it would not be necessary to use water at all, but when a new one was to be added at the bottom of the series, it should be previously heated with 2 or 3 pails of boiling water which would take out at the same time most of the gold.

In all the cases mentioned in this paper, the solution was made slightly acid and then saturated with  $\text{H}_2\text{S}$  which precipitates Sulphides of  $\text{Ag}$ ,  $\text{Au}$ ,  $\text{Pb}$ ,  $\text{Cu}$  &c. The sulphide was then run down with proof lead or with soda, iron &c as in lead assay: on a larger scale the latter method would be the best & cheapest. The precipitation



by  $H_2S$  is not very convenient, but it seems to be the best way. On a large scale it might be more convenient, for the bottle next to the generator would be saturated first and could then be taken away and the gas passed directly into the next one.

To prevent waste of  $H_2S$ , where there are several solutions to precipitate (as in the 8 bottles in #3) the method used was to pass the gas from one bottle into the next and so on, by closing each one with a stopper having two apertures.

By this method, the pressure is increased in the generating flask, so that there would soon be a limit to the number of bottles that could be worked in one set.

In the process mentioned above, the pressure was equal to about eleven feet of water, and to diminish it somewhat, an aspirator was attached to the last one, by which the pressure could be varied: and as it is necessary to put fresh acid into the

generator occasionally, & as it would not be convenient to have a whistle tube 11 feet long or to open the flask every time, all that was needed was a short whistle-tube & a stop-cock attached, & when fresh acid was needed, a vacuum is formed by means of the aspirator and the acid runs in without any trouble.

The  $H_2S$  was made by sulphide of iron and  $HCl$ , this being the most convenient way. The free chlorine in the first leachings, uses up a great amount of  $H_2S$ , but if it be allowed to stand for some time the greater part of it is got rid of.

The production of a large amount of chlorine is one of the necessities in this process, and the rapid generation of it is a matter of some importance, as it may make the difference between 4 hours and 12 in saturating a charge.

In working the experimental lot of 20 lbs. the  $Cl_2$  was made from  $MnO_2$  &  $HCl$



and it took 4 hours to saturate it once, not to mention the disadvantage of being obliged to heat it & therefore to watch it.

Afterwards in the two larger lots, the cl. was obtained from bleaching-powder and  $H_2SO_4$  and it required only 5 hours to saturate nearly 5 times as much ore, and the apparatus only had to be attended to occasionally, to add fresh acid.

The gas was made in bottles holding something over  $\frac{1}{2}$  gallon. About a cupful of the bleaching-powder was put in and shaken up with  $\frac{1}{3}$  the bottle-full of water to wet it thoroughly - otherwise a cake of  $CaSO_4$  is formed over the top on addition of the acid, and the lower part is not attacked. The acid is added of the strength of 1 part  $H_2SO_4$  @ about 10 of water: the sufficient dilution of the acid is important, as otherwise the mass soon becomes partly solidified and the action gets very weak. It took two such charges

to saturate 92 lb. of the ore, and, <sup>as</sup> no heat was required, a bottle can be used instead of a flask for a generator, and there is less danger of breaking.

Allowing that 75% of the total value of the raw ore ~~was~~ <sup>could be</sup> extracted, the process could probably be made profitable on a richer ore.

100 lb. would then yield \$ .75 To get this out requires about 45 cents worth of coal and salt, and also bleaching powder, acids, sulphide of iron, fluxes for running down the sulphides, and cupels. (and labor.)



guard.

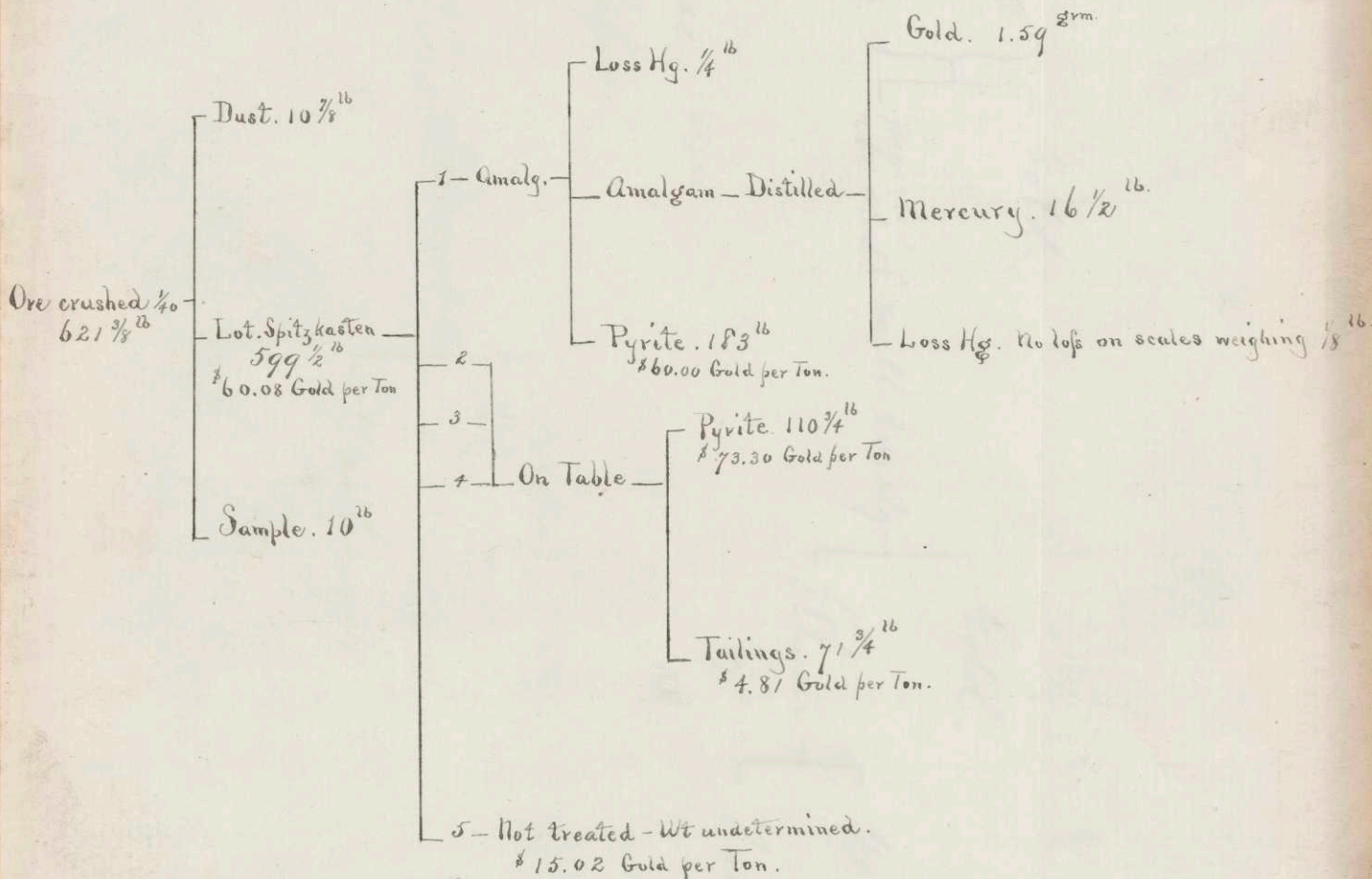
Mass. Institute of Technology.

MINING AND METALLURGICAL LABORATORY.

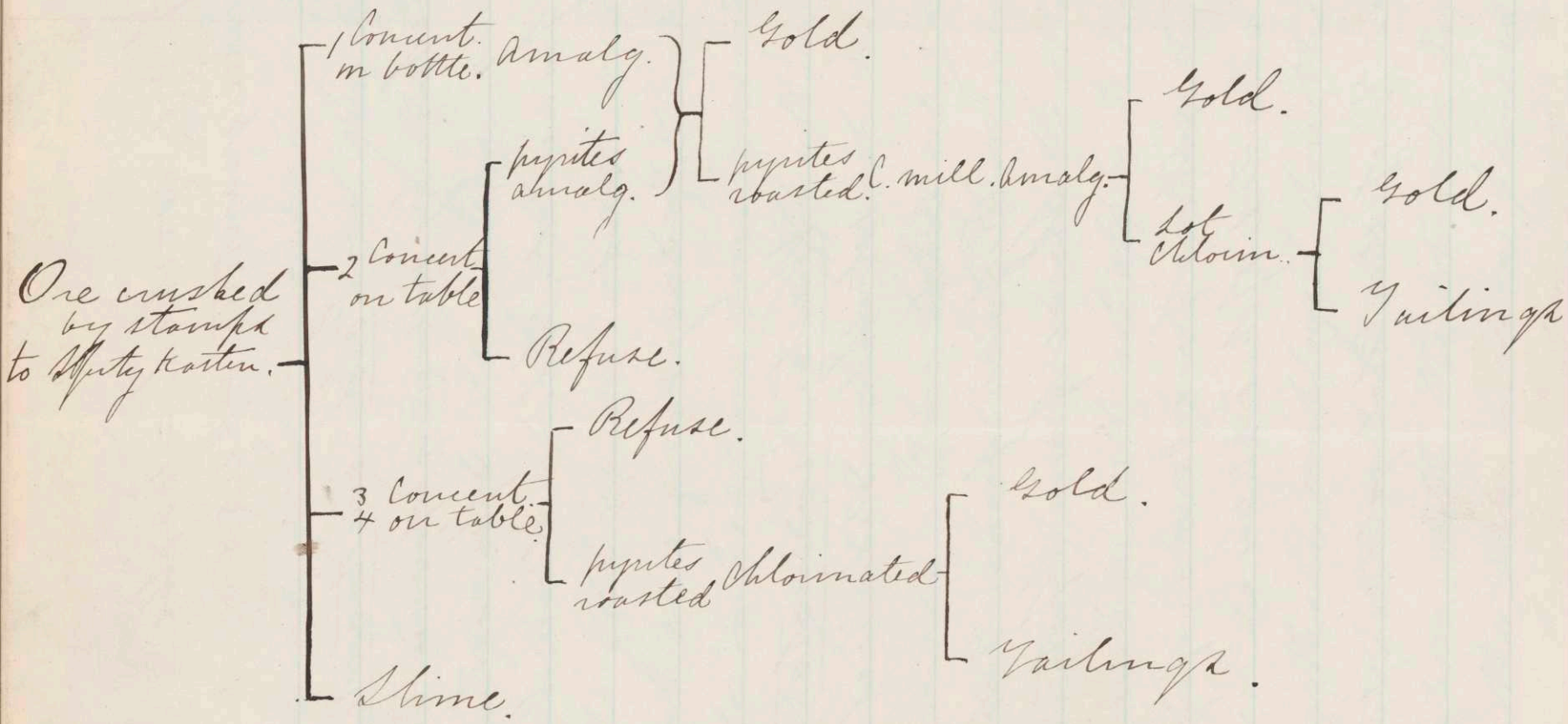
Boston, June 2<sup>nd</sup>, 1877

# Gold Ore

Pewabic Lode, Russell Gulch, Colorado.



C. B. Wheelock.





Garden Lode ore. Col.

W. C. Holman.

Wt of ore worked = 467 lbs.  
Gold in this quantity = \$2.53

Gold Extracted.

Gold taken by mercury before roasting = .6419 gm  
" " " " after roasting = .5807 gm  
" from 1st Chlorination. = 1.0813 gm  
" " 2nd " = 1.510 gm  
" " Assays. = .0197 gm

Waste Products.

1st Chlorinated Tailings.  
2nd " "  
Refuse from Table.  
Slime from Spitz Kasten.

In 467 lbs of ore \$2.53 of gold.  
Extracted of this \$1.64

64.82% of the total gold extracted.

Report on the Garden Lode  
ore of Colorado.

H. H. H. H. H. H.

A pyritiferous gold ore consisting principally of iron pyrites and quartz; about 20% of the former to 73% of the latter; also, calcopyrite, zinc-blende, galena, calcite and dolomite making up the rest.

Most of the gold exists or did exist in the pyrites, but as a great deal of this has suffered decomposition consequently retarding free the gold, it is difficult to determine experimentally how much the quartz actually carries. This lot of ore judging from its decomposed state has been taken out near the surface and results obtained by amalgamation will not apply to the same ore taken from lower levels where this force has not been so active.

The two methods employed for the extraction of the metal are:

Amalgamation with the Atwoods Amalgamator, and the Chlorination process.

The main features of this process are:



1st, the treatment of the moistened pulverized roasted ore in closed vessels with chlorine gas; 2ndly the lixiviation of the ore, and 3rdly the precipitation of the gold from solution. The following is an outline of the work done on this ore.

From the Blake's Crusher the ore is fed into the Stamp-mill where it is reduced to sand, and passing out through sheet iron screens, is by troughs conducted into the Spitz-Kasten. The Battery works off 64 lbs of ore per hour, each stamp making 104 falls a minute, with a water supply to the battery of 115 lbs per minute.

To the discharge of the first Spitz-Kasten box is attached a concentrator which makes a very fine separation of the coarsest pyrite from the quartz and gives a rich product containing free gold.

From the three other discharges were obtained products of different coarseness but relatively of the same composition though varying slightly in value.



The slime from the overflow was too poor to work.

These three products were successively treated on the end-bump table and a very lean separation obtained.

The 1st S.K. product and the concentration from the 2nd were treated separately on the Atwood's amalgamator, but owing to an oversight of the fact that mercury retains gold which can not be separated as amalgam, the two results were obtained as one on the distillation of the mercury. The quantity of amalgam was so exceedingly small compared to the amount retained in the mercury that it was put in and distilled with the rest.

The ore is now reduced to four buckets full of concentrations which are roasted in two lots, the 1st and 2nd together and the 3rd and 4th. The first lot assayed \$17.16 per ton and the second \$20.49. The roasting in each case was kept up for over six hours until a dead roast was obtained.

After this the first was ground to powder



in the Chilean mill and reamalgamated  
then both portions separately chlorinated.  
It will be seen on looking at the  
tabular statement of the waste products  
that the second chlorination succeeded  
as well as could be expected, but that  
in the first there was a great loss of gold  
in the tailings. For this loss I am not  
able to account but am quite confident  
it is not the result of an imperfect roasting.  
I am inclined to think that some  
deleterious matter might have accidentally  
got into the ore before chlorination and  
precipitated the gold in the chlorinator.

Reduced to scale of one ton = 2000 lbs.  
Value per ton = \$10.85

total 2.4746 grams.

| Wt of product.       | Wt of Gold. | % of Gold. | Value in \$. | Value per ton. |
|----------------------|-------------|------------|--------------|----------------|
| 57 $\frac{1}{4}$ lbs | .725 grams. | .0023%     | \$.48        | \$ 16.88       |
| 58 $\frac{1}{4}$ lbs | .135 grams. | .0005%     | \$.09        | \$ 3.01        |
| 193 lbs              | .436 grams. | .0005%     | \$.29        | \$ 3.01        |
| 62 lbs               | .226 grams. | .0008%     | \$.15        | \$ 4.82        |

At this rate, off \$10.85 per ton  
\$7.03 would be extracted.



Garden Lodge Ore.  
Col.

Fletcher