

Thesis.

Treatment of Vershire Copper
Ore.

Harry C. Southworth.

In treating this subject
I shall divide the thesis into four parts:

I. The ore, with its ultimate analysis
and the minerals contained.

II. A description and account of the
treatment of the ore and the
results obtained.

III Deductions from my experience in
working the ore, with the modifications
of treatment suggested by that experience,
and the comparison of the methods
with remarks upon them.

IV. Tables etc.

Part I.

The ore treated by me is a sulphurett ore from Vershire, Vermont.

It is remarkably free from gangues, which is found to consist of a mica schist composed of quartz and two kinds of mica.

The following table shows the approximate mineral composition of the ore, as calculated from the elemental composition, with the specific gravities:

Minerals . . .	Percentages	Sp. Gr.
Quartz -----	6.4	2.6
Chalcopyrite ---	32.7	4.3
Blende -----	3.5	4.
Pyrotilite -----	56.	4.5
Muscovite -----	unde.	
Diotite -----	"	
Feldspar ?	"	

From the foregoing table, the specific gravities of the ore is found to be about 4.3, which is the same as that of chalcopyrite.

The elements found in the ore, with the percentage of each, are shown in the following table:

Elements.	Percentages
Sulphur -----	33.46
Silica (insoluble) ----	6.40
" " (soluble in acids) -	.15
Copper -----	11.25
Iron -----	46.47
Zinc with some Cobalt ----	1.66
	<hr/> 99.39
Alumina -----	Trace
Magnesia -----	" "
Alkalies -----	" "
Nickel -----	?

The amount of Cobalt in the ore is not very large, but it was sufficient to give the zinc precipitate a decided gray color and gave to the borax bead a very deep blue. The alumina magnesia and the alkalis probably exist only in the gangue, and their percentage is probably very small. The presence of nickel I was not able to prove.

All percentages given in this work will be taken from the most trust-worthy of a series of results, except in the case of the percentages in the latter part, where I was not able to make duplicates.

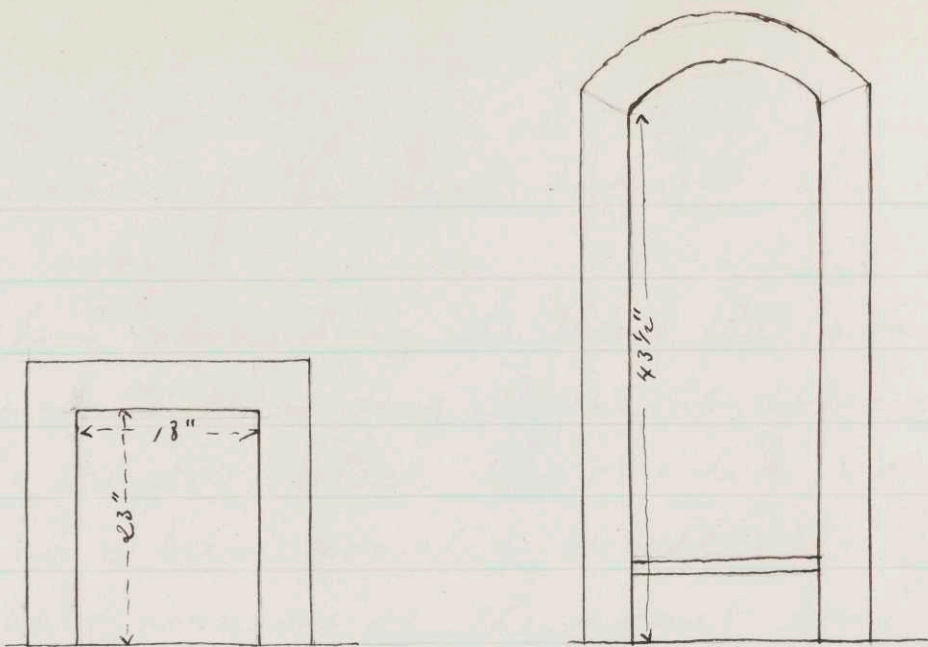
Part II.

As the ore came from the mines in just the right size for kiln roasting, the first thing necessary to do with it, was to roast, in order to get rid of a part of the sulphur.

This has been accomplished in former years in the reverberatory furnace, but as it was considered desirable to try kiln roasting this year, my first work was to construct a kiln.

This was constructed of fire brick, the front wall not being laid in mortar, so that the ore could be charged at the front and the bricks piled up as it was done.

The shape and dimensions of the kiln are shown in the following roughly drawn figure:



As seen in the figure, the kiln was covered with an arch through which the smoke was carried off by the means of an iron stove pipe $4\frac{3}{4}$ " in diameter.

Subsequent experience proved that the smoke pipe was rather too small, especially after being partially filled with an accumulation of sulphur salts.

The first roast was started with $638\frac{5}{8}$ lbs of ore and 5" of fuel as the first charge.

The fire was lighted at 10.53 A.M.
At 5 P.M. the greatest heat was attained.

and at 9 the next morning the kiln was cool.

Upon overhauling the kiln, the roast was found to be a very poor one, especially in the centre, where the ore did not seem to have been roasted at all.

The pooriness of the roast was partially due to the large amount of fine ore accumulated at the centre which, which caused the draft to be mostly on the sides near the walls.

The loss in weight of the ore in this roast was 1.9%, by which I should judge that about $3\frac{1}{2}$ - 4% of the sulphur must have been driven off.

Finding the ore so poorly roasted the first time, I concluded to try some coke with the second ~~roast~~; accordingly I charged the kiln with 5" of wood and a thin layer of coke, upon which was piled the ore - part of which was in the kiln during the first roast - with

a few layers of coak mixed in, the total amount of coak used being 28½ lbs.

This charge was taken out about 24 hours after it was lighted, and the roast was found to be a very good one, especially in the centre, where a large part of the ore was found to be clinkered, which showed that it was very thoroughly roasted.

The third charge consisted of the poorly roasted ore of the first two charges, with the usual amount of wood and the coak increased to 44½ lbs.

This time the coak charge was found to be too great, as the ore was partially melted and ran through the grate bars four or five hours after starting the fire.

On discharging, the upper half was found to be well roasted, with the exception of about 2" on top, while the lower half was caked together in a spongy mass and looked as though it had not lost much sulphur.

The fused mass was not very troublesome to get out, as it stuck very slightly to the walls of the kiln, but it was not very easily broken up.

After the third roast I sampled the ore for the purpose of determining the amount of sulphur still contained, and found upon analysis that it was about 21.61%.

As the amount of sulphur still in the ore was so large that a matte got by running it down in the blast furnace would be very poor, I resolved to give it another roast, and accordingly I picked over the roasted ore and charged the kiln with that which was the least-roasted, together with the usual amount of fuel, mixing considerable of the coak with the top layers in order to get as good a roast as possible on top.

On discharging, I found that the lower

part of the mass was caked together, while the upper part had hardly got more than an average roast.

Although the ore was so unsatisfactorily roasted, it was thought best that I should run the ore through the blast furnace without further roasting.

Accordingly I sampled the roasted ore, and an analysis showed the result of the four roastings was the reduction of the percentage of sulphur from 33.46% to 19.66% with a loss of 6.8% in weight of ore.

From the above can be seen that I had left 18.32% of the sulphur, supposing the ore to be reduced to its original weight, that 15.14% of the weight of the ore was lost as sulphur, and that of the sulphur in the ore 45.25% was driven off by the four roasts.

In accomplishing this result, 105½ pounds of coke were used, with wood

and shavings enough to start the fire; the comparative efficiency of this and the other method will be spoken of in Part III.

after finding the percentage of sulphur left, I prepared for the blast furnace run by calculating the amount of material to be added in order to make a certain slag; the proportion and the material added will be seen in the table, on first blast furnace run, part IV.

I calculated for a slag containing as follows:

	percentages
Protoxide of iron - - -	48.8
Silica - - - - -	33.
Lime - - - - -	12.4
Alumina - - - - -	6.

The matte was calculated to contain about 20% of copper, with 32.3% of sulphur and 47.8 percent of iron.

The blast furnace used has an area of about $1\frac{1}{4}$ square feet at the tuyers, there being three $\frac{3}{4}$ " tuyers at a height of about 4" above the top hole.

In this run the fore hearth was used; this consists of an iron frame, which is first packed full of steep, and then scooped out so as to hold quite a respectable quantity of the melted material; The slag and the matte is allowed to flow from the furnace as fast as it is melted into this place, where it settles; and by allowing the slag to run off continuously at the top, and by drawing off the matte at intervals lower down, a very perfect separation of the two are made.

The results got with the fore hearth were not very satisfactory,

although a very good separation of the matte and slag was got at a time when everything was at a full heat.

The principal points against the fore hearth as it was used this year are, that the weight of the accumulated slag caused the matte to rush out with such force that it was very liable to spatter, causing some loss of matte, that the heat rendered it very uncomfortable to work around the place, and that considerable coal was used during the run to keep up this heat and considerable foul slag was made at the end of the run, which had to be run down in crucibles.

However the objection of foul slag disappears when large quantities are run, and spattering might be avoided by some different arrangement.

For the details of the furnace run, I refer you to the table in part IV,

and now proceed to give the results of it and show how nearly I realized the calculated results.

In this run, 350 lbs. of coarser were used, together with char coal and wood enough to start the fire, and 12 hods of char coal for the forehearth.

The products were 52 $\frac{1}{4}$ lbs of matte, 403 lbs of slag, including some "basic Revere", 44 $\frac{1}{2}$ lbs of impure matte, 166 lbs of foul slag and 3 hods of a mixture which was jigged to get rid of the coal and brick.

By running this impure material, I got 50 $\frac{1}{2}$ lbs. of matte and about 1.75 lbs. of slag.

The total matte got was 571 $\frac{3}{8}$ lbs., which was 80 lbs. more than calculated for.

When sampled and analyzed, it was found to contain 16.79% of copper, or about 3% less than calculated for.

For the complete analysis of both the slag and the matte, see next page.

Matte.

	Percentages
Iron - - - - -	53.14
Copper - - - - -	16.79
Sulphur - - - - -	25.61
Insoluble residue -	.75

Slag

	Percentages	
Protoxide of iron - - -	29.77	
Silica - - - - -	39.51	
Lime - - - - -	13.77	
Alumina - - - - -	12.15	
Oxide of zinc - - - - -	.49	
Oxide of Copper - - -	.55	-From commercial sample.
Sulphur - - - - -	.21	

From the above will be seen that, in the matte, the calculation was very nearly realized, but in the slag, the silica, iron

and
aluminium appear to be considerably out.

The gain in silica and aluminium was due to the slags attacking the fire bricks, and also some silica must have been gained from the ashes of the coal used in the roasting and running down, while the decrease in iron was owing mainly to the amount of iron which went into the matte and to the dilution of the slag by silica.

In summing up after this run I find that I have the following figures:

In the original ore -- 97.39 lbs.

Added with acid Residue 3.06 ..
100.45

Copper in matte -- 95.93 ..

Taken out by run slag -- 4.52
1.77

Taken out by slag run from 2.75

the foul slag not analyzed, but } 1.75 - very probably more
supposed to contain 1% more at least } 1.00

These figures leave 1 lb of copper

to be accounted for by losses which cannot be estimated with any great degree of accuracy.

It is probable that an appreciable amount of the ore was lost up chimney, as I had about ~~two~~ hods of very fine, light stuff, which was made in taking two samples and in handling the ore; also some of the matle which splattered was probably not recovered.

However, if none of the copper was lost in this way, the error in sampling, where the lumps of ore are as large as they were in this case, would account for the amount left.

I think there must have been very little copper lost in handling, as I was very careful to avoid loss in this way.

I will not enter into the details of roasting the matte, except to mention the changes in the plan.

Instead of starting the fire with wood alone, I used a hod of charcoal and a smaller amount of wood.

This I think gives a better start to the fire, and makes less smoke than a large amount of wood.

I also tried the experiments of plastering up the front and checking ^{the} ~~the~~ ^{draft} by a piece of sheet iron in front of the grate bars, in order to make the roast last longer, thereby expecting to get a better one.

On the first experiment I came to no conclusion, but I found that the second gave very unsatisfactory results, as, although it made the fire last nearly twice as long, it did not give so good, seeming to supply just air enough to burn out the carbon and leave most of the sulphur.

It did, however, partially prevent smoking, and, although as early as the third roast, I came to the conclusion that I was not giving the kiln draft enough, I kept it there to check the smoke as much as possible.

And I noticed that several times when I gave the kiln more draft, somebody took the pains to check it again.

One reason that the kiln smoked so badly with free access of air, was, probably that the funnel was beginning to be stopped ^{up} by an accumulation of sulphur salts.

In the six roasts, I used five hods of charcoal and 217 lbs of coke, with wood and shavings enough to start the fire. This amount of coke was somewhat larger than I should have used, if I had not tried the experiment of checking the draft.

The matte was roasted from a matte containing 25% of sulphur to

one containing about 10%, and although this was hardly as good a roast, as I wished for, I was compelled by the shortness of the time to run it in this shape.

The only peculiarity in the blast-furnace, when running for the second matte, was that the tuyers were placed about 8" higher than usual, in order to give a greater body of material at each tap.

The slag and matte run for were calculated to be as follows:

Slag		Matte	
	Percentage		Percentage
Protoxide of iron -	58.	Iron ---	26.8
Alumina - -	8.	Copper ---	46.
Silica - - - -	28.	Sulphur	27.2
Lime - - - - -	6.		

In making up the slag, I used some "basic Revere"; as will be seen in the table, part-IV, and, as no very reliable analyses have been made of it, I cannot give the exact percentages which should occur in the slag; however I added but a little, and I know that the approximate percent. of silica and lime is about the same as in the other slag that I used.

I will now proceed to give the results of the run, the details being given in the table on second blast-furnace run, part-IV.

The products of this run consisted of 216 lbs. of matte, 889 $\frac{3}{4}$ lbs. of slag, including some "basic Revere" used in cleaning out the furnace, 9 $\frac{1}{4}$ lbs. of impure matte, and 57 lbs. of a foul slag including a jigged product.

The impure products were run down in number 25 black lead crucibles, the details of which will be

shown in Part IV.

The products from the crucible run were $3\frac{1}{4}$ lbs. of matte, $35\frac{3}{4}$ lbs of a mixture of cast iron, matte and copper, and 23 lbs. of slag.

The total matte got was $22\frac{1}{4}$ lbs. which was 5.9 lbs more than calculated for.

The analyses of the slag and matte gave the following percentages:

Slag.

	Percentages
Protoxide of iron	45.47
Alumina - - - -	6.21
Silica - - - -	32.98
Oxide of copper - -	2.17

Matte.

	Percentages
Iron - - - - -	39.02
Copper - - - - -	33.96
Sulphur - - - -	31.14
Insol. residue - -	1.55

Owing to lack of time, I was not able to make a complete analysis of the slag, or even to make duplicates; however the only element of importance not-determined is the calcium and this generally occurs in the slag in about the calculated amount.

The cast iron got from the crucible run was not sampled, owing to the want of time to run it down into a convenient form to sample; but some pieces look as though there was considerable copper contained, while others seem to be a very pure cast iron. The matte sticking to this product could not be separated without breaking it up a great deal and losing much of it; therefore I concluded not to break it off until I ran the iron down into one lump, when it might be separated with greater ease.

I present the following figures as a summation of the results of the run:

Weight of copper in first matte	---	95.93 lbs
" " " added by slag	---	1.74
		<u>97.67</u>

Weight of copper in second matte	---	76.16
" " " carried off in slag	---	21.51
		<u>15.39</u>
Carried off by $35\frac{3}{4}$ lbs of "cast-iron" and other losses		6.12
		<u>6.12</u>

Although the "iron" was not analyzed, I should judge by its looks that it contained as much as 15% of copper, which would account for 5.35 lbs of that missing, leaving only .77 lb. to be accounted for in other ways.

After separating the second matte I prepared to give it a thorough roast in the reverberatory, by crushing it to the size of $\frac{1}{2}$ ".

The details of the roasting will be found under their proper head in part IV.

After roasting, the matte weighed nearly the same as before, having gained about the same weight of oxygen as it lost in sulphur.

The amount of sulphur in the roasted ^{matte} was not determined, as it was not absolutely necessary to know it in order to plan the charges to be run, and as my time was then very limited.

As the amount of matte was small, it was thought best to run it down in number 25 black lead crucibles, and accordingly I planned a slag which was to consist of 40% silica and 60% of protoxide of iron.

In order to get the above slag, 25 lbs. of sand was added to every 100 lbs. of matte, and in order to effect the reduction of the oxides, 7 lbs of powdered anthracite was added to the mixture.

After running several charges of the ore, I found that considerable matte

was being formed, and in order to lessen the amount and make the sulphur do a part of the reduction, I decreased the amount of coal, and added in its place some of the matte formed in the first charges.

This had the effect of decreasing the amount of matte formed, but it also attacked the crucibles to such an extent that I was obliged to forego the experiment, in order to make my crucibles last.

As it was, my old crucibles gave out with the second charge in them, and my new ones were nearly worn out at the end of the job.

I tried the blast upon the work of running down, but the heat got up was so intense and was got up so quickly that the charge spattered very badly.

I also noticed that if the crucibles were left a great while at an intense heat without the blast, boiling and spattering

commenced.

The cause of the "boiling" was probably a reaction, by which the matte was decomposed with the evolution of sulphurous anhydride, as I noticed that the amount of matte was smaller in the charges that were boiled; however I did not allow the charges to boil when I was able to help it, on account of the loss caused by spattering, and because the crucibles seemed to be attacked much more by the boiling liquid.

The result of the crucible run is as follows:

Black copper $58\frac{5}{8}$ lbs.

Rich matte $16\frac{3}{4}$ "

Slag not weighed but by calculation about 140 lbs.

On an analysis of the black copper, I got three results: namely, 97.01%, 98.80% and 99.13%.

The first of these results I cannot consider trustworthy, as on making the solution, sulphur separated in black flakes, which probably contained some copper.

The wide difference in the results I suspect to be partially due to the defects of the sample.

A determination of the copper in the matte was made, and it was found to contain about 97.75% of the subsulphide of copper.

Not being able to believe this result, I determined the iron and found, to my surprise, that there was only 1.18% in the sample.

There was not time to analyze the slag, and I cannot, therefore, give any figures upon it.

The summation of the above is as follows:

Black copper at 98% = 57.47 lbs.

Rich matte at 77.95 = $\frac{13.05}{70.52}$

Amount of copper in original matte = $\frac{76.16}{5.64}$

By these figures may be seen that 5.64 lbs. of copper is to be accounted for. How much of this went into the slag and how much was lost in handling, I am not prepared to say; but if the slag were to contain it all, it would contain only about 2.22%, and it is quite possible that it may contain that amount, especially as on the surface of the slag there is quite a large amount of copper in the shape of globules held up by the grains of unmelted sand and the coal floating on the surface of the slag.

For comments upon the foregoing results see part IV.

Part - III.

My experience with the Kiln has shown me that its principal advantages over the reverberatory furnace is in the saving of a great amount of fuel and labor, while the reverberatory saves time and does its work more thoroughly.

At the Institute the Kiln does not make so great a saving as when used on a large scale in practical work, owing to the small bulk of ore operated upon and the large amount of radiation; but, on the other hand, although the ore requires much more handling and fuel, the roasting is done in a shorter time.

For the charges for the Kiln, I have concluded that about 1 hod of charcoal with wood and shavings enough to start it, should be used with from 30 to 40 lbs. of coke, the weight of coke used being smallest during the first

roasts, when there is the largest amount of sulphur in the ore.

In running the kiln I tried several experiments: first, giving the kiln all the draft possible; second, checking the draft so as to cause the fire to last longer and have a slower roast; and third, plastering up the front of bricks, both to prevent smoking and to prevent draft above the grate bars.

The result of these experiments satisfied me that, to get the best results, as much air must be let in as possible without melting the charge.

As to the third experiment, I found that its principal use was to prevent smoking.

In order to compare the work of the Kiln with that of the reverberatory furnace, the following figures are submitted:

	Kiln	Reverberatory	
Amount of ore roasted	965 $\frac{3}{4}$	1019 $\frac{1}{2}$	Lbs.
" " " fuel used	105 $\frac{1}{2}$ coke	739 coal	"
" " " labor	8	52	hours
" " " sulphur lost	131	122	Lbs
Percentage of " " "	15.14%	about 12%	

The ore run in the reverberatory was the same kind as that run in the Kiln, the reverberatory being used by Mr Bartol, to whom I am indebted for a part of the information concerning the run.

The labor spent in Kiln roasting consists in discharging, recharging and picking over the ore, and I have found that the time spent in picking over

is well employed

Another point where the Kiln excels the reverberatory is in the fact that an ore must be crushed much more finely for the reverberatory ~~than for the~~ than for the Kiln.

I am not very well satisfied with the results which I obtained, because I think that it is not only possible to get a black coffee in two runs, but that it ought to be done; however the final product which I did get was a very rich one, the matter even being richer than black coffee sometimes is.

I attribute my partial failure to my not being able to get good roasts, not having got a good one at either time; but it must not be concluded that, because I failed to get a good roast, ~~that~~ the Kiln was a failure; on the contrary the results given by it were very good and the

failure was due entirely to lack of time.

This lack of time was owing to my not getting to work upon the ore directly after the semi-annuals, and to the time lost in building the Kiln and making arrangements for a start.

Also some time was lost, owing to the impossibility of our not interfering with each others work in the laboratory.

If I were to run again, I think I should prefer to have about 800 lbs of ore to 1000 as a charge to be worked. The only trouble with such a small quantity would be that it would make a rather small charge for the blast furnace, especially in the second run.

Starting with 800 lbs of ore it should be roasted the first time down to an ore containing 10% of sulphur.

This would require from 8 to 10 roasts, which would take not more than 20

days, during which time, work could be done on analysis.

Perhaps this seems to be too long a time to be spent on roasting, but I think it would save time in the end, and if the work was started in good season it could well be spared.

A reduction of the sulphur to 10% would give, by calculation, a matte containing 44% of copper, but practically the matte would probably contain from 30 to 40 percent.

This matte would be very nearly, if not quite, as rich as my second matte. It would weigh about 300 lbs, and I think that the best way to roast it would be to roast partially in the kiln and then to crush and roast in the reverberatory.

In this way, I am confident very good results would be obtained if the other details, ^{were arranged} as well as they have

been this year

If 1000 lbs. were to be run, about ten days more would be required to roast the ore.

It is a good thing to bear in mind that the ore should be handled as little as possible, both on account of the loss occasioned by it and on account of the amount of fine stuff formed by the breaking up of the ore.

Owing partly to the want of time and partly to the reason that the machines are all of the usual kind, I have not given much of a description of them.

I have made but little account of the power and labor expended in the work, because of the great difference between working in a small way and practice.

In closing I may say that, although my results have not been as good as might have been desired, I have, ^{probably} learned considerably than I should if everything had run so smoothly.

I am much indebted to Professor Richards and Mr Foster for the kind advice given me about the treatment of the oil.

Harry C. Southworth.

Part IV.

Owing to the details of the first series of kiln roasts being in Part II, I do not give them here.

In running down the foul slag from the first run in crucibles I made 15 pours, the average time between the pours being $1\frac{3}{4}$ hours during the 2^d day, when 11 pours were made.

When running down the foul slag from the second blast-furnace run, a blast was used under the crucibles, but the stuff was so infusible that I made only 7 pours in two days.

In running for black copper, only 2 pours were made the first day, because both of my crucibles gave out with the second charge in them. These crucibles, however, had done

considerable work having been charged
13 times.

On the second day, 7 pours were
made from a new set of crucibles, and
on the third day, 6 pours were made.

The intervals were mostly from $1\frac{1}{2}$ to $1\frac{3}{4}$
hours long.

Second kiln roast.

First charge 569³/₈ lbs matte (Monday)
coke 40 ..

Started at 10.45. The front bricks were plastered up and at 3 P.M. charge was somewhat melted, stopped melting when the draft was checked. Good roast.

Second roast. Wednesday

coke 30 lbs, charcoal $7\frac{1}{2}$ hod and $\frac{1}{2}$ the usual amount of wood. Started at 12.30. Draft-checked; not a very good roast.

Third roast. Friday.

Coke 38 lbs, 1 hod charcoal.

Started at 4 PM heat; lasted through Saturday; good roast. The draft was checked.

Fourth roast. Monday.

Coke 34 lbs, charcoal 1 hod; not a very good roast; draft checked to prevent smoking.

Fifth roast, Wednesday.

1 hod char coal and 35 lbs of waste for fuel. Started at 12.30.

a poor roast: draft checked to prevent smoking.

Sixth roast, Saturday.

coke 40 lbs, $\frac{1}{2}$ a hod of char coal;

poor roast.

The second matte was roasted in the reverberatory furnace, there being three charges of about 75 lbs, each, which were roasted eight hours each.

The second charge was run by Mr Foster while the others were run by myself.

The total amount of coal used was 495 $\frac{1}{2}$ lbs.

Table on first blast furnace
run.

Charges.

Charge	I	II	III	IV	V
Ore	30	40	30	60	
Acid Revert ^{slag} --	10	13 $\frac{3}{8}$	16 $\frac{1}{2}$	20	
	40	53	66	80	
Basic Revert --					40
Coke	10	10	10	10	10
	1:4	1:5.3	1:66	1:8	1:4

Pap Record

Feed Record

Time	Interval	Buggy	matte kettle	Time	interval	Depth or change	Charge	
March 21 st								---
4.30								---
22 ^d 9.00								---
9.15								---
				9.30		1 3/4'		
				9.40				---
				9.51				---
				10.		1 3/4'	V	
10.08								---
				10.10	10	2	V	
				10.18	8	2 1/4	V	
				10.25	7	2 3/4	V	
				10.38	13	2 1/2	I	
				10.45	7	2 3/4	I	
				10.55	10	2 1/2	I	
10.57		I				2 1/2		
				11.03	8	2 1/2	II	
11.07						2 3/4		
11.10	13	II						
11.14	4							
				11.15	12	2 3/4	III	---
11.28	14	III						
				11.30	15	2 3/4	III	

--- Fire lit, chips charcoal and 3 hods coke

-- 3 hods coke

-- Blast on

-- 1 hod coke

1 " " "

-- Slag just coming

--- First-~~cap~~ scraped from the fore hearth.

11.35			I				
11.43				11.43			--
11.45		V					
11.48		<u>VI</u>	II				
				11.50	20	2 ³ / ₄ '	III
12.		<u>VII</u>					
				12.3	13	2 ³ / ₄ '	III
12.04			III				
12.11		<u>VIII</u>					
12.17	13		IV				
				12.18	15	2 ¹ / ₂ '	III
12.19		<u>IX</u>					
12.30	13		V				
12.33		<u>X</u>					
				12.35	17	2 ¹ / ₂ '	III
12.43	13		VI	12.43	18	2 ³ / ₄ '	IV --
12.46		<u>XI</u>					
				1.03	20	2 ³ / ₄ '	III
1.11	27		VII				
				1.15	12	2 ³ / ₄ '	III
1.16		<u>XII</u>					
1.30							
		<u>XIII</u>		1.34	19	2 ³ / ₄ '	III
1.36		<u>XIV</u>					

--- Evidently run slag.

--- About 3 kettles full ran onto the floor.

--- Hereafter used buggy for matte.

--- Foul matte ran over onto buggy, remelted.

1.37	26		V III	143	19	2 ³ / ₄	III	
1.47	10		IX					
1.56		XV						
1.59	12		X					
2.02		XVII						
				2.07	14	2 ³ / ₄	III	---
2.12	13		XI					
2.14		XVII						
2.26			XII					---
2.30		XVIII		230	23	2 ¹ / ₂	IV	
2.42	16		XIII					---
				245	15	2 ³ / ₄	0:80	---
2.52		XIX						
				3.00	65		1:9 RFR	---
3.02			XIV					---
3.07		XX	:					
				314	14	2 ³ / ₄		---
3.20		XXI						
3.22			XV					
				326	18	3'	V	
3.38		XXII						
3.43.			XVI					
				345	19	2 ³ / ₄		---

--- Run slag out.

--- Slag again.

--- 60 lbs foul slag 20 acid Revue⁰ coke (by mistake) ^{80:0}

--- 51 3/8 ore 17 acid Revue 7 coke.

--- Full buggy no slag.

--- 64 foul slag 11 coke.

--- 10 coke,

3.54	XXIII		4.00	15	2 1/2'	
4.03	XXIV					
4.04						- - -
4.06		XVII			1 1/2'	
4.16	XXV				1/2'	- - -
4.22						- - -
4.26						- - -

--- Buggy foul

--- Blast-off

--- Foul play

--- Front-out.

α

Table.

Second blast furnace run:

6 charges

	I	II	III	IV	V	VI	VII	VIII	IX	X	XI	XII	XIII	XIV
Matte	22 $\frac{1}{2}$	27 $\frac{1}{2}$	33 $\frac{1}{2}$	39	44 $\frac{1}{2}$	33 $\frac{1}{2}$	41 $\frac{1}{2}$	50	58 $\frac{3}{8}$	66 $\frac{1}{2}$		64 $\frac{1}{2}$		
1st run slag	11 $\frac{1}{2}$	14 $\frac{1}{2}$	17 $\frac{1}{2}$	20 $\frac{1}{2}$	23 $\frac{1}{2}$	18	21 $\frac{1}{2}$	26	30 $\frac{3}{8}$	34 $\frac{1}{2}$		32 $\frac{1}{2}$		
Basic R.	2 $\frac{1}{2}$	3 $\frac{1}{2}$	4 3 $\frac{1}{2}$	4 $\frac{1}{2}$	5 $\frac{3}{8}$	4	5	6	7	9	40	8		
Sand	2 $\frac{1}{2}$	2 $\frac{1}{2}$	3 4	4 $\frac{1}{2}$	4 $\frac{1}{2}$	3 $\frac{1}{2}$	4 $\frac{1}{2}$	5	8	6 $\frac{1}{2}$		6 $\frac{3}{4}$		
Limestone	1 $\frac{3}{8}$	1 $\frac{1}{2}$	2 $\frac{1}{8}$	2 $\frac{1}{8}$	2 $\frac{1}{2}$	2	2 $\frac{1}{2}$	3	3 $\frac{1}{2}$	4		3 $\frac{3}{4}$		
total	40	50	60	70	80	60	75	90	105	120		115 $\frac{1}{4}$		
Coke	10	10	10	10	10	15	15	15	15	15	10	15	15	
Ratio	1.4	1.5	1.6	1.7	1.8	1.4	1.5	1.6	1.7	1.8		1.8 $\frac{2}{3}$		
Poul buggin coke														25 $\frac{3}{8}$ 12

Tap record

Charge record

Time	Buggy	interval	Plugged	Time	charge	depth	interval	
April 25 th								
4 P.M.								--
Apr. 26								--
9.16								
9.26						2'		--
9.35						2 ³ / ₄ '		
9.50				9.50	XI	2 ¹ / ₂ '		--
				10.00	XI	3 ³ / ₄	10	--
				10.15	XI	3 ³ / ₄	15	--
10.28	1							
10.33			1	10.34				
	2			10.34	IX	2 ¹ / ₂ / ⁴ / ₄	19	
10.38	2							--
10.41	3							
10.50	4							--
10.57				10.53				--
10.57	5			10.57	IV	2 ¹ / ₂ / ³ / ₂	23	--
11.04	6							--
11.05			12					
				11.12	IV	2 ³ / ₄ / ³ / ₂	15	
11.14	7							--
11.15			13					
11.25	8			11.25				--
11.26			14					

-- Fire lit- 2 hods charcoal 3 hods coke.

-- Blast-on.

-- 2 hods coke.

-- 1 hod coke

-- flame showing.

-- S lag running.

-- Foul, not plugged immediately.

-- Buggies filled foul.

-- Flame at-top.

-- Foul.

-- Foul.

-- Good run buggies full.

-- 2 full buggies.

		Plugged		Depth	Charge		
11.40	9		1130	$2\frac{1}{4}/3\frac{1}{4}$	IX	18	--
11.41		1					
			11.47	$2\frac{3}{4}/4$	IX	17	
11.55	10						--
11.56		1					
			12.05	$2\frac{3}{4}/4$	X	18	
12.10	11						--
12.12		1					
12.23	12		12.25	$2\frac{3}{4}/3\frac{3}{4}$	IX	20	--
12.26		1					
12.40	13						--
12.41		1					
12.44			12.44	$2\frac{3}{4}/3\frac{3}{4}$	X	19	
12.36	14						--
12.57		1					
			1.02	$2\frac{3}{4}/3\frac{3}{4}$	IX	18	
1.11	15						--
1.12		1					
			1.18	$2\frac{3}{4}/4\frac{1}{4}$	IX	16	
1.25	16						--
1.28		1					
			1.38	$2\frac{3}{4}/3\frac{3}{4}$	XII	20	
1.39	17						--

-- 2 buggies foul on top.

-- 2 1/2 buggies.

-- 2 1/2 buggies.

-- 2 1/2 buggies.

-- 2 buggies and a little more.

-- 3 full buggies.

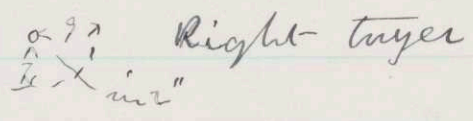
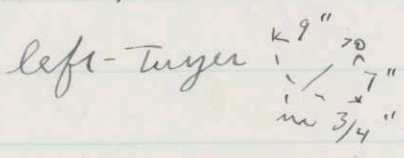
-- 3 full buggies + some on floor.

-- 3 full buggies and some on floor.
also scattered.

-- nearly 3 buggies.

1,40			1					
1,53	18			153	XIII		15	
1,56			1					
2,09	19			2,09	XIV		16	- -
2,11			1					
2,24	20							- -
				2,25	22 lbs. of coke			
2,27			1					
				2,33		2 1/2	8.	- -
2,40	21							- -
2,45								
2,53	22							- -

The furnace when pulled down, cooled and cleaned, showed that the bricks were burnt as in the following diagrams:



-- charged foul buggies 6-5-4-3-2-1 in this.

-- $3\frac{1}{2}$ buggies.

-- $3\frac{1}{4}$ buggies.

-- 15 lbs coke.

-- nearly 3 one foul.

-- a few drops.

Number of determinations made upon thesis work.

	Cu	Fe	Sil ₂	S	Zn	Ca	Al ₂ O ₃	
Ore - - - - -	6	2	8	2	1			18
Roasted ore - - -				2				2
Vershire ore slag -	1	2	2	1	1	1	1	9
First run slag -	1	1	3	1	1	1	1	9
First matte - - -	1	1	1	1				4
Roasted matte - - -				2				2
Second matte - -	1	1	1	1				4
Second slag - -	1	1	1				1	4
Third matte - -	1	1						2
Black copper - -	3							3 = 57

The large number of determinations made upon the Vershire ore is due to the pooriness of the sample first taken, and also the small number of duplicates made by me is owing partially to my lack of time on that account.

However, where the results were not satisfactory, duplicates were made, except in one or two cases.

Table used for calculating slag:
Second run.

	Slag.				Matte.			
	SiO ₂	FeO	Al ₂ O ₃	CaO	Cu	Fe	S	
Roast Matte ---	4.8	55.8			15.46	9.6	9.7	%
620 1/2 lbs	24.8	346.2			95.9	59	60	lbs.
Run Slag ---	39.5	29.8	12.2	13.8	44			%
322.7	127.5	96.2	39.4	44.5	1.4			lbs.
Basic R.*	44?	28?	12?	8?				%
74	32.7	20.8	9.1	5.9	.3?			lbs.
Sand ---	100							%
32	32							lbs.
Limestone ---				56.				%
19.2				19.8				lbs.
approx. %	28.	58.	8.	6	45.	27.2	27.7	%
	217.	463.	60	48	97.6	59.	60.	lbs.

Formula of slag (Al₂O₃)₂ 25(FeO, CaO) 14 SiO₂

" " " matte Cu₁₀ S₁₂ Fe₇ or 5 Cu₂S 7 FeS

* No reliable analyses of the

Basic Reverse slag have been made but
The above percentages are probably ^{correct} within 4 or 5%

Table showing the composition of the ore:

	Cu.	Fe	S	Zn	SiO ₂	
Calcopyrite	11.25	10.00	11.45			32.70
Pyrrhotite		35.20	21.23			56.48
Blende		1.30	.77	1.50		3.57
Quartz					6.40	6.40
	11.25	46.50	33.50	1.50	6.40	99.05

Table used for calculating slag:

For first run.

	SiO ₂		FeO		Al ₂ O ₃		CaO		Matte:		S	
	%	lbs	%	lbs	%	lbs	%	lbs	%	lbs	%	lbs
Ore 806 lbs.	6.5	52.4	27	217.8					12.07		29.	19.6
Acid R. 269-	44.	119.2	15.	40.3	11.9	32	24.7	65.4	24.1			
	32.6	171.6	43.8	258.1	6	32	12.4	65.4	20.3	108.45	97.5	234
												32.158

Formula of slag -- Al₂O₃ 16(FeO, CaO) 10 SiO₂

" " " Matte Cu₃ S₂ Fe₁₇ or 4(Cu₂S) 17(FeS)

The ^{above} percentages in the matte do not agree with those given before owing to the fact that in the other calculation I did not take the copper in the slag into account.