Smelting of a Dry Silver Ore from Butte Montana

No. 1339

E. L. Hamilton

1891

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Contents continued

Note. -

In order to make the description clearer and to bring out the main facts of my work better, I have added an appendix giving the details and records ~~of~~ the runs in it; thus avoiding complications in the general description

E. L. Hamilton.

1891.

Smelting
of
A Dry Silver Ore
from
Butte Montana

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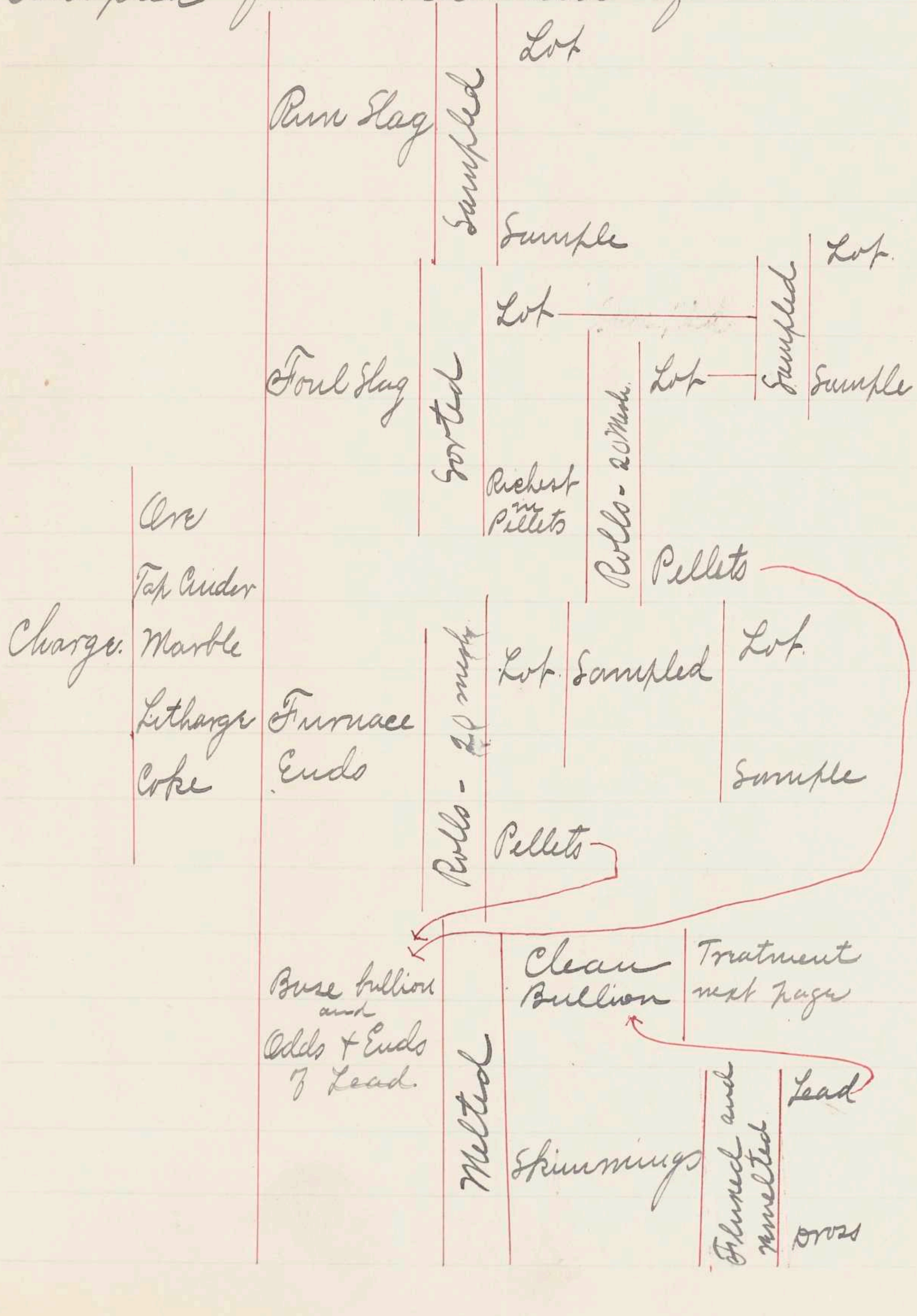
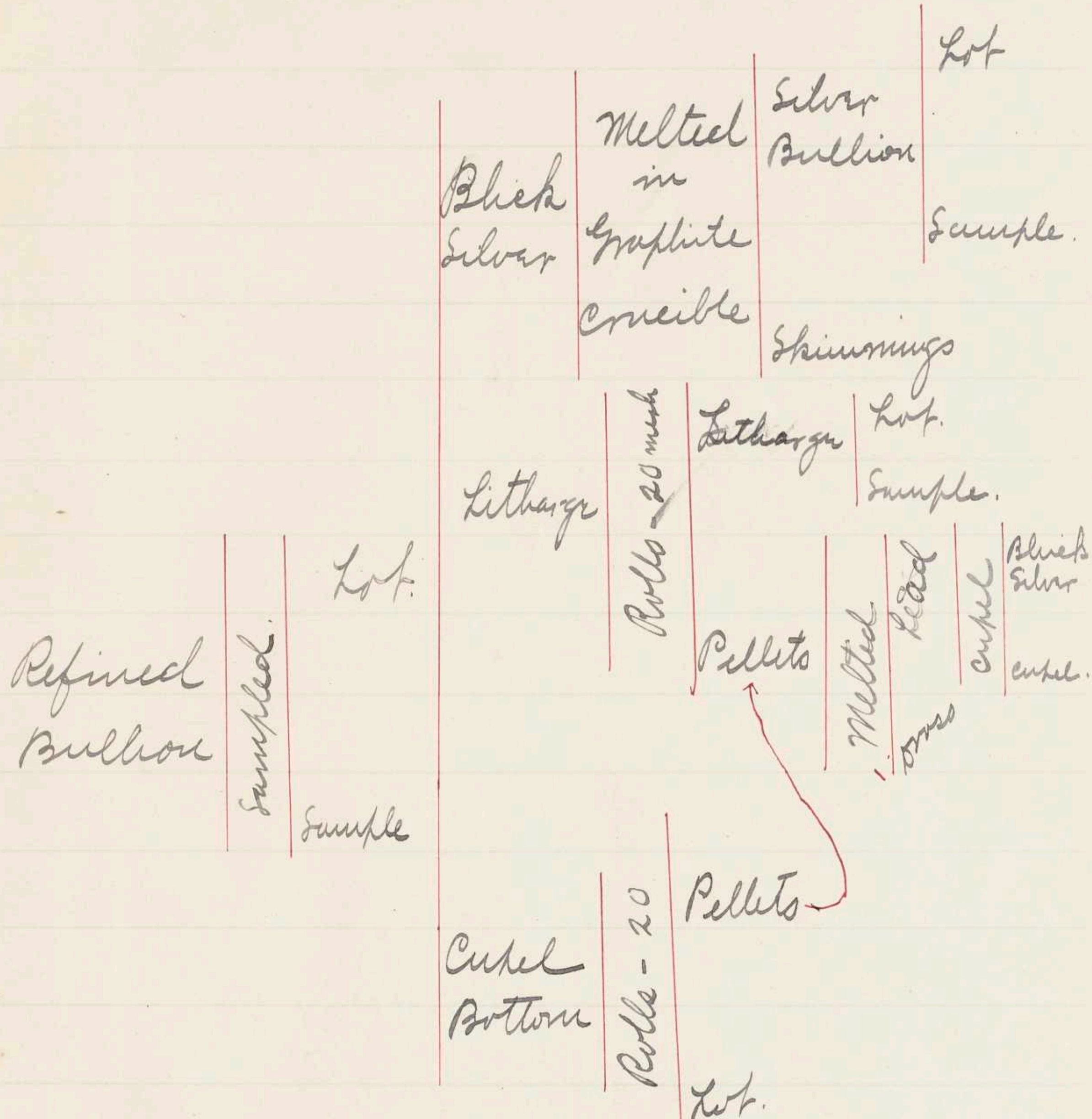
Scheme
Adopted for Treatment of Ore.



Scheme Continued.



- Description of Ore -

No 1339

The ore, the treatment of which forms the subject of my thesis, was sent to the Institute by Mr. Chas. W. Goodale of Butte Montana, to be treated by the pan amalgamation process. Mr Sonneman '90 began work in that line and after his work was well along, Mr Goodale sent word that he would like to have the ore treated in the blast furnace, which I undertook.

The ore is a dry ~~carbonate~~ silver ore from Butte Montana and consists chiefly of carbonate of manganese (rhodochrosite) and quartz with only .66 % of lead. It varied in size from irregular pieces 1" in diameter to pieces the size of a walnut, there being only a small amount of fines

- Sampling Ore -

The ore was first emptied on the floor and thoroughly mixed by repeated shoveling from one place to another. It was then quartered and the first and third quarters taken and mixed and quartered again, this time taking the second and fourth quarters which were again mixed and quartered. These operation were repeated until the amount of ore from the last quartering amounted to about two hods-full. This was put through the Gates crusher, sampled down to 3/4 of a hod full and then put through the rolls using a 40 mesh sieve. After the ore had passed through the sieve it was sampled down to a small bottle full and then put through a 60 mesh sieve by means of the bucking board. It was then set aside for assays and analysis

– Composition of the Ore –

SiO_2	38.40
FeO	6-80
Al_2O_3	.89
MnO	24.56
CaO	2.1
CO_2	16.08
S.	4.3
Pb	.68
Cu	Trace

Fluxing and Slag materials

1. Tap Cinder – This was used in order to furnish iron

to take care of the silica in the ore and fluxes. It was sampled down to two hods full, passed through the Blake & Gates Crushers and a sample made in the same manner as the ore sample was made.

2 - Marble.

This was added to give a liquid slag and to aid in setting iron free from the tap cinder, A sample was obtained in a similar way to that of the tap cinder and set aside for analysis

3. Litharge.

As there was practically no lead in the ore, litharge was used in order to furnish lead to collect the precious metals The litharge I used consisted.

of cupel bottoms (soaked with litharge) and various odds and ends from the assaying labra-tory, and on that account it was difficult to get a fair sample. It was quartered to a 3/4 of a hod full and sampled in the usual way. The pellets on the screen from rolls ~~and~~ ~~was~~ reckoned as so much me-chanical lead in the litharge. The sample was set aside with the other samples for analysis.

4.— Coke Ash.

Coke produces an ash which helps form the slag and it there-for has to be taken into account in the slag calculation. The quantity of ash was reckoned as 13.2% of the coke and an analysis made of the ash.

Having taken samples of the ore and materials to be charged in the furnace I next made the complete or partial analyses as required, of the different samples in order to get data for making the slag calculation.

Mr. James found from experience that a slag with 32% SiO_2, 27% Fe + Mn. or 35% of FeO, and 23% CaO gave the best results and I therefore aimed in my slag calculation to get a slag as near that as possible (For calculations see Appendix p. - 41)

The furnace charges as computed in the slag calculation are as follows:-

	Kilos
Ore	50
Top Cinder	10.65
Marble	24.45

Furnace charge continued

Litharge	16.65
Coke	16.50
Total	118.25

The fuel charge was based upon a consumption of 6 parts of charge mixture to 1 of coke, and it was found sufficient to keep a good liquid slag.

Blast Furnace Run.-
(For details and records see appendix p's 32,56)

The action in the blast furnace is reducing. The fuel burns to carbonic acid, CO_2, but is immediately reduced to carbonic oxide, CO, in the presence of an excess of carbon. This carbonic oxide, CO, and carbon act as a reducing agents, reducing the metals from a higher to a lower state of oxidation, or to a metallic state. The earthy matters, silica, ferrous oxide etc, go into the slag and are thus separated from the metallic lead. The silver and gold go into the lead.

The furnace used was the small cupola furnace in the mining laboratory, a description and drawings of which can be found in Mr. Sully's ('88) thesis, on lead smelting. Previous to the run I put the iron bottom in, cemented it and made a bottom of steep, which was dried by a small fire the day previous to the run. The fire was started with wood and charcoal and the furnace was then filled with coke to about two feet above the tuyeres. The blast was turned on and the furnace blown in with two loads of Brown's rich lead slag. At 9. A M the first regular charge was added. Charging ended at 3.30 P.M. Tap hole was plugged at 9.12. First appearance of slag at 9.33, of lead at 10.25. The furnace was tapped about every fifteen minutes. Last tap was made at 4.11.; Blast off at 4.13. At 4.15 breast was ~~to~~ torn down and at 4.30 the fire was hauled. The run ended at 4.30. Smelting lasted from 7.45 A.M. (when first coke was added) to 4.30 P.M. a period of 8 hours and 45 minutes. The highest blast pressure.

was 6.50 oz. the lowest 2.50 and the average 4.30 oz. The slag was very liquid and the foul slag contained quite a large amount of shot. Throughout the run globules appeared on the surface of the slag, both in the small and large buggies, and these globules appeared to drive just like lead globules, but as the slag in the large buggies contained hardly at any lead shot, they could not have been lead. Prof. Richards thought that there might be two slags of different specific gravities or that the globules might be a matte of iron or manganese. I did not have time to get the specific gravity of the bottom and middle parts of the run slag (that is the slag in the larger buggies) - but an analysis of the bottom and middle and also of the running sample showed about the same percentages of iron, silica and sulphur, which goes to show that the slag was of about the same composition throughout and leaves the solution of the globules unsolved. The run was free from hitches of any sort, due to the easy fusibility of the ore and to the good liquid slag formed.

The materials charged are given on the following pages.

Materials Charged.

	Kilos
Brown's Lead Slag	30.60
Coke to start fire	54.4
Ore	612.00
Tap Cinder	130.30
Marble	298.40
Litharge	202.80
Coke	207.80 ?
Slag recharged	165.70
Total	1701.50
Slag recharged	165.70
Total charge =	1535.80

See page 28

Products were as follows:-

Slag, 1st two taps thrown away	33.60
Foul slag (small buggies)	183.255
Run " (Large ")	599.85
Furnace Ends	50.74
Base bullion	75.92
Coke unburned	31.28
	974.645

- Treatment of Products -

(1) The Slag from the first two taps was thrown away as it was probably fouled by the Brown's lead slag used in blowing in the furnace.

2 - Foul Slag - This was sorted and the richest part of it containing lead shot, and amounting to about two hods full were passed through the rolls and the lead pellets collected on a limiting sieve of 20 mesh. The pellets were added to the base bullion. The fine slag which passed through the sieve was added to the main lot of foul slag and the whole sampled, and the sample assayed for lead, silver and

gold and analyzed for lead.
3 - Run Slag - This was thought to consist of two slags of different specific gravity or to contain a matte of iron or manganese at the bottom, so a grab-sample was taken of the bottom and also of the middle of the cakes and a partial analysis made of each sample. During the run a running sample was taken in the following way; - at each tap, when the slag overflowed the small buggy and ran into the large one, a little was allowed to run on a shovel and quickly plunged into water in order to get a glassy sample. This was passed through the rolls and then through a 60-mesh sieve and a complete analysis made of it.

4- Furnace Ends - The bottom of the furnace made of steep naturally absorbs considerable lead and that part of the bottom that contained any lead was sorted out, and together with any odds & ends that could not be classed otherwise, made up what is called furnace ends. These were passed through the rolls and the lead collected on a limiting sieve of 20-mesh. The lead was added to the base bullion, and the lot was sampled and analized for lead and assayed for lead gold and silver

5- Unburned coke. This was used simply to help account for the material charged.

6- Base bullion - consisting of 61.27 kilos of lead cakes, 14.65 kilos of lead in pellets and lead from the bottom of the furnace, was melted down in a black lead crucible, the dross skimmed off and the lead ladled out and poured into moulds The skimmings with a small amount of furnace ends rich in pellets, was fluxed with Na_2SO_4, glass and charcoal and smelted and poured into a large buggy. The result was a cake of lead with a 3/4-in matte on top, the matte coming from the sulphur in the salt cake, and 10.10 kilos of slag containing 10.32% of lead. The total lead from this melting amounted to 67.12 kilos and this was cupelled for the silver & gold

Cupelling Run. — Details p 61

Owing to the small amount of lead to desilverize and the limited amount of time on my hands, I decided to cupel the lead in preference to treating it by the Parkes-process. Desilverization by the cupelling process consists of an oxydizing action by which the lead is oxidized to litharge and either absorbed or drawn off, leaving the gold and silver, which are less easily oxidized, behind. The furnace used, was the reverberatory furnace in the mining laboratory with a fixed roof and movable hearth. The furnace has four blasts of air, an undergrate blast, one over the grates, one in the roof over the hearth and one coming directly on to the.

surface of the bath from behind and at an angle. The hearth was put in place the day previous to the run and a small fire kept up to dry it. At 8. A.M. a coal fire was started and at 9.30 the first charge was put in. Last charge was put in at 11.40; at 2.45 nearly all the lead was oxidized; at 2.55 the silver blicked, and at 3.05 it was taken out. The run lasted from 9.30 A.M to 3.05 P.M. a period of 5 hours and 35 minutes. The charging period was 2 hours and 10 minutes and the period of oxidation after the last charge was added was 3 hours and 15 minutes - Owing to the hearth being poorly made about 2.719 kilos of lead leaked out into the litharge. After considerable experimenting with the different blasts we found that, with the surface blast full on, the undergrate about 1/4 and the top blasts on about 1/2, we got obtained the best oxidizing and driving action. Boiling occured during the entire run.

Materials Charged.

1. 67.12 kilos of lead

– Products –

1	Blick Silver	.319 kilos
2	Litharge	48.7725
3	Cupel-bottom	20.906

– Treatment of the Products –

1 – Blick Silver – The bottom of the hearth was examined and all the small buttons of silver

were picked out and added to the large lump of thick silver, which had been freed from slag etc by hammering. The silver was then put into a number 3 graphite crucible and melted down, a little borax and glass was added and the skimmings taken off clean with an iron rod with a disk on the bottom; the silver was next poured into an ingot mould previously greased and charcoal added on top of the hot silver. The silver was taken out of the mould hot, and cold water poured on it. I repeated the melting operation three times the result being a very pure ingot of silver, which was bored to get a sample and the sample was assayed.

2 - Litharge - This was passed through the rolls and the lead pellets collected on a 20-mesh sieve. The lot was sampled and the sample assayed for gold and silver. I endeavored to recover the silver in the pellets by making a cupel and treating the lead in ~~the~~ small reverberatory furnace with a movable top. Instead of first melting the pellets and getting the lead in the form of an ingot, I put them directly on the cupel in the furnace, and in spite of the high heat which Prof. Richards and I obtained with a small 1/2" upper surface blast and an undergrate blast, we were unable to remove the scum which formed on top of the

lead, and therefore we could not get any oxidizing action. This scum was so thick that we thought it was the bottom of the cupel, and that the cupel had broken and the lead had all run through, so we cooled the furnace. When I took out the cupel, I found the lead was still in the bottom, so I collected it and melted it into an ingot. I next made a cupel and attempted to cupel the lead in a pot furnace, but before adding the lead, the cupel ring and sheet iron on which the cupel was placed, warped from the high heat required and broke the cupel. My next attempt was to make a cupel which just fitted in the muffle. Unfortunately the bottom of the muffle was poor, and after the lead had been driving nicely for about 1/2 an hour the bottom of the muffle broke, cracking the cupel which allowed the lead to run through onto the ash pit. I collected the lead and remelted it into an ingot and turned it over to Mr. Lodge. After my first experiment failed, I assayed the lead for gold and silver so that I knew how much gold and silver it contained even if I didn't recover it in the metallic state.

3 - Cupel - Bottom - This was passed through the rolls, the pellets collected and added to those from the litharge. The lot was sampled and assayed for gold & silver

- Results -

- Blast Furnace -

Table I

	Kilos
Total Raw material charged	1535.80
Total Products	974.65
Lost by weight	561.55
Loss in per cent	~~36.53~~

? This is not correct See page 16

Of that 36.53% loss (up the chimney & in handling), 13.02% was due to the carbon of the coke; 8.07 to CO_2 in marble. and 6.44 to CO_2 in the ore making 27.53% of the loss,-accounted for in loss due to the burnt carbon of the coke, and CO_2 in the marble and ore. and leaving a net loss of 9% due to fumes flue dust handling etc. Of that 9%-, 3.10% was due to loss in lead fumes, making a loss 5.9% due to dusting handling etc.

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Table II of Results.

Feed to the Blast Furnace

	Weight in Kilos	% Lead	Wt in Kilos of Lead	oz silver per ton	oz au per ton	% ag	% au	total oz ag	total oz au.	Wt. ag in Kilos	Wt au in Kilos
Ore	612.00	0.68	4.16	12.30	.16	.04217	.00055	8.2983	.1082	.258081	.003366
Litharge	202.80	73.94	149.95	23.42	.385	.0802	.00132	5.229	.0925	.162646	.002877
Total			154.11					13.5273	.2007	.420727	.006243
				Products of the Blast Furnace							
Base bullion	67.12	—	67.12	158.00	12.77	.5417	.00438	11.691	.0945	.363589	.0029699
Run Slag	599.85	3.52	21.12	0.66	.09	.00226	.000308	.7265	.05939	.022597	.0014754
Foul Slag	183.255	4.17	7.64	1.02	.11	.0035	.00037	.20705	.0218	.0064392	.0067804
Furnace Ends	50.74	18.90	9.59	14.08	.05	.048	.0017	7831	00277	.024355	.0000863
Dross from melting bullion	10.10	10.32	1.01								
Total			106.48					13.4076	.17846	.416980	.011302
				Feed to the Cupel.							
Base bullion	67.12	—	67.12	158.00	12.77	.5417	.00438	11.691	.0945	.363589	.0029699
				Product of the Cupel Run.							
Litharge	48.725	—	—	3.966	—	.0136	—	.2131	—	.006626	—
Cupel bottom	20.906	—	—	16.65	—	0571	—	.3838	—	0119373	—
Lead from Litharge	1.28	—	—	526.164	—	01804	—	.7427	—	.0231	—
Silver bullion	.319	—	—	—	—	—	—	10.26	.0127	.319	—
	—	—	—	—	—	—	—	11.5996	—	360.65	—

Results continued.

From the tables we find that :-

		Kilos
(1) Total lead in charge	=	154.11
" " " products	=	106.48
Loss by weight	=	47.63
" in per cent	=	30.91
% of lead accounted for in bullion	=	43.56
" " " " by-products	=	25.53

2. The per cent of total charge lost in lead fumes = 3.10%.

3 - Total silver in charge	=	13.5273
" " " products	=	13.4076
Loss by weight	=	.1197
" in %	=	.885
% of silver accounted for in bullion	=	84.63
" " " " by-products	=	12.685

Results continued.

4.— Total gold in charge = .2007 oz

" " " products = .17846

Loss by weight = .02224

% lost = 1.108

% of gold accounted for in bullion = 47.09

" " " " byproduct = 51.80

Cupel-Run.

5- Total silver in bullion = 11.691

" " " products = 11.5996

Loss by weight = .0914

" in % = .782

% of silver accounted for in bullion = 87.76

" " " " by-products = 14.45

6. — Total gold in bullion = .0845

" " " products = .0127

The gold was almost wholly lost or else the deficiencies are due to inaccurate assays.

Results continued

Refined-Bullion bar.

According to my assays I found the fineness of the bar to be as follows.

Gold finess = 1.255

Silver " = 995.87

Total " = 997.126

Mr Lodge found the finess to be, I think he said of silver alone, 998.00

- Money value of silver -

Owing to the poor results on gold and also to the very small amounts of it I have neglected to report gold in making the money value of the bar.

Silver, 10.26 oz at $1.00 per oz = $ 10.26

Silver 13.53 oz (theoretical) at $1.00 per oz = $ 13.527

Lost $ 2.267

Loss in % during entire work = 24.16

These results lead one to conclude that for dry silver ores, the small cupola furnace gives very good results on silver and gold as well, (my results didn't show it on gold) but for lead the furnace is too short and great loss results in fumes flue dust etc.

Appendix -

Mineralogical composition of the ore.

Main minerals:-

Rhodochrosite	$MnCO_3$
Galena	PbS
Pyrite	FeS_2
Sphalerite	ZnS.
Quartz	SiO_2
Calcite	$CaCO_3$

Possibly a little granite and feldspar are present.

— Analyses —

Ore

SiO_2	38.40	MnO	24.56
FeO	6.80	Mn	16.08
Fe	5.30	S	4.3
Al_2O_3	.89	Pb	.68
CaO	2.1	Cu	trace

Analyses continued

Tap Cinder.		Marble.
SiO_2	12.67	2.70
FeO	80.01	.34
CaO	.30	52.88
CO_2	—	41.55
Fe	62.23	
Al_2O_3	3.82	
MnO	2.91	
Mn	2.25	
S	.41	

C Coke Ash and Coke.
Coke ash was analyzed by Mr. Farr.

~~Coke~~ Ash

SiO_2	= 47.60	Fe_2O_3	= 77.71
$Al_2O_3 + Fe_2O_3$	= 46.11	Al_2O_3	= 28.40
Fe	= 12.40		
CaO	= 1.5		

Coke ash = 13.20 % of coke

Analyses cont.

Coke contains

SiO_2	6.27
Fe	1.66
CaO	.20
Al_2O_3	3.75

Litharge

SiO_2	3.49	Pb	73.94
FeO	2.04	CuO	trace
CaO	4.48	CO_2	3.52

Rever Slag.

	Running Sample -	Middle of R.S.	Bottom
SiO_2	32.30	32.91	31.87
Fe	14.49	14.26	13.83
S	.36	.32	0.38
FeO	18.98		
Al_2O_3	1.20		
MnO	19.59		
Pb	3.52		
CaO	20.16		
Cu	.18		

Analyses concluded

Foul slag simply analyzed for lead.

Pb = 4.17 %

Furnace Ends

Pb = 18.90 %

Base bullion

Cu = 1.90 %

Assays

- Assays -

Products	Method	Oz per ton Ag.	Oz per ton Au	% lead
Ore	Scorif.	12.30	.16	
"	"	12.00	.12	
"	"	11.90	.11	
"	"	12.10	.14	
"	cru.	11.68	13	
"	"	12.01	.16	
Feed Litharge	cru.	23.42	.385	
"	"	23.07	.332	
"	"	23.02	—	
Base bullion	Scorif.	158.00	10.67	
" "	"	157.10	11.93	
" "	"	159.00	12.27	
Run Slag	cru.	.65	.12	
"	"	.66	.09	2.32
"	"	.65	.08	1.96
"	"	.61		

Assays concluded

Products	Method	oz per ton Ag	oz per ton Au.	% Lead.
Foul Slag	em.	1.02	.10	2.37
" "	"	.93	.11	2.11
" "	"	.86		
Furnace E.	"	13.08		
"	"	13.75	.05	17.87
"	"	13.23	.07	17.64
"	"	14.02	.04	
Dross	"	—		10.32
Cupel-litharge	"	3.616		
"	"	3.966		
Cupel bottom	"	15.11		
"	"	16.68		
"	"	16.16		
Collected lead-lith-	Scorif.	516.154		
		Fineness	Fineness	
Silver bullion	cupel.	977.126	1.265	
" "	"	996.18	1.255	

Slag-calculation -

In order to determine the fluxes to be used in a blast furnace an analysis of the ore must be made. The analysis of my ore showed that tap-cinder, marble litharge and coke would be probably make a good slag. After determining upon the fluxes, I constructed from the analyses the following composition table.

I Composition table.

	SiO_2	Fe	CaO	Mn	Al_2O_3	S	Cu	Pb
Ore.	38.40	5.30	2.1	19.03	.89	4.3	—	.68
Tap-cinder	12.67	62.23	.3	2.25	3.82	.4	—	—
Marble	2.70	0.33	52.88	—	—	—	—	—
Litharge	3.49	1.58	4.48	—	—	—	5.60	73.94
Coke	6.27	1.66	.20	—	3.75	—	—	—

Slag-Calculation Continued.

Mr James after considerable experience found the following slag to give the best results. I have therefore aimed in my calculations to approximate as near to his slag as possible. - His slag is as follows.-:

(1) SiO_2	32 %			32
Fe + Mn	27	or	FeO	35 %
CaO	23			23
Others	18	or		10

As a basis for my calculation I took 800 kilograms of ore to work from, and from the composition table found that 800 kilos of ore contained :-

(2) - SiO_2	Fe	CaO	Mn.	Al_2O_3	S
307.20	42.40	16.80	152.24	7.12	34.4

My next step was to find the amount of SiO_2 in excess of the

- Slag-calculation cont. -

iron and manganese in ore

32 : 27 = 307.20 : x

x = 259.20

152.24 + 42.40 = 194.64 from (2)

∴ 259.20 − 194.64 = 64.56 = excess of SiO_2 over Fe + Mn in ore.

Now find the excess of Fe + Mn over SiO_2 in tap cinder.

32 : 27 = 12.67 : x

x = 10.69 Fe + Mn

(1) 62.23 + 2.25 = 64.48

64.48 − 10.69 = 53.79 = excess of Fe + Mn over SiO_2 in tap cinder.

Now if 100 tap-cinder gives an excess of 53.79 of Fe + Mn over SiO_2 in tap-cinder then to supply the deficit of 64.56 of Fe + Mn in ore we obtain from the proportion

53.79 : 64.56 = 100 : x

x = 120

Slag-calculation cont.

Therefore 120 kilos of tap-cinder will furnish sufficient Fe + Mn to equalize the SiO_2 x's in ore. Next find excess of CaO in marble over SiO_2

$32 : 23 = 2.70 : x$

$x = 1.94$

amount CaO in marble = 52.88
1.94

x's of CaO over SiO_2 = 50.94

Now find the amount of CaO necessary to take care of SiO_2 in tap cinder and ore

SiO_2 in ore = 307.20 kilos
SiO_2 in Tap-cinder = 15.20
322.40

$32 : 23 = 322.40 : x$

$x = 231.70$

Reckoning Al_2O_3 and CaO together in ore and tap cinder we have

Slag-calculation cont.

20.40 + 11.70 = 32.10

231.70 − 32.10 = 199.60 = the amount of SiO_2 in excess of CaO and Al_2O_3 in ore and tap-cinder. We just saw that in 100 parts of marble the excess of CaO over SiO_2 was 50.94

∴ 50.94 : 100 = 199 : X

X = 391.83 kilos of marble necessary to equalize the SiO_2 in tap cinder and ore.

Thus far the charge equals

Ore	800
Tap cinder	120
Marble	391.83
	1311.83 or about 1312.

The litharge necessary to collect the gold and silver is assumed to be about 15% of the charge, which is 15% of

Slag calculation continued.
1312 or 266.43 kilos of litharge and the coke ratio is about 1 of coke to 6 of charge.
∴ 800 + 120 + 391.83 + 266.43 = 1578.26 ÷ 6 = 263.04 approximately 264 which is the coke necessary for the charge.
The next step was to calculate from the composition table the amounts of the ingredients in each material in the charge based on the above approximate amounts.

II

	Wt.	SiO_2	Fe	Mn	CaO	Al_2O_3
Ore	800	307.20	42.4	152.24	16.8	7.12
Tap cinder	120	15.20	74.68	270	3.6	4.58
Marble	391.83	10.58	1.33	—	207.20	—
Litharge	266.43	9.29	4.20	—	1.93	—
Coke	264	16.55	4.38	—	.53	9.90
	1842.26	358.82	126.99	154.94	240.06	21.60

Slag-calculation continued.

Total SiO_2 = = 358.82
" Fe + Mn = 281.93
" $CaO + Al_2O_3$ = 261.66
902.41

Since there was 18% of other ingredients in Mr James' slag 902.41 is only 82% of the whole

$\therefore \frac{902.41}{82} = 1100.49$ = whole charge

by first approximation.

Having approximately the amount of the whole charge and the sums of the different ingredients in all the materials it only remains to find the ingredients in the whole charge

$\frac{358.82}{1100.49} = 32.62$ for SiO_2

$\frac{281.93}{1100.49} = 25.61$ for Fe + Mn

$\frac{261.66}{1100.49} = 23.78$ for CaO.

Slag-calculation continued. -
From the first approximation we see that SiO_2 & CaO are higher than in Mr James slag and that Fe + Mn is lower. By adding more tap-cinder to the charge we decrease the SiO_2 and CaO and increase the Fe + Mn. i.e. I added about 20 kilos of tap-cinder and reckoned the percentages CaO above by adding to the sums of the different ingredients the increase of each ingredient due to the increase of 20 K's of tap-cinder

SiO_2	Fe + Mn	CaO + Al_2O_3
358.82	281.93	261.66
2.53	12.90	82
361.35	294.83	262.48
294.83		
262.48		
.82 ⎣918.66		
1120.32		

Slag calculation continued.

$$\frac{361.35}{1120.32} = 32.25\% \; SiO_2$$

$$\frac{294.83}{1120.32} = 26.32 \; Fe + Mn$$

$$\frac{262.48}{1120.32} = 23.43 \; CaO.$$

This shows that still more tap-cinder must be added.
Added 10 more kilos.

SiO_2	Fe + Mn	$CaO + Al_2O_3$
361.35	294.83	262.48
1.27	6.45	.412
362.62	301.28	262.89

362.62
301.28
262.89
.82 926.79
1130.23

$$\frac{362.62}{1130.23} = 32.08 \; SiO_2$$

$$\frac{301.28}{1130.23} = 26.66 \; Fe + Mn$$

$$\frac{262.87}{1130.23} = 23.26 \; CaO.$$

Slag-calculation continued
IV and final approximation
Added 20 more tap-cinder

SiO_2	Fe + Mn	$CaO + Al_2O_3$
362.62	301.28	262.48
2.54	12.90	.824
365.16	314.18	263.304

365.16
314.18
263.30

.82) 942.64
1149.51

$\frac{365.16}{1149.51} = 31.77\%\ SiO_2$

$\frac{314.18}{1149.51} = 27.33\%$ Fe + Mn

$\frac{263.30}{1149.51} = 22.90\%$ CaO

Thus my final calculation gave me a slag with the following percentages:

		actual slag
SiO_2	31.77	32.30
Fe + Mn	27.30	29.66
CaO	22.90	20.16

Calculation of Charge.

Ore	= 800 Kilos	Obtained from Slag-calculation
Marble	= 391.83	
Litharge	= 264.43	
Coke	= 264.00	
Top-cinder	= 170.48	

$$\frac{170.48}{800} = .213$$

$$\frac{391.83}{800} = .489$$

$$\frac{264.00}{800} = .330$$

$$\frac{264.43}{800} = .333$$

Then the charge equals

Top-cinder	.213 × 50	= 10.65
Marble	.489 × 50	= 24.45
Litharge	.333 × 50	= 16.65
Coke	.33 × 50	= 16.50
Ore	50 × 1.00	= 50
		118.25

Preparation of furnace.

The day previous to the run the iron bottom was set in place and cemented there with mortar. Steep moistened just so as to hold together when squeezed in the hand, was put into the furnace and tamped to within a couple inches of the tuyer bowl. A crucible was cut out of this solid bottom, starting 2 inches below the tuyers and sloping towards the center, and extending about 2" below the tap hole so as form a basin for the lead. The breast wall was next put in place leaving the tap hole open and a small fire started to dry the crucible.

Blast furnace Run cont.

Calculated charge

Ore	50 kilos
Tap-cinder	10.65
Marble	24.45
Litharge	16.65
Coke	16.50
	118.25

Shifts:

No of shift	Time	Feed	Tap.
1	8.30 – 12.30	Haskings	Hamilton
		Fraser	Bennett
2.	12.30 – 4.30	Harris	Weston
		Stoddard	Lincoln.

Firing furnace.

March 26th 1891.

At 7.30 A.M fire started with shavings wood and charcoal.

7.45 A.M 1 hods coke (16 kilos) added
8.00 A.M. 1 " " (" ") "
8.45 A.M 2 " " (32 ") "

Charging Record. B. F. Run.

March 26th 1891.

Time of Charging	Ore K.	T.C K.	Marble K	Litharge K	Coke K	Slag K.	Remarks
A. M. 9.00	50.	10.65	24.45	16.65	[illegible]	30.60	Brown's lead slag for blowing in
9.20	50.	10.65	24.45	16.65	16.50		
9.27	"	"	"	"	"		
9.47	"	"	"	"	"		
10.27	"	"	"	"	"		
11.00	"	"	"	"	"		
11.32	"	"	"	"	"		
12.03	"	"	"	"	"		
12.31	"	"	"	"	"		
1.05	"	"	"	"	"		
1.35	"	"	"	"	"		
2.05	"	"	"	"	"		
2.10	"	"	"	"	"	12.5	Foul slag from trough.
2.30	12	2.5	6.30	4.50	4.25		
2.40					6.3	57.2	Last coke
3.15						36.6	All ore in
3.30						24.8	Foul slag.
3.40						33	" "
3.50						21.6	" "

Materials Charged

Brown's lead slag	30.60
Coke to start	54.40
Ore	612.00
Tap-cinder	130.30
Marble	298.40
Litharge	202.80
Coke	207.30
Slag recharged	165.70
	1701.50
Slag recharged	165.70
	1535.80

Total charging time for ore charges = 6 hours.
Average rate of charging for full charge was 25 minutes
Rate of charging ore was 120 kilos per hour.

Blast furnace run.
Tap-Record.

Time Tapped	Time Plugged	Interval tapping	Blast Pres. oz.	Temp.	Remarks.
A. M. 9.30	9.33	10	5.00	126°	Good free flow, & large
9.43	9.45	10	5.75	134	Same
9.55	9.57	10	4.00	185	Same
10.07	10.09	16	2.75	170	Smaller - globules
10.25	10.27	18	2.50	190	Largest yet
10.45	10.47	18	3.50	175	Lead observed
11.05	11.07	18	4.00	168	Largest flow. thicker
11.25	11.30	17	4.00	166	Large lead flow.
11.47	11.50	10	6.00	178	Slag liquid, flow free
12.00	12.02	13	5.75	194	Great many globules on surface
12.15	12.17	15	4.25	180	Same
12.32	12.39	14	3.25	172	Same
12.52	12.55	21	3.50	176	Globules Large free flow
1.16	1.24	15	6.50	196	Same
1.39	1.42	13	4.50	134	"
1.55	2.02	16	6.25	114	"
2.18	2.21	19	6.50	128	Largest flow yet
2.40	2.43	17	6.02	178	Still globules

Blast furnace run
Tap-record continued.

Time Tap.	Time plug	Interval	Blast oz.	Temperature	Remarks
327	330	24	3.25	180	Large liquid flow globules
348	355	18	2.75	170	Extra large flow
400	4.05	.05			Medium "
411		.06			Small pot half full.
413	-		-	-	Blast off.
415					Began to tear out breast
430					Fire braked
4.30					End of run.

Additional remarks on the run.
Throughout the entire run globules were noticed on the slag in both buggies, which appeared to drive just like lead globules, but the slags showed too few pellets to account for the number of globules which entered the large buggies. Chemical analysis showed about the same percents

- Remarks on blast furnace run -

of SiO_2, Fe and sulphur in the samples taken from the bottom and middle of the cakes from large buggies and also in the running sample, so that the theory that the bottom of the slag was a matte of iron or manganese is not sustained.

At 3.50 I put a cold flat iron rod in the top of the furnace and considerable fume condensed on it. At 2.21 the tuyers were stopped in order to have the engine oiled. Occasionally the tuyers became choked. They were freed by poking an iron rod in, or the choked tuyer was stopped for a short time until it had cleared

Blast furnace run cont.

itself. At times a blue flame was noticed at the tuyers which indicated time to tap.
At 8.35 A.M. the ~~front~~ tuyar was inserted and at 9.00 it was removed. At 9.07 the hole in front was inserted. At 9.12 the Al tuyars were inserted and the tap hole plugged.

Products.

Slag 1st 2 taps thrown away	33.60
Foul slag	599.85
Rim "	50.74
Base bullion	75.92
Coke unburned	81.28
	974.28.

14.65 kilos of lead were collected from slag, furnace ends & other sources..

Bloom Run continued.

1.285	Kilos	Pellets	from	foul slag
1.625	"	"	"	trough
5.810	"	"	from	furnace Ends
5.930	O "	"		odds & ends
14.65				

I attribute the smooth manner in which the furnace ran, to the fact that the actual slag came very near to the slag of Mr James. which is a very free flowing liquid slag.

Cupel Run. April 9th 1891.

Preparation for the run.

The furnace used was the reverberatory furnace with a movable hearth, and it had four blasts, First an undergrate blast, secondly, an overgrate blast, thirdly a blast in the top of the furnace over the hearth and designated in my record as the top blast, and thirdly a surface or back blast which plays upon the surface of the bath at an angle from the back of the hearth.

Two days previous to the run a hearth was made from a mixture of 75% powdered marble, and 25% fire clay, moistened so as to just hold together when squeezed in the hand

Cupel Run continued

It was tamped hard as possible and in two layers.

The width of test-ring was 20" outside
" length " " " " 24" "
" nose projected 3"
Width of nose = 5"
Thickness of ring 1/2"

A basin was cut into hearth gradually sloping from a distance of about 3 inches from the outside edge, to the center.

Greatest width of basin = 14 1/2"
" length " " = 17 1/2
" depth " " = 2 1/8"

The day before the run the hearth was set in place and a small fire kindled to dry it.

Cupel - Record April 9th 1891.
A coal fire was started at 8 A.M.

Time	Charged	Remarks.
9.30	38.5 K	Good fire Undergrate & top blasts on
9.40	4. K.	
9.43	3.87	Little litharge forming
9.48		Top blast off surface on full
9.48-10.0		Experimenting with blasts
10		Top 1/4 on undergrate off, surface on.
10.10	9.30	Litharge forming well
10.12	1.20	Top on, undergrate off, surface 1/2 on
10.20-10.25	4.25	Litharge forming splendidly.
10-30-10.05	4.90	Fumes large
10.34		Top off surface on 1/2
10.40		Lead began to leak at nose
10.50		All blast off except surface 3/4 on
11.00		Lead stopped leaking.
11.05		Leaking commenced again
11.18 11.40	5.20	All lead in.. Lead stopped leaking
11.30		Surface on 3/4 top on 1/4 undergrate on a little

Cupel Record concluded.

Time	Charge	Remarks
12.10		Top off, back blast on full
1.10		Top on 1/4 back on 3/4
2.05		" " full " " 1/4
2.15		" " 1/2 " " full undergrate on 1/4
		Blast nearly all escaped through large pipe up to the present time
2.55		Silver blicked
3.05		Silver out.

Products.

Blick silver .319

Litharge 48.77

Cupel bottom 20.96

Run lasted from 9.30 A. M to 3.05 P. M a period of 5 hrs and 35 minutes

Charging period = 2 hrs and 10 "

Period of oxidation after last charge added = 3 hrs and 15 minutes

Before the charge of bullion

Cupel Run continued -
was added, litharge was sprinkled where the two layers came together in order to have it melt and fill up the cracks. -
Owing to the cupel having been poorly made, lead leaked out at the nose. Great care should be exercised in making the hearth.

Abstract of Thesis

E. L. Hamilton
1891.

Abstract Of Thesis

The work assigned me, by Prof. Richards, for my Thesis work, was the treatment of a Dry Silver Ore from Butte Montana, for the recovery of the precious metals. The ore consisted chiefly of carbonate of manganese and quartz, running 12.3 oz of silver per ton 0.16 oz gold per ton and 0.68 % lead.

My method of treatment was as follows:- The ore was fluxed with tap cinder, marble coke, and litharge, the litharge being added for the purpose of furnishing lead for the collection of the gold and silver. The ore properly fluxed to give a good

slag, was smelted in the blast furnace and the lead resulting from the ore and containing the precious metals, was melted down in a black lead crucible, the dross skimmed off and the melted lead poured into moulds.

It was then treated by the cupellation process, in the reverberatory furnace with a movable hearth. Desilverization by this method consists of an oxidizing action by which the lead is oxidized to litharge, and either absorbed by the hearth, or drawn off leaving the silver and gold, which are much less easily oxidized, behind. The silver obtained in this way is called blick-silver. It was melted down in a graphite crucible with a little borax and silica, the skimmings were removed, and the refined silver poured into an ingot-mould and then sampled by boring and assayed to determine its fineness.

I treated 612.00 kilograms of ore using 202.8 kilos. of litharge as the amount necessary to furnish sufficient lead for the collection of the silver and gold. That amount of ore contained 8.30 oz. of silver and 4.16 kilos of lead, and that amount of litharge contained 5.23 oz. of silver and 149.95 kilos of lead, making a total of 13.53 oz of silver and 154.11 kilos of lead in the charge to be accounted for.

In the blast furnace run 84.63% of the silver was recovered in the bullion and 12.685%

was accounted for in the by-products, leaving only .885 % as loss in fumes, flue-dust, handling etc.
43.56 % of the lead was recovered as available bullion, 25.53 % was accounted for in by-products and 30.91 % was lost in fumes flue dust handling etc.
In the cupellation run 67.12 kilos of refined bullion containing 11.69 oz of silver, were treated and of that amount of silver 87.76 % was recovered as refined silver, and 14.45 % was accounted for in by-products, leaving .782 % as loss by volatilization.
Of a total of 13.53 oz of silver in the blast furnace charge, 10.26 oz was obtained as marketable silver making a loss of only 24.16 % in the entire treatment, and most

of that loss is accounted for in
the by-products.
The silver brick was 998. thousandts fine.
In this abstract I have omitted
accounting for the gold as there was
such a small amount of it, that
the slight error in my assays
throw my calculations out
The conclusion arrived at is that
for gold and silver, in a dry carbonate ore, the small cupola
furnace gives very good results
but for lead it is too slow for
good work.

Respectfully submitted
E. L. Hamilton
May 18th 1891