

Notes on Electrolytic Lead

30

Mr. Addicks

March 15, 1928.

NOTES ON THREE LEAD REFINERIES.

U.S.S. Lead Refining Plant at East Chicago - Betts Process.
A.S. & R. " " " Omaha - Parkes and Betts Process.
Anaconda Lead Products at East Chicago - Sperry Process - White Lead.

Items	BETTS PLANTS		PARKES Omaha
	East Chicago	Omaha	
(1) Production per yr. Lead	37000 tons	16,000 tons	more than 200,000 tons
(2) " " " antimonial	2,000 tons		?
(3) " " Silver	5,000,000 ozs.		25,000,000 ozs.
(4) " " Bismuth	70,000 lbs.	220,000 lbs.	
(5) Analysis of Anodes	Lead	97.8%	98.4%
	Silver	100 to 200 ozs.	
	Gold	1/2 oz.	
	Bismuth	0.05 to 0.30%	1.5%
	Copper	0.025%	
	Arsenic	0.073%	
(6) Analysis of Cathode	Antimony	1.60%	
	Lead	99.9	99.9%
	Bismuth	0.0015	0.020
	Silver	?	?
(7) Loss of 100% acid per ton lead	8 lbs.	7 lbs.	
(8) Cost of 100% acid	10 to 12¢ per lb.	15 to 20¢ per lb.	
(9) " " H ₂ SO ₄	\$25.00 ton		
(10) " " Fluorspar	\$40.00 "		
(11) Composition of Acid	4200 lbs. spar		
	5500 " 66% H ₂ SO ₄		
	Heat to 110° and		
	pass thro closed tank of pure sand until saturated with silica		
(12) Composition of Electrolyte	?	10% lead 11% total acid of which 7% com- bined with lead and 4% free	
(13) Glue used per ton lead and cost	1 1/2 lbs. or equiv. goulac, a paper pulp byproduct	2 lbs. @ 15¢ lb.	
(14) Circulation of electrolyte	5 to 6 gals. per min. per tank	5 gals.	

- (15) Size of Tanks 2'6"x11'7"x3'9" deep Smaller
concrete 4" coating but will
3/8" asphalt asbestos be en-
see blueprint larged
- (16) Concrete tanks are first primed with a sticky sizing paint made
by Johns-Manville concrete primer. A pail of asphalt is heated in
a kettle and an equal weight of flake asbestos is stirred in and
the mix trowelled 3/8" all over the priming coat on tank while sticky
enough to bind the asphalt-asbestos mix firmly to the concrete.
When cool and hard this 3/8" covering is trowelled smooth by heating
surface slightly with blow torch. This coating is allowed to harden
several days and is then ready for use. It is considered a very
important discovery after many years of search. The flake asbestos
is #M 3 grade -carried by V. Mikesell Co., Chicago. The asphalt is
sold by Asphalt Products Co., Chicago.
- (17) Size of Anodes 2'x3'x1 1/2 at Smaller but
top tapering will be made
slightly larger
- (18) Wt. of Anodes 470 lbs.each
- (19) Spacing of " 1 think 4" centers 4 1/2" centers
- (20) Size of Cathodes 2'2"x3'2"x3/8"thick
weight 125 to 150 lbs.
- (21) Power required for electrolysis 1 KWH per 15 lbs.lead 1 KWH to 11 to 12 lbs.
- (22) Current density 16 amp. per sq.ft. 15 amps.
- (23) Voltage 0.48 per cell 0.65 per cell
- (24) Days in tank anodes 8 9
- (25) " " " cathodes 4 4 1/2
- (26) Lead loss less than 1% Less than 1%
- (27) Silver " None on commercial None on commercial
assays assays
- (28) Water wash and Evaporation ? 100 cu.ft. per day
each - one replaces
the other
- (29) Number of monthly men 35 645 Incl. Parkes plant
Average rate " " \$185.00 ?
Estimated monthly payroll \$5300.00 ?
Number day paymen 160 390 " " "
Average rate day pay men \$100.00 ?
Estimated payroll \$16000.00 ?
(30) Estimated payroll all \$6.50to7.00 per ton lead
" power @ 1¢ plus \$2.00 " " 1¢ plus
" acid loss \$1.00
Supplies &c ? \$1.00
Genl.Exp., Ins. mTaxes &c ?
Depreciation ?
(31) Loading By trolley trucks
35-100 lb.bars Also very efficient
One man load
90 tons per day on ry.cars

Some of the points that impressed me were as follows:

- (A) The acid loss of 7 to 8 lbs. and the cost of acid 10 to 20 cts. per lb. is less than I had understood. I was told that at Trail the loss is only 3 to 5 lbs. per ton of lead.
- (B) Losses of acid are considered to be largely mechanical, not due to decomposition. No gelatinous silica in slimes. Tank construction and lining very important in cutting down acid losses.
- (C) Glue is a very essential constituent of the electrolyte. It is required in increasing amounts with impurities in bullion. At East Chicago they use goulac with the glue and say it is more efficient. Trail uses French glue - Omaha uses Cudahy glue.
- (D) Lead produced at Omaha contains 0.02 bismuth because the anode contains 1.5% bismuth.
Lead produced at East Chicago contains 0.0015 bismuth because anodes contain only 0.05 to 0.30 bismuth.
A high bismuth anode gives a high bismuth cathode.
- (E) Both Betts plants soften bullion to remove arsenic and antimony before Bettsing and at Omaha they also zinc off the silver as they consider it cheaper to Parkes rather than recover silver from Betts slimes. Also say the tie up of silver in slimes is three times as long as when recovered by Parkes.
It costs Omaha \$1.00 per ton of lead to zinc off silver but they get back 84¢ of the dollar by making zinc chloride and selling. They use one tank for this chloridizing with chlorine gas.
Omaha has very efficient softening furnaces 13' x 30' inside lining - treating 300 tons of bullion per day. They consider best practice not to soften below 0.025%.
- (F) Mr. Betterton was of the opinion that a 96 to 97% bullion could not be treated by Betts process because of high acid loss and high costs for glue and other abnormally high expense. See Mr. Addicks letter of Jan. 30 for comments on this phase of matter.

In striking contrast to these two Betts plants is the Anaconda electrolyte plant at East Chicago. There the Sperry process is used to produce about 26 tons per day of what they claim is a superior quality of white lead - a basic lead carbonate having the formula $2 \text{PbCO}_3 \cdot \text{Pb}(\text{OH})_2$. This is sold to paint makers to be mixed with ore for sale and distribution.

The electrolyte used in this process is a mixture of sodium acetate and sodium carbonate. The process is described in detail in attached reprint of the Transactions of Am. Inst. of Mng. Engrs. Sept. 1925.

This process may be of interest to the Cerro Corp. for the reason that it is claimed that an impure anode may be used without affecting the quality of the white lead produced. The slimes resulting from the decomposition of the anodes are sent to the U.S. refinery at East Chicago for recovery of bismuth, silver and other elements - so that the process itself would have to be supplemented by a bismuth recovery plant the same as a Betts plant.

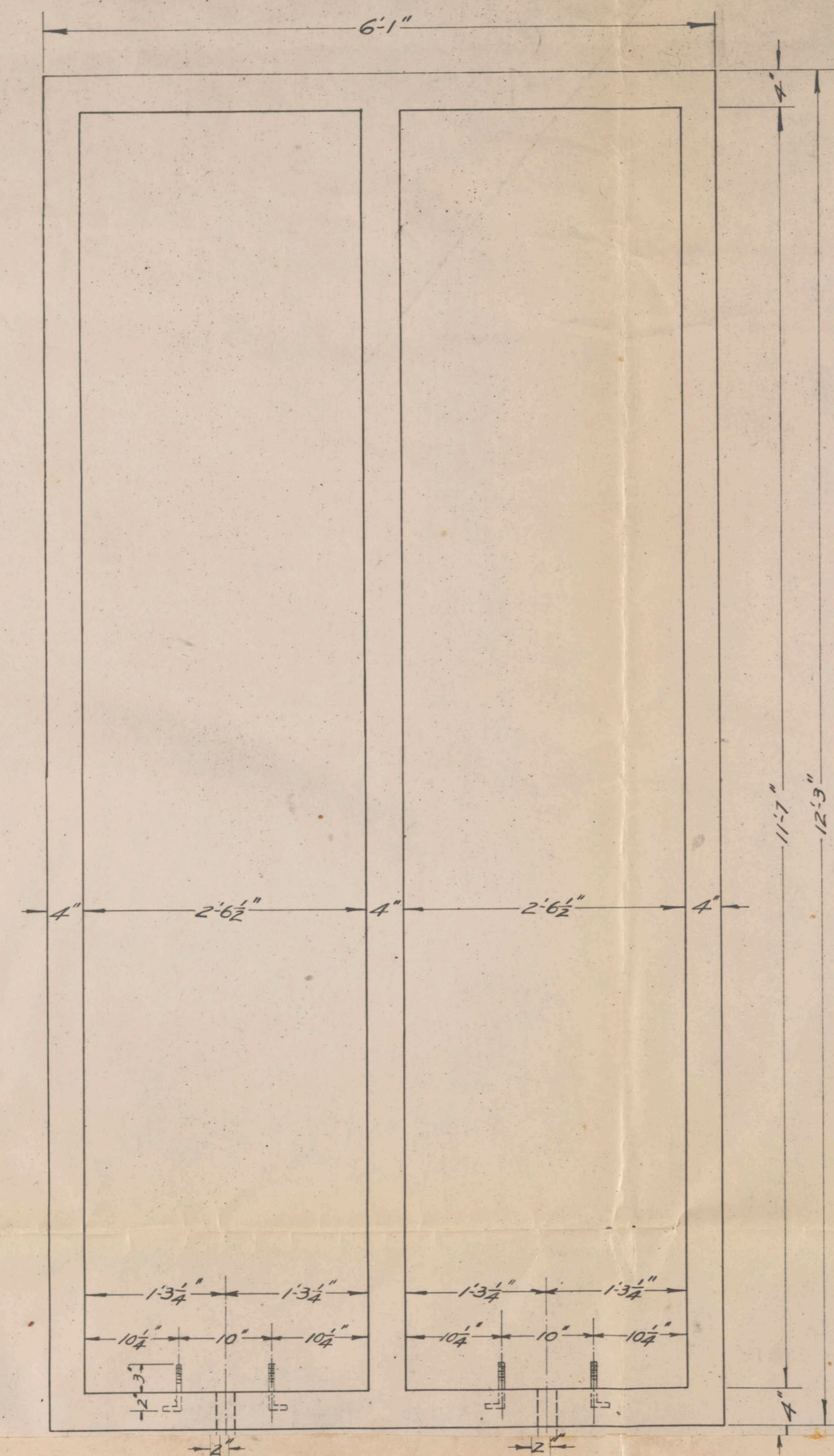
No definite costs could be secured at any of these refineries but I got the impression that the U.S. plant could operate at lower figures. That too would depend upon what Omaha Parkes Dept. charges their Betts plant for desilverizing as compared to U.S. costs for slime treatment.

The cost of electrolysis at both plants should be more or less the same considering grade of bullion and bismuth contents.

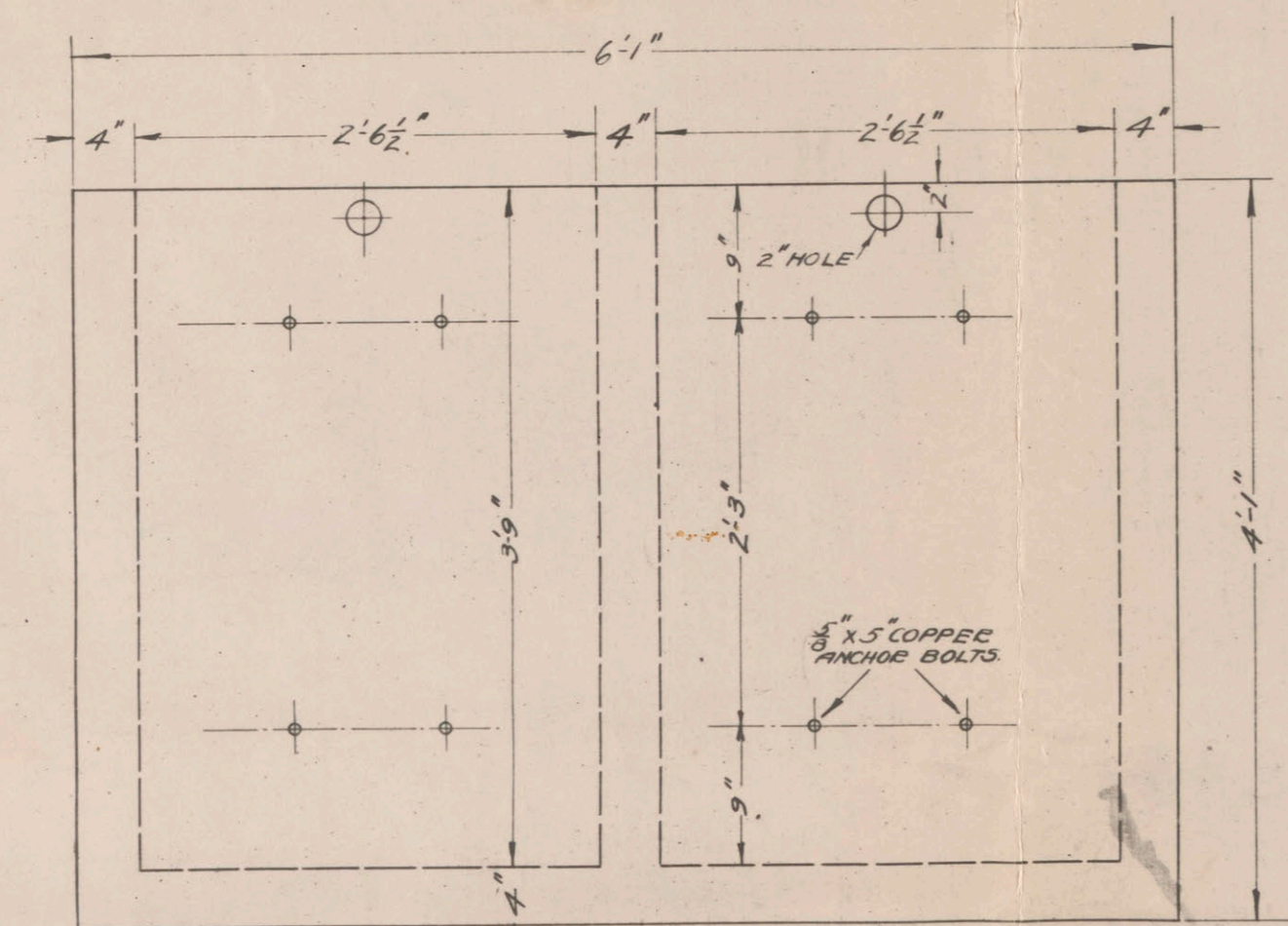
I am indebted to the following men for courtesy of plants:

	<u>U.S.S.</u> <u>East Chicago</u>	<u>Anaconda</u> <u>East Chicago</u>	<u>Omaha</u> <u>_____</u>
Supt.	J.J.Mulligan	G.E.Johnson	J.O.Betterton
Asst.Supt.	E.W.Merrick	R.G.Bowman	
Chief Eng.	O.Langley		
Research Eng.		F.H.Donahue	

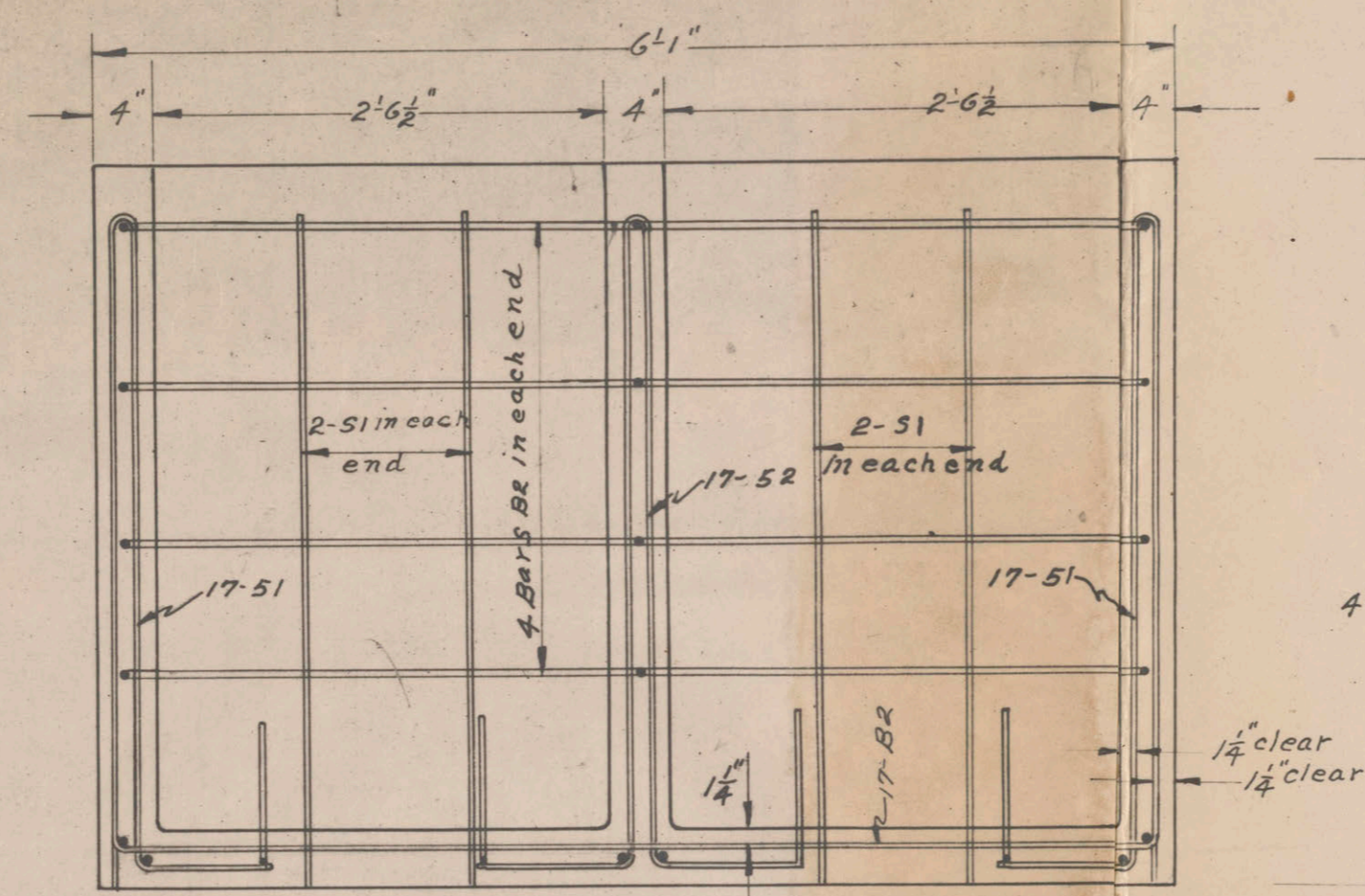
C.V.DREW



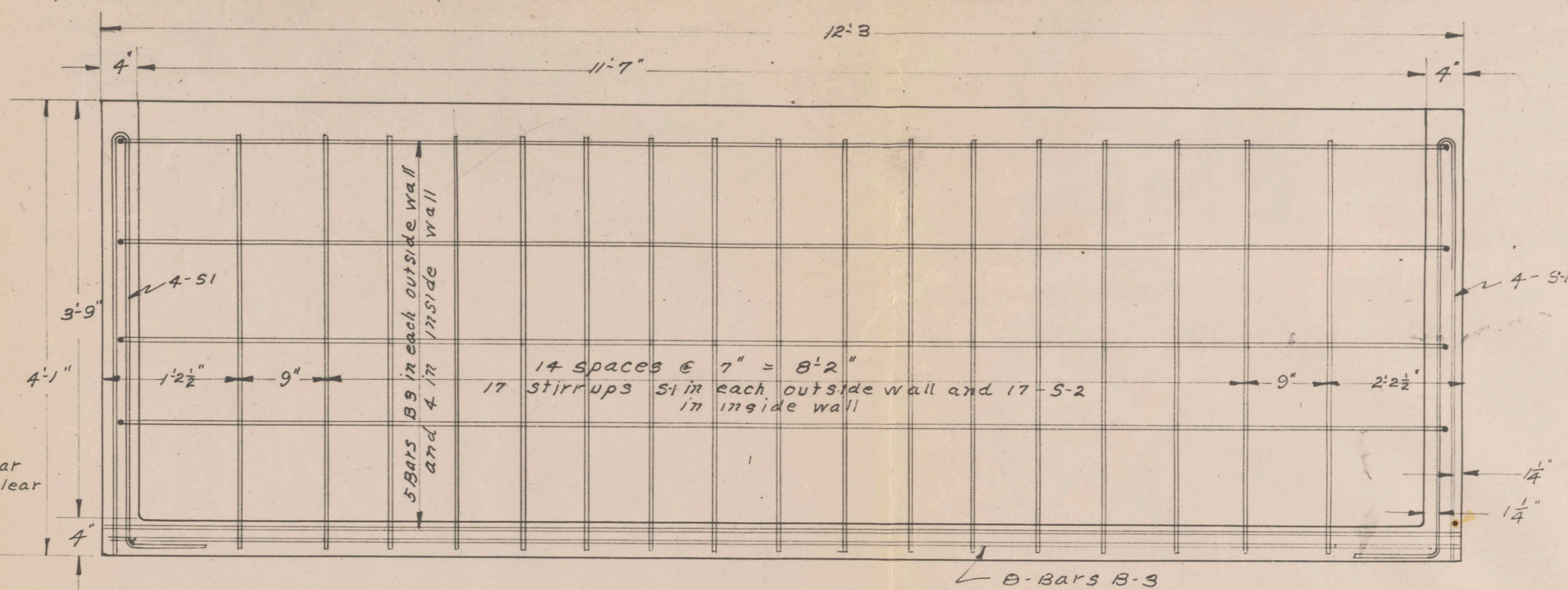
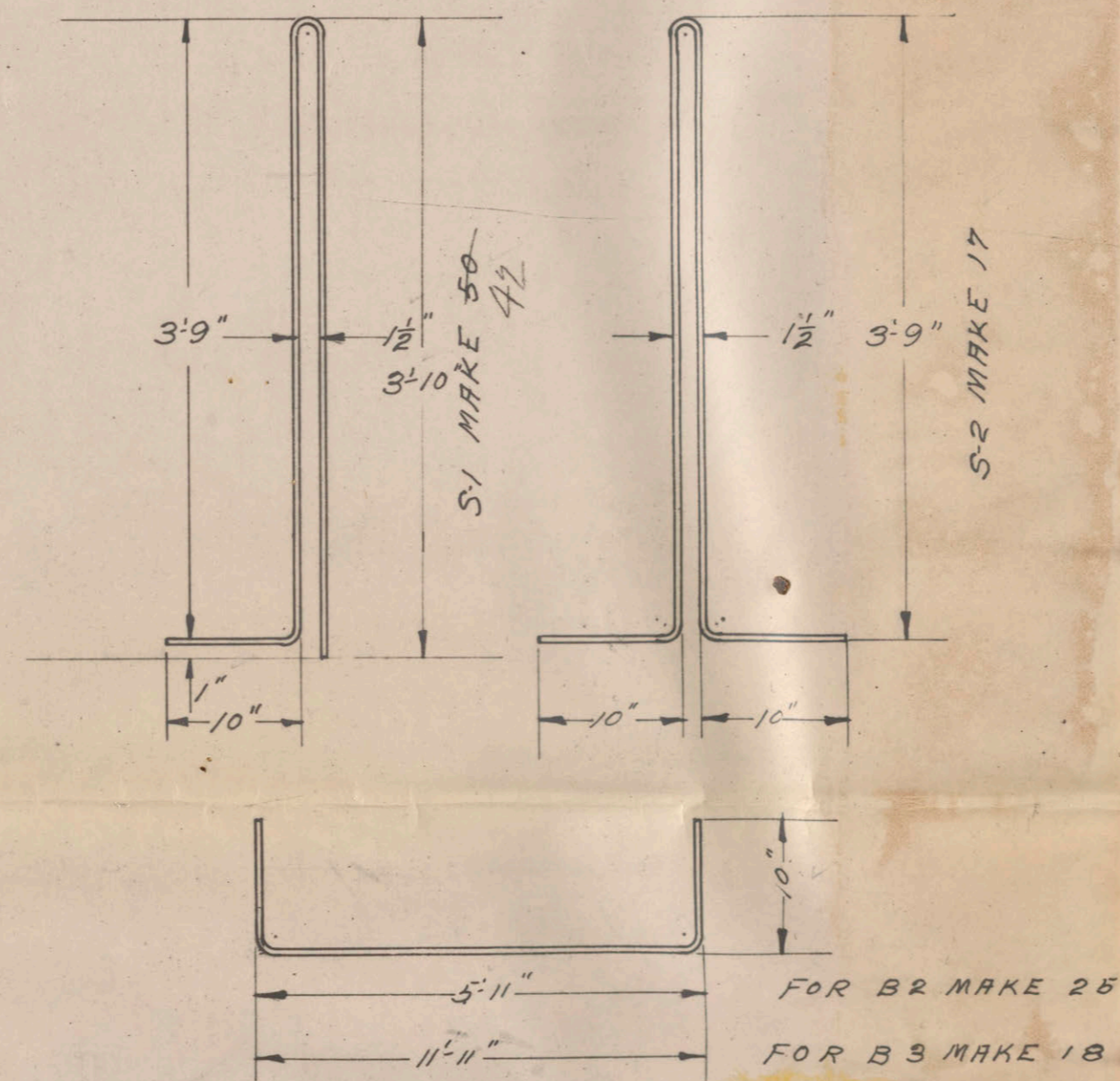
-PLAN-



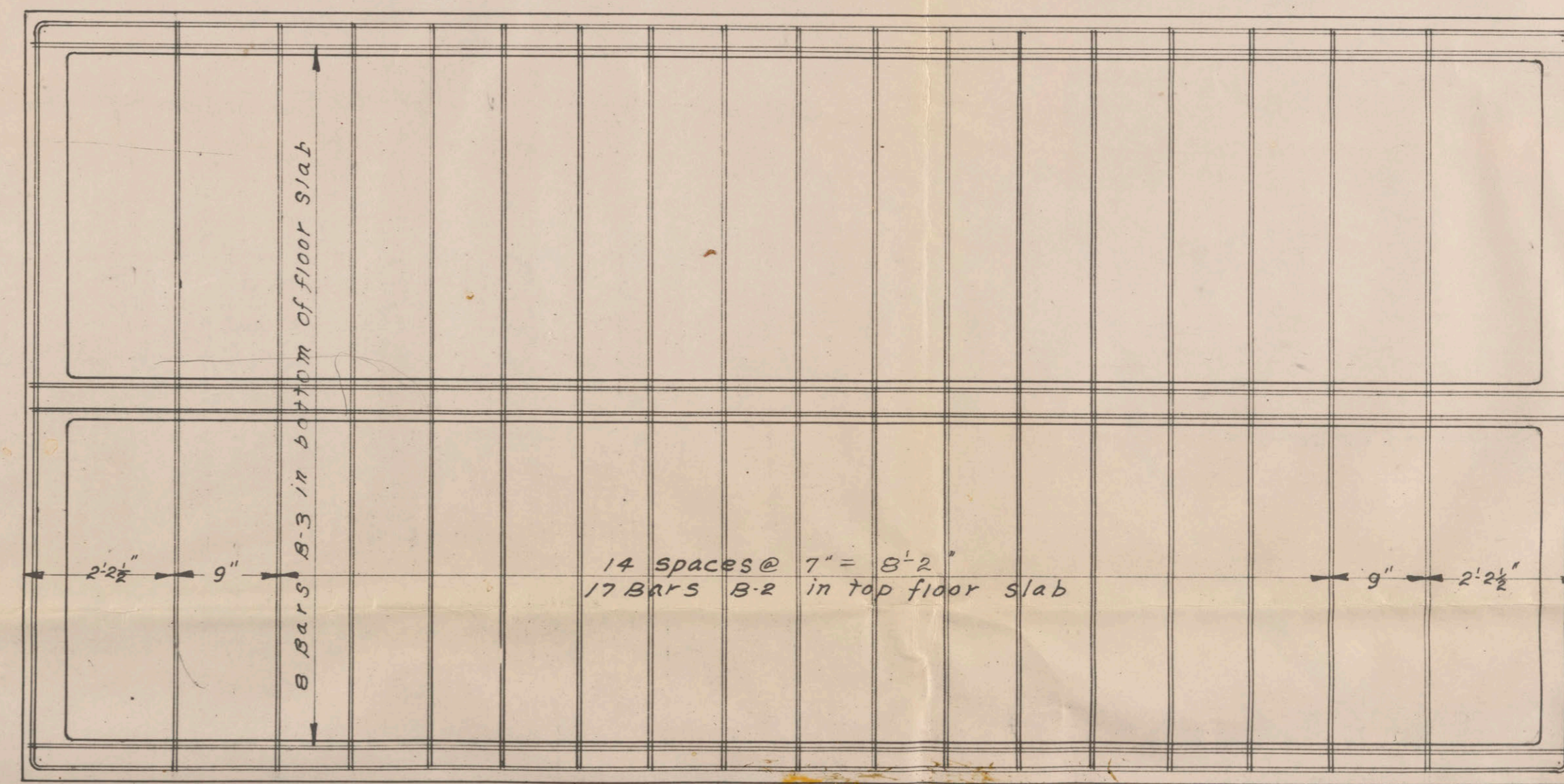
-END VIEW-



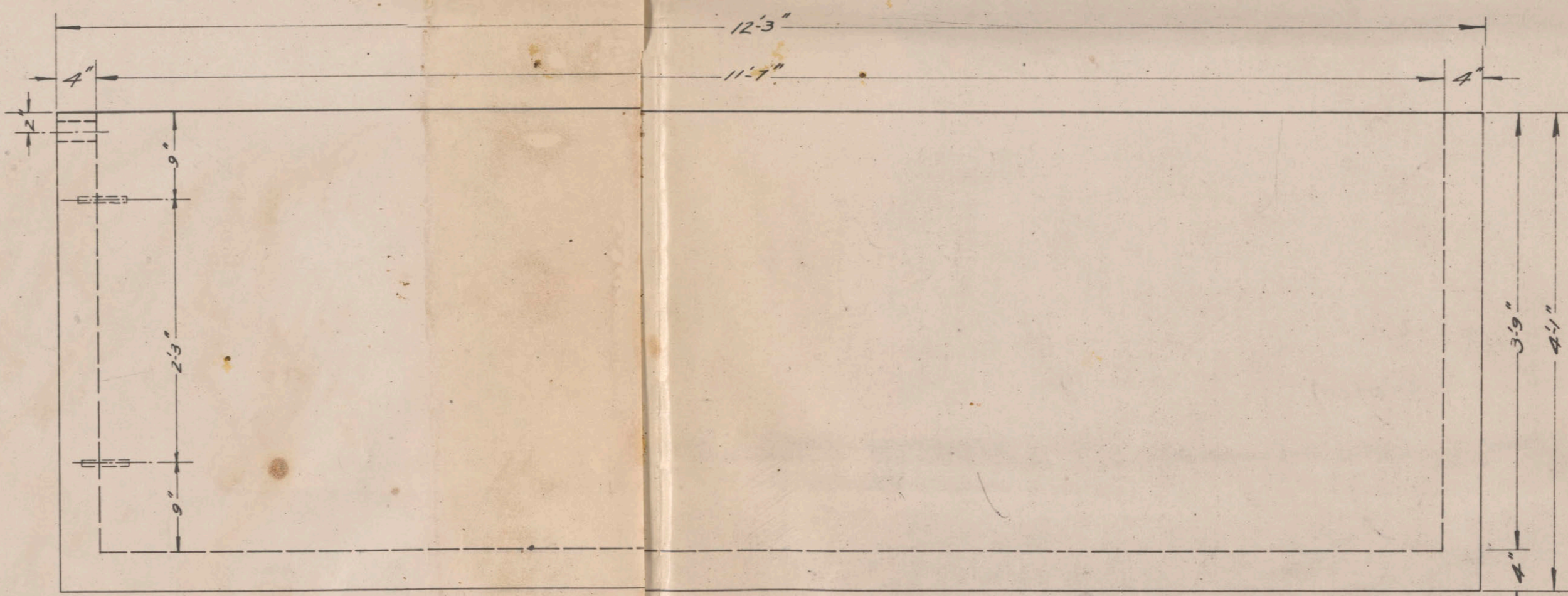
-SECTION-



-SECTION-



-PLAN-



-SIDE VIEW-

NOTE: THIS TO BE CONSTRUCTED OF RE-
 INFORCED CONCRETE.
 REINFORCING FURNISHED BY CEMENT GUN
 CONSTRUCTION CO. ANCHOR BOLTS FURN-
 ISHED BY U.S.S.L. INC.
 REINFORCING DESIGNED BY C.G. CONST. CO.
 SUBJECT TO APPROVAL OF U.S.S.L. INC.

REVISIONS	
DATE	REMARKS

U. S. S. LEAD REFINERY, INC.
 EAST CHICAGO, INDIANA
TANK HOUSE
 CONCRETE ELECTROLYTIC SECTIONS

DRAWN *E. Engelmann* DATE 10-12-25
 TRACED *E.*
 APPROVED *Chapman* ENGINEER. SCALE 1" = 1'00". NO. 780.

A-153.

51 Maiden Lane.

New York, N. Y., Oct. 4, 1927.

Mr. C. V. Drew, V. P.,
Cerro de Pasco Copper Corp.,
44 Wall St., New York, N. Y.

Dear Mr. Drew:

Betts Process Lead Refining:

I give below, in some detail, such data as I have at hand bearing upon the probable cost of treating your lead bullion by the Betts process.

There are only three plants to be considered - Grasselli, Omaha and Trail. Below are assays from several sources as to the character of bullion handled at various periods in these plants:

	Cerro.		Grasselli.			Trail.	
	With Flue Dust.	Without Flue Dust.					
Pb. %.	97.5	98	94.4	97.3	97.8	98.5	98.9
Ag.oz.p.t.	125-150	200	75	167	119	100	120
Au.oz.p.t.	--	--	1.4	2.0	1.2	0.5	0.4
Bi.	1.00-1.25	0.10-0.15	0.13	0.37	0.22	0.01	--
As.	0.5	0.5	0.006	0.68	0.14	0.28	0.35
Sb.	0.5	0.5	0.33	1.25	0.76	0.32	0.28
Cu.	0.2	0.2	0.12	0.07	0.04	0.22	0.05

I have no figures for Omaha, as that plant has been operated not on a work lead, but on the high bismuth material resulting from Pattisonizing. I understand the bismuth has at times reached 3% and that it averages about 1.5%. The Trail plant

Mr. C. V. Drew, - - - Sheet #2. - - A-153. - - Oct. 4, 1927.

represents the other extreme - electrolyzing a bullion with insufficient bismuth to yield a byproduct.

The principal point about your own lead is that it is uncomfortably high in arsenic. There is no metallurgical difficulty in handling this, but it means the production of a certain amount of arsenical hard lead as a by-product which has at times a limited market. Grasselli has produced such a product running 85% lead, 13% antimony and 2% arsenic, sold for coffin trimmings. East Chicago (Anaconda) is now shipping arsenical ^{hard} lead for making into sheets for lining tanks at the Potrevillos leaching plant. As your flue dust is very high in arsenic, I take it that your estimated assays are subject to some correction if flue dust is omitted.

Your bullion, therefore, is suitable and with minor corrections for impurities will be treated at costs comparable with those obtained elsewhere.

Sources.

I have been in intermittent contact with the Betts process for over twenty years. While some of the figures given below reached me while in the employ of others, so many years have gone by that I believe the figures may be disclosed without harm to any one.

1906.

I was responsible for the staff at Chrome conducting the research work, checking up the results obtained at the first

Mr. C. V. Drew, - - - Sheet #3 - - A-153. - - Oct. 4, 1927.

Trail plant. We built a section of a Betts plant and treated about 150 tons of U. S. Mining Company lead before the Grasselli plant was built.

1907.

In 1909 I was afforded an opportunity to study a careful comparison of costs between Parkes and Betts, made by Ernst F. Eurich in 1907 for 100 tons a day of a bullion running 98.29% lead, 60 ozs. per ton silver, 0.01% bismuth, 0.50% arsenic, 0.50% antimony, and 0.50% copper. His conclusions were \$4.52 per ton Parkes and \$5.75 per ton Betts, including 10% depreciation and 5% interest on metal tie-up. The investment was \$175,000 for Parkes and \$304,000 for Betts. The difference in favor of Parkes was reduced to \$0.30 after he allowed for metal recoveries.

1909.

In 1909 Arthur L. Walker made a similar comparison for 80 tons a day of Torreon bullion for which we were considering building a lead plant at Chrome. The analysis was about 233 ozs. per ton silver, 2 ozs. per ton gold, 0.018% bismuth, 0.75% antimony. He figured \$4.50 to \$5.00 Parkes and \$7.00 Betts and a Parkes investment of \$120,000 to \$150,000, built as an addition to the existing plant already possessing land, offices, shops and silver refinery. He did not estimate on the Betts investment.

I studied the question as a basis for plant recommendations, using the actual costs being obtained at Grasselli at the

Mr. G. V. Drew, - - Sheet #4. - - A-153. - - Oct. 4, 1927.

time, which were \$7.00 operating and \$9.00 including metal loss and depreciation on 80 tons a day of lead running 175 ozs. per ton silver. The acid loss was running about 30 lbs. per ton in those days, and the cause of much anxiety, but was later reduced to about 7 lbs., as I remember.

1910 - 1913.

I was quite active on the committee of the A. S. T. M. which was drawing up specifications for fine lead and have very complete analyses of lead in my files. The great difficulty in Betts lead is to keep the copper down to the best Parkes figure.

1917.

From 1917 to 1920 I was consulting engineer to the Burma Corporation, which owns one of the great lead-silver-zinc mines of the world, and while studying their problems I visited most of the lead plants of the country, including Grasselli and Trail. The former plant at that time stated that their costs had been "reduced."

1918.

I was talking the other day to a man who was in a responsible position at Trail ten years ago. He said that during the war things had got into very bad shape and the lead refinery costs were around \$25 a ton with a production of 30 tons a day, but that in 1918 the situation was attacked by putting labor as far as possible on a contract basis and the costs were driven all the way down to \$11 a ton with an acid loss of only three

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pounds per ton.

In 1918 Hofman published his revised "Metallurgy of Lead." On page 588 he says:

"At present this cost (Parkes process), including the reworking of intermediary products, ranges from \$3.25 to \$5.00 per ton, depending upon the size of plant and the facilities for handling material. If overhead charges, such as interest, salaries, marketing, etc., are taken into account, the above figures will have to be doubled."

Averaging and doubling we obtain \$8.25 as the Parkes plant cost.

On page 636 he says: "The cost of electrolysis is 50 percent more than that of zinc desilverization." This would make Betts \$12.38 per ton.

1921.

In 1921 I was asked some questions regarding the Betts process by a foreign client and in connection with that review I was given a glimpse at the cost sheets of the Omaha plant by one of the A. S. & R. officials. The only notes I have retained of that interview are to the effect that the cost at that time was \$11.50 a ton and that the plant was not used for bullion running less than 0.35% bismuth. As to the tonnage, I do not think the plant has been changed in size for a long time; Hofman's "Metallurgy of Lead" states it treated 18,000 tons in 1917, which would be an average of 50 tons a day. Mr. Blayloch told me the other day that the Omaha acid loss is 17 lbs.

1927.

This brings us to the present investigation. We have first the Colcord estimates of \$1,000,000 investment and an

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overall cost of \$23.57 for 50 tons a day of bullion running 1.25% bismuth.

Next, in a chat with W. C. Smith, he stated such costs did not correspond with his experience at Grasselli where he held a responsible position for many years, and further he knew as a later employe of the A. S. & R. that Grasselli had bid lower terms than this in trying to get their bullion to treat.

Then we have the statements of S. G. Blayloch, the general manager at Trail, that a 100 ton plant could certainly do \$12 and indirectly that the Trail 400 ton-a-day plant was not over \$1.50 above present Parkes costs (or about \$8.75).

I summarize this scattered information as follows:

<u>Plant.</u>	<u>Date.</u>	<u>\$ p.t.</u>	<u>Tons</u> <u>p.day</u>	<u>Acid Loss</u> <u>lbs.p.ton.</u>	<u>Ozs.</u> <u>Ag.</u>	<u>%</u> <u>Bi.</u>	<u>\$</u> <u>Investment.</u>
"Eurich"	1907	5.75	100	--	60	0.01	304,000
"Walker"	1909	7.00	80	--	253	0.02	---
Grasselli	1909	9.00	80	30 (a)	175	0.21	500,000
Trail	1918	11.00	30	3	100	0.01	---
"Hofman"	1918	12.38	--	--	---	---	---
Omaha	1921	11.50	50	17	(100?)	1.50	---
Trail	1927	8.75	400	3.5	100	0.01	---
"Colcord"	1927	23.57	50	9.4	200	1.25	1,000,000

(a) Later reduced to 7 lbs.

In addition we have the following comparisons between Parkes and Betts:

		<u>\$ p. ton</u> <u>Parkes.</u>	<u>\$ per ton</u> <u>Betts.</u>	<u>Ratio.</u>
Eurich	1907	4.52	5.75	1.27
Walker	1909	4.75	7.00	1.47
Hofman	1918	8.25	12.38	1.50
Blayloch	1927	7.00	8.75	1.25

At first glance these figures seem to indicate that the Colcord estimate is surprisingly high. We have to consider, however, location, time, composition of bullion and size of plant before making comparisons. First of all it will be well to examine the Colcord report to see just how it has been built up.

Analysis of the Colcord Report.

I have at hand the detailed cost sheets of the Grasselli plant (now known as East Chicago) for 9 months of 1909 and the structure is identical with that used by Mr. Colcord, showing that he has taken the cost sheets of this plant and made such modifications as he considered necessary for your capacity and composition. The American Bureau of Metal Statistics for 1926 gives his capacity as 40,000 tons per annum, which is 109 tons per day. He has undoubtedly taken his present cost sheet and seen what he could eliminate in cutting down capacity. The tendency here would be to figure high.

We have no reason to believe Grasselli is treating bullion with any considerable amount of bismuth. The "maximum" in Liddell's "Hand Book of Non-Ferrous Metallurgy," given by Smith in his article, is 0.37%. The Mineral Industry for 1926 states the total output of bismuth to be 300,000 lbs., coming wholly from Omaha and Grasselli. Taking a total lead output of 60,000 tons, the average bismuth recovered would be 0.25%, or say 0.3% in the bullion, and we know Omaha treats high bismuth

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material. We can safely say, therefore, that Mr. Colcord scaled up his figures to your 1.25% bismuth from cost sheets for about 0.2% bismuth.

In the latter connection I note his figures on 40% anode scrap and a cathode production of 14 lbs. a KWH, while Mr. Smith in his article gives as an average for fifteen months at Grasselli 19.65% scrap and 18.1 lbs. per KWH. These differences would account for \$0.23 and \$0.35 a ton respectively, presumably chargeable against impurities.

Further we can parallel the estimate (omitting lighter handling) with the 1909 Grasselli costs with the following results:

	1927 Colcord.	1909 Grasselli.	Ratio.
Handling and Casting,	3.16	1.31	2.4
Electrolyzing,	6.23	2.83	2.2
Slimes Treatment,	5.23	1.47	3.6
Operating,	14.62	5.61	2.6
General Expense,	4.78	1.39	3.4
Metal Loss,	0.87	0.99	0.9
Depreciation,	2.78	0.85	3.2
Total,	23.05	8.84	2.6
Tons per day,	50	80	0.6

It is noticeable at once that the silver building has been heavily increased and if we reduced the impurities to the regular Grasselli grade we could deduct about \$1.80 a ton. Examining General Expense for its heavy increase, the item of \$1.42 for taxes seems excessive, but we can let that pass for the moment as well as the depreciation. We have, therefore, \$0.23 + \$0.35 + \$1.80 = \$2.38 to deduct for bismuth and can take \$23.05 - \$2.38 = \$20.67 as a comparable basis with Trail.

Looking now at the question of size of plant, we can take the "estimate", Trail 1927 and each of the nine months of 1909 for which I have Grasselli figures. I believe if we multiply the latter by 1.75, to allow for the change in prices, we shall be near enough for our purposes. While common labor is now three times as high, commodities are only 1.5 and there has been a great increase in the productivity of labor as well as the introduction of much labor saving machinery.

In this way we arrive at the following table:

<u>Plants.</u>	<u>Tons per day.</u>	<u>1909.</u>	<u>1927.</u>
Colcord Estimate,	50	--	20.67
Grasselli,	67.8	10.63	18.70
"	77.0	9.28	16.20
"	78.5	9.06	15.80
"	80.1	9.04	15.80
"	81.7	9.31	16.30
"	83.5	9.36	16.30
"	83.5	9.42	16.40
"	86.7	8.49	14.80
"	88.3	9.20	16.10
Trail,	400	--	8.75

These figures are plotted in the accompanying diagram and the results are quite striking. I have included a companion line for copper from a report I worked up last year. The processes are very similar, but copper costs twice as much, due to its much higher melting point and lower electrochemical equivalent.

Of course you will ask how I explain the low 1918 figures for Omaha and Trail. In the first place I suspect they do not include depreciation and metal loss, and in the second, in both

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cases a small Betts plant was on the ground of a large metallurgical aggregate and the general expenses were as low per ton as if the Betts plant had been large.

Conclusions.

I believe Mr. Colcord has built upon a present Grasselli cost of around \$13, adding for impurities and taking a much smaller tonnage divisor. At any rate it is evident that for economical work a plant should treat at least 200 tons a day if it is to stand on its own feet. No doubt a much smaller one will suffice if built on to an existing plant.

Similarly in investment, the original Grasselli plant stood at \$500,000 for 100 tons a day, or \$5,000 per daily ton. Mr. Colcord estimates 1,000,000 for 50 tons, or \$20,000 per daily ton. This seems too much.. Probably he would get 100 tons a day by adding \$250,000, making it \$12,500 per daily ton, or 2.25 times as much as building in 1907. This seems possible.

Yours very truly,

L. A.
B.

Adx/B.

LEAD BULLION.

	Au.	Ag.	Cu.	Pb.	Fe.	As.	Sb.	Bi.	Se+Te	S.
U. S. Mining	1.37	75.0	.12	99.35	.0079	.0060	.3342	.134	No	Tr.
Luna Lead	.03	32.3	.18	99.04		.193	.375	.061		

DE LAMAR'S COPPER REFINING COMPANY

Chrome, N.J., June 11, 1906.

Synopsis of Lead Experiments.

	Ex.No.1 250% Ano.	Ex.No.2 1st Crop 400% Ano	Ex.No.2 2nd Crop	Ex.No.3 1st Crop 450% Ano.	Ex.No.3 2nd Crop 450% Ano.	Ex.No.4 1st Crop 450% Ano.	Ex.No.4 2nd Crop 450% Ano.	Ex.No.5 1st Crop 450% Ano.	Ex.No.5 2nd Crop 450% Ano.	Ex.No.5 3rd Crop	Ex.No.6 1st Crop 425% Ano.	Ex.No.6 2nd Crop 425% Ano.	Ex.No.7 1st Crop 425% Ano.	Ex.No.7 2nd Crop 425% Anode
Wght. good anodes charged	26,484	42,541		46,235		46,776		43,680			38,720		44,901	
Weight of Scrap	10,129		8,679		12,764		13,870			8,429		10,892		
" " Cathodes	17,222	19,221	14,594	15,498	17,622	15,671	19,077	16,176	16,522	5,426	16,804	15,690	10,103	15,790
" " Starting Sheets	1,702	1,623	1,416	1,568	1,650	1,768	1,560	1,530	1,508	1,456	1,710	1,456	1,624	1,592
" Cathodes Deposited	15,520	17,598	12,978	13,930	15,972	13,903	17,517	14,646	15,014	4,970	15,094	12,234	8,579	14,198
Aver. days in circuit	5.09	6.20	4.58	5.41	6.37	5.00	6.25	4.87	5.04	1.41	4.87	4.20	2.91	4.78
Average Amperes	3,989	3,986	4,079	3,962	4,012	3,921	4,039	4,058	4,424	4,223	4,058	4,455	4,068	4,223
" Volts	1.70	1.42	1.32	1.14	1.22	1.10	1.07	1.23	1.16	1.90	1.27	1.10	1.16	1.05
" Temperature °F	86	86	92	80	81	82	86	82	82	82	82	84	79	86
Gallons per Minute	4	4	4½	4½	6	6.2	7	5	6	6	5	6	5	5
Efficiency	94	87.4	85.2	79.8	76.7	85.7	85.1	90.9	82.4	88	93.7	84.3	88.6	86.5
Lbs. per Kw. Hour	18.8	20.8	20.9	23.7	21.3	26.4	27.0	25.1	24.1	25.4	25.6	24.7	25.8	28.4
% Ano. Scrap (Anode)	38.2	-	20.4	-	40.5	-	33	-	-	19	-	28	-	-
% " " (Cathode)	86.5	-	28.3	-	64.	-	29	-	-	24	-	39	-	-
Size Anode under Solution	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'	3' x 2'
" Cathode " "	3'1"x2'2"	3'1"x2'2"	3'1"x2'2"	3'1"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"	3'½"x2'2"
Thickness of Anode	11/16"	1½"	-	1½"	1½"	1½"	1½"	1½"	-	-	1 3/32"	-	1 3/32"	-
Highest Volts Recorded	3.6	2.6	1.6	1.4	1.6	1.6	1.4	1.45	1.40	2.4	1.46	1.60	1.4	1.2
Lowest " "	1.2	1.2	1.2	.8	1.1	1.2	.9	1.10	1.10	1.2	1.07	1.00	1.0	.8
Highest Temperature	94	94	92	82	82	82	82	82	82	90	88	88	86	92
Lowest " "	72	72	87	70	82	82	84	75	82	80	75	81	68	78
Highest Gallons	4½	6	5	6	7	7	8	6	7	7	6	6	6	7
Lowest " "	2½	2	3½	3	4	4	6	4	4	5	4	4	4	3
Acid at start	13.20	13.20	12.67	12.04	11.87	11.87	12.10	15.01	14.09	12.98	15.01	14.09	12.98	-
" " finish	12.67	12.67	12.04	11.87	11.50	11.50	15.10	15.19	12.98	-	15.19	12.98	12.00	12.26
Density Cathode base	11.6	11.6	11.8	11.5	11.8	11.7	11.2	13.4	11.90	12.6	11.4	13.00	10.00	11.20
" Anode "	12.7	12.7	13.6	12.6	12.8	12.7	12.9	15.3	13.10	14.2	13.0	14.8	12.4	12.7
Silver in cathodes	.37	.15	.44	.47	.75	.40	.65	.42	1.01	.47	.52	.70	-	-
Gold " "	.002	Tr	.01	.01	.0027	.003	.01	Tr	.01	.006	.004	.005	-	-
Volts per tank	.42	.34	.34	.28	.30	.27	.26	.307	.29	.47	.317	.217	-	-
Lead contents at art	5.04	5.04	5.28	4.97	5.22	5.22	5.79	5.62	5.86	6.19	5.62	5.86	6.19	6.25
" " finish	5.22	5.22	4.97	5.22	5.20	5.20	6.31	4.13	6.19	-	4.13	6.19	7.03	8.75
Aver. Weight Cathodes	159	177	151	143	163	145	168	161	126	41	147	117	-	-
No. cathodes per tank	-	-	-	-	-	-	-	23	29	26	27	26	29	29
" anodes " "	-	-	-	-	-	-	-	24	28	25	26	25	28	28
Deposit	-	-	-	-	-	-	-	Soft and porous	Fair; Slightly Modulated	Tough	Fairly Tough Modulated	-	Fair Modulated	Tough Smooth

145.

Grasselli Plant.	B O I L E R R O O M		C O S T, 1 9 0 9.			
	August. ¢ p.1000#	September. ¢ p. 1000#	October. ¢ p.1000#	November. ¢ p.1000#	December. ¢ p.1000#	Yearly. ¢ p.1000#
Labor,	3.577	3.939				
Supplies,	0.112	0.011				
Repairs of B. & E.,	0.112	0.390				
Building Repairs,	0.038	—				
Unloading Coal,	0.387	0.577				
Water,	0.864	0.795				
Fuel,	15.172	13.040				
Total,	16.292	18.751				
Steam Produced,	15,003,960	12,209,491				
Less feed pumps &c.,	450,000	450,000				
Cost basis,	12,553,960	11,759,491				
¢ to Engines,	85.6	84.3				

BOILER ROOM COST, 1909.

(A)

Grasselli Plant.	January. ¢ p.1000#	February. ¢ p.1000#	March. ¢ p.1000#	April. ¢ p.1000#	May. ¢ p.1000#	June. ¢ p.1000#	July. ¢ p.1000#
Labor,	3.09	3.40	3.54	3.91	3.55	3.343	3.174
Supplies,	0.09	0.06	0.06	0.03	0	0.147	0.013
Repairs of B. & E.,	0.30	0.11	0.12	2.61	0.21	0.123	0.424
Building Repairs,	0.01	—	0.01	0.12	0.01	—	—
Unloading Coal,	0.43	0.40	0.27	0.64	0.29	0.465	0.718
Water,	0.47	0.43	0.46	0.52	0.36	0.691	0.726
Fuel,	14.86	14.62	15.32	10.81	15.44	14.438	14.729
Total,	19.25	19.02	19.78	19.64	19.66	19.207	19.764
<hr/>							
Steam Produced,	12637421	10539187	11584251	11612772	12775386	12032393	12857370
Less feed pumps, &c.	398414	350000	400000	400000	450000	450000	450000
Cost Basis,	12239007	10189187	11184251	11212772	12325386	11582393	12407370
<hr/>							
% to Engines,	62.7	74.3	73.4	76.5	79.5	79.1	84.6
<hr/>							

BOILER ROOM COST, 1909.

(A)

Crasselli Plant.	January. \$ p.1000#	February. \$ p.1000#	March. \$ p.1000#	April. \$ p.1000#	May. \$ p.1000#	June. \$ p.1000#	July. \$ p.1000#
Labor,	3.09	3.40	3.54	3.91	3.35	3.343	3.174
Supplies,	0.09	0.06	0.06	0.03	0	0.147	0.013
Repairs of B. & D.,	0.30	0.11	0.12	2.61	0.21	0.123	0.424
Building Repairs,	0.01	—	0.01	0.12	0.01	—	—
Unloading Coal,	0.43	0.40	0.27	0.64	0.29	0.465	0.718
Water,	0.47	0.43	0.46	0.52	0.36	0.691	0.726
Fuel,	14.86	14.62	15.32	10.61	15.44	14.438	14.729
Total,	19.25	19.02	19.78	18.64	19.66	19.207	19.784
Steam Produced,	12637421	10539187	11584251	11612772	12775386	12032393	12657370
Less feed pumps, &c.	398414	350000	400000	400000	450000	450000	450000
Cost Basis,	12239007	10189187	11184251	11212772	12525386	11582393	12407370
% to Engines,	62.7	74.3	73.4	76.5	79.5	79.1	84.6

(A)

Grasselli Plant.	BOILER ROOM COST, 1909.					
	August.	September.	October.	November.	December.	Yearly.
	¢ p.1000#	¢ p. 1000#	¢ p.1000#	¢ p.1000#	¢ p.1000#	¢ p.1000#
Labor,	3.577	3.938				
Supplies,	0.112	0.011				
Repairs of B. & E.,	0.112	0.390				
Building Repairs,	0.036	—				
Unloading Coal,	0.387	0.577				
Water,	0.864	0.795				
Fuel,	13.172	13.040				
Total,	18.232	18.751				
Steam Produced,	13,003,960	12,209,491				
Less feed pumps &c.,	450,000	450,000				
Cost basis,	12,553,960	11,759,491				
% to Engines,	85.6	84.3				

Grasselli Plant. \$ per ton	ANODE CASTING DEPARTMENT COST 1909.						
	January.	February.	March	April.	May.	June.	July.
Labor	.25	.22	.20	.21	.20	.186	.177
Supplies			.01			.002	.003
Repairs of M.&E.	.04	.03	.04	.05	.02	.010	.016
Building Repairs.			.01			.004	.007
Fuel	.04	.05	.04	.04	.05	.050	.049
Kettle Repairs.							
Experimental.		.04					.045
Steam, Power & Light.	.07	.03	.06	.04	.05	.043	.042
By-Prod. Treatment		.40	.09		.12	.063	.075
	.38	.77	.45	.34	.42	.363	.413
Tons Anodes Cast,	2054	2376	2239	2521	2510	2767	3081
Tons Dross,	80	124	133	129	102	98	88
Ratio Dross to Anodes	2.57		5.94	5.12	4.06	3.66	2.86
Ratio Scrap to Anodes Charged.							39.6
Tons Anode Scrap Treated	1036	1089	1266	1297	1330	1119	1220
Charge to Tank Room	354.31	832.59	564.65	449.97	572.72	335.122	410.53
Charge to Anode Cast.	431.00	993.99	433.97	425.45	506.13	669.266	859.57
	<u>785.31</u>	<u>1816.58</u>	<u>998.62</u>	<u>875.40</u>	<u>1080.85</u>	<u>1004.38</u>	<u>1270.10</u>

Grasselli Plant. \$ per ton.	August.	September.	October.	November.	December.	(A-1)
Labor.	.211	.210				
Supplies,	.003					
Repairs of M. & E.	.052	.023				
Building Repairs,	.004	.019				
Fuel,	.043	.052				
Kettle Repairs,						
Experimental						
Steam, Power & Light,	.056	.049				
By-Prod. Treatment,	<u>.099</u>	<u>.099</u>				
	.467	.451				
Tons Anod. Cast.	1990	2396				
Tons Dress,	569	44.6				
Ratio Dress to Anod.	1.85	1.86				
Ratio Scrap to Anod.	33.3	30.4				
Tons Anod. Scrap Treated	1369	1244				
Charge to Tank Room	652.49	560.44				
Charge to Anode Casting	<u>277.64</u>	<u>518.99</u>				
	930.13	1079.43				

Grasselli Plant.

REFINED BAR KETTLE COST, 1909.

(B-1)

\$ per Ton.	January.	February.	March.	April.	May.	June	July.
Labor,	.26	.30	.22	.26	.25	.231	.266
Supplies,			.01			.004	.008
Rep. to Machy & Equip.	.04	.02	.05	.02	.02	.031	.044
Building Repairs,			.01			.004	.001
Fuel,	.05	.06	.06	.07	.05	.040	.054
Kettle Repairs,				.01		.001	
Steam, Power & Light,	.05	.03	.04	.04	.04	.04	.041
By-Prod. Treatment,				.19	.10	.119	.100
	.40	.41	.39	.59	.46	.404	.514
Tons Refined Bars,	2190	1929	2097	2191	2322	2248	2369
Tons Liguator Prod.						146	126
Tons Dross	98.5	124	258	182	146	158	185
Ratio Dross to Ref. Prod.	4.5	6.4	12.31	8.30	6.30	7.03	7.81
Ave. Sb in Ref. Elec. lead,	.0249	.0071	.0071	0.0066	0.0147	.00826	.0063
" " Common "		.076		0.097	0.0442	.03260	.023
" Ag " Ref. Elec. "		0.42	.64	0.47	0.278	.271	.28
" " Common "		0.77		0.30	0.304	.322	.31
" Au " All Lead							.0027
" Au & Ag in Ref. Prod.	0.55						
" Au in Common lead,		.0031	.0037	.0030	.0036	.0039	

Grasselli Plant.

(B-1)

\$ per ton.	August.	September.	October.	November.	December.
Labor,	.284	.268			
Supplies,	.006	.006			
Repr. to Mach & Equip.	.057	.030			
Bldg. Repairs,	.003	.019			
Fuel,	.064	.054			
Kettle Repairs,	.015	.028			
Steam, Power & Light,	.033	.041			
By-Prod. Treatment,	<u>.072</u>	<u>.025</u>			
	.554	.471			
Tons Refined Bars,	2521	2369			
Tons Lignator Prod.,	18	50			
Tons Dress,	139	115			
Ratio Dress to Ref. Prod.	7.03	4.85			
Ave. Sb. in Ref. Elec. Lead,	.0050	.00477			
" " " Common "	.0038	.01160			
" Ag " Ref. Elec. "	.23	.21			
" " " Common "	.22	.19			
" Au " All Lead,	.0027	.0027			
" Au & Ag. in Ref. Prod.					
" " in Common Lead,					

SUMMARY OF GRASSELLI REFINERY COSTS, 1909.

GRASSELLI PLANT.	January.	February.	March.	April.	May.	June.	July.
	\$ p.t.	\$ p.t.	\$ p.t.	\$ p.t.	\$ p.t.	\$ p.t.	\$ p.t.
General Items,	1.40	1.46	1.43	1.29	1.27	1.198	1.165
Anode Casting,	0.19	0.48	0.18	0.16	0.20	0.278	0.331
Furnace Room,	2.85	3.03	2.71	3.04	2.79	2.665	2.723
Refined Bar Castings	0.38	0.41	0.54	0.50	0.42	0.487	0.491
Silver Refinery,	0.59	0.60	0.52	0.59	0.70	0.705	0.474
Parting Plant,	0.77	1.15	0.79	0.75	1.02	0.907	0.756
Yard, Store & Shop,	0.61	0.70	0.64	0.57	0.59	0.677	0.455
Laboratory,	0.29	0.31	0.29	0.25	0.33	0.250	0.219
Liquator,	0.05	0.07	0.13	0.07	0.06	—	—
Total,	7.13	8.21	7.03	7.22	7.38	7.167	6.614
Metal Interest,	0.40	0.21	0.23	0.21	0.13	0.008	0.180
Depreciation,	0.82	1.07	0.88	0.84	0.82	0.856	0.791
Metal Loss,	0.91	1.14	0.92	1.04	1.09	1.006	0.906
Inventory,	0.02	—	—	—	—	—	—
Total,	9.28	10.63	9.06	9.31	9.42	9.037	8.491
Bismuth,	-0.23	+0.90	+0.92	+1.36	+1.61	+1.400	+1.267
Antimony,	-0.17	-0.20	+0.17	+0.47	+0.56	+0.152	—
Copper,	—	—	—	—	—	—	+0.447
Net Total,	9.68	9.93	8.31	7.48	7.25	7.485	6.777

SUMMARY OF GRASSELLI REFINERY COSTS, 1909.

Grasselli Plant.	August. \$ p.t.	September. \$ p.t.	October. \$ p.t.	November. \$ p.t.	December. \$ p.t.	Yearly. \$ p.t.
General Items,	1.270	1.432				
Anode Casting,	0.104	0.207				
Tank Room,	2.732	2.948				
Refined Bar Casting,	0.546	0.453				
Silver Refinery,	0.563	0.626				
Parting Plant,	0.897	0.811				
Vars, Store & Shop,	0.468	0.513				
Laboratory,	0.228	0.289				
Liquator,	--	--				
Total,	6.808	7.279				
Metal Interest,	0.241	0.128				
Depreciation,	0.777	0.823				
Metal Loss,	0.992	0.926				
Inventory,	0.377	0.200				
Total,	9.195	9.356				
Bismuth,	+ 2.158	+ 3.133				
Antimony,	+ 0.356	+ 0.039				
Copper,	+ 0.210	+ 0.367				
Net Total,	6.471	5.817				

Grasselli Plant. \$ per ton.	YARD, STORE & SHIP COSTS, 1909.						
	January.	February.	March.	April.	May.	June.	July.
Watching,	.09	.09	.09	.09	.08	.079	.077
Track Repairs,							
Fire System	.01					.001	.001
Electric Lights,	.01	.01	.01	.01	.01	.009	.007
Sewer, Drains, Roads, Fences,	.03	.02	.02	.02	.04	.044	.025
Store Expenses,	.10	.12	.09	.08	.03	.076	.073
Shop Expenses,	.06	.06	.05	.05	.06	.084	.056
Bldg. Repairs,		.04				.001	
Sampling,	.04	.05	.02	.02	.01	.021	.011
Hdlg. Prod. from Cast. Bldg.					.07	.063	.055
" " " stock pile,					.06	.057	.052
" Anodes Received,					.13	.133	.093
Cost Hdlg. bullion rec'd in) shape of pigs or slabs from) car to stock pile or kettle)		.18	.23	.20	.02	.067	.066
Track Repairs, to Tracks,	.02	.01	.02	.02	.02	.097	.049
Handling Product,	.14	.17	.13	.14			
" Tintic Bullion,	.20				.07		
Mis. Credits,				.03			.036
	<u>.70</u>	<u>.75</u>	<u>.66</u>	<u>.60</u>	<u>.56</u>	<u>.732</u>	<u>.533</u>
Credit Storage Charge,	.09	.05	.02		.06	.055	.076
	.61	.70	.64		.60	.677	.456
Tons Cathodes	2312	2034	2334	2450	2505	2404	2601

Grasselli Plant.
\$ per ton.

YARD, STORE & SHOP COSTS, 1909.
August. September. October. November. December.

(3-1)

Watching,	.073	.078		
Track Repairs,	.009	.012		
Fire system,	.002			
Electric Lights,	.006	.017		
Sewers, Drains, Roads, Fences,	.024	.040		
Store Expense,	.081	.090		
Shop Expense,	.050	.054		
Building Repairs,				
Sampling,	.012	.017		
Hdlg. Prod. from Casting House,	.061	.063		
" " " Stock Pile,	.037	.020		
Handling Anodes Received,	.130	.116		
Cost Hdlg. Bullion rec'd in shape of pigs or slabs from car to stock pile on Kettle)	.032	.066		
Miscellaneous Credits,				
	<u>.540</u>	<u>.573</u>		
Credit Storage,	.072	.060		
	<u>.468</u>	<u>.513</u>		
Tons Cathodes,	2650	2505		

Grasselli Plant. \$ per ton.	GENERAL EXPENSES. 1909.							(P-1)
	January.	February.	March.	April.	May	June.	July.	
Management, Salaries & Exp.	.33	.38	.32	.32	.30	.262	.269	
Office Expenses.	.18	.14	.17	.15	.20	.145	.053	
Building Repairs.	.01					.004	.012	
Royalties.	.31	.34	.29	.32	.32	.290	.367	
Insurance.	.06	.11	.08	.08	.07	.094	.075	
Taxes.	.07	.06	.06	.06	.05	.060	.055	
Traffic	.03	.04	.03	.03	.03	.029	.027	
Salaries of Clerks.	.26	.31	.27	.34	.25	.254	.236	
Engineering Expenses.	.09	.01	.09			.009	.003	
Stationery & Printing.		.03	.05	.04	.04	.033	.042	
Sale of Brass.								
Legal & Advertising.			.07		.03			
Doctors Expenses.						.008 _a	.010	
Strike Expenses.						.019	.018	
Tintic Telegrams.						.005	.003	
	<hr/>	<hr/>	<hr/>	<hr/>	<hr/>	<hr/>	<hr/>	
	1.36	1.46	1.43	1.34	1.29	1.198	1.165	
Tons Cathodes.	2312	2034	2354	2450	2505	2404	2601	
" Lead.	2564	2172	2351	2553	2530	2434	1985	
Ozs. Silver.	387904	471218.08	374432	264729	505635	395319	381527	
" Gold.	3890	4474.581	3699	3790	3699	3084	3572	

a = in red.

Grasselli Plant. \$ per ton.	GENERAL EXPENSES.			
	August.	September.	October.	November. December.
Management, Salaries & Exp.	.201	.209		
Office Expense,	.062	.122		
Building Repairs,		.001		
Royalties,	.404	.404		
Insurance,	.075	.079		
Taxes,	.054	.057		
Traffic,	.026	.031		
Salaries of Clerks,	.244	.246		
Engineering Expenses,	.117	.234		
Stationery & Printing,	.082	.036		
Sale of Brass,		.062*		
Legal & Advertising,				
Doctors Expense,		.001		
Strike Expenses,	.003			
Tintic Telegrams.				
	<u>1.270</u>	<u>1.432</u>		
Tone Cathodes,	2650	2505		
" Lead,	2668	2421		
Oss. Silver,	478905	394545		
" Gold,	4823	4871		

*-In red.

Grasselli Plant.
\$ per ton.

Silver Refinery Cost, 1909.

(D-1)

	August.	September.	October.	November.	December.
Labor	.145	.173			
Supplies	.091	.046			
Reprs. of Machy. & Equip.	.049	.054			
Building Repairs,	.001	.019			
Fuel,	.046	.057			
Furnace Repairs,	.072	.026			
Insurance,	.004	.008			
Experimental work,	.004	.024			
Pole Account,					
High Grade Matte Treatment	.022	.021			
Anti-Slag desilverization,	.027	.096			
Steam, Power & Light,		.040			
By-Prod. Treatment,	.044	.021			
Work on Fluedust Scrub. Mud	.055	.050			
Total Grasselli Exp.	.563	.626			
Chrome Charges,	.897	.811			
London "					
	<hr/>	<hr/>			
	1.460	1.437			
Cost per 1000 ozs. Dore,	7.84	8.85			
Ozs. Dore skipt. to Chrome,	494,100	406,763			
Tons Cathodes,	2650	3505			

SILVER REFINERY COST, 1909.

Grasselli Plant. \$ per ton.	January.	February.	March.	April.	May.	June.	July.
Labor	.22	.24	.18	.18	.19	.162	.147
Supplies	.07	.06	.07	.07	.04	.032	.050
Repre. of Machy & Equip.	.09	.04	.03	.01	.05	.042	.027
Building Repairs.		.01	.01	.02		.001	.002
Fuel,	.11	.10	.07	.04	.04	.070	.060
Parance Repairs,		.05	.05	.21	.02	.025	.004
Insurance,							.003
Experimental work,	.01	.02	.01			.032	.010
Pole Account,	.02	.01			.01		
High Grade Matte Treatment			.01		.01	.022	.021
Anti-Slag desilverization,					.02	.048	.034
Steam, Power & Light,	.07	.07	.05	.06	.05	.069	.057
By-Products Treatment,			.04		.23	.179	.012
Work on Fluedust, Scrub. Mud						.023	.046
Excess B.P. Stack Cost,				.06			
<hr/>							
Total Grasselli/Expense,	.59	.60	.52	.65	.70	.705	.474
Chrome Charges,	.77	1.15	.79	.75	.02	.738	.723
London "						.169	.033
<hr/>							
Total	1.36	1.75	1.31	1.40	1.72	1.512	1.231
Cost per 1000 ozs. Dore,	7.89	7.43	8.07	9.16	8.24	9.49	---
Ozs. Dore shipt. to Chrome,	397,697	461,745	384,149	373,792	524,011	354,945	392,531
" " " " " London,						53,332	
<hr/>							
Tons Cathodes,	2312	2034	2354	2450	2505	2404	2601

Grasselli Plant. \$ per ton.	August.	September.	October.	November.	December.
Labor,	.728	.747			
Supplies,	.141	.119			
Repairs of M. & E.	.047	.051			
Building Repairs,	.002	.001			
Electro. Replacement,	.342	.368			
Preparing Starting Sheets,	.048	.055			
Slimes,	.059	.051			
Tank Repairs,	.055	.044			
Handling Slimes,	.031	.033			
Steam, Light & Power,	.084	.088			
Making Starting Sheets,	.183	.187			
Recasting anode Scrap,	.246	.223			
Electrolytic Current,	.740	.915			
By-Prod. Treatment,		.026			
Experimental Work,	.018				
Filter Press Reprs.	.009	.039			
	<u>2.732</u>	<u>2.947</u>			
Tons Cathodes Prod.	2650				
K.W.H. Electrolyte.	330981				
Deposit per K.W.H.	16.00				
Ave. Efficiency for Mo.	89.4				
Acid Loss by Inventory,	68143				

Crasselli Plant.

TANK ROOM COSTS, 1909.

(C-1)

\$ per ton.	January.	February.	March.	April.	May.	June.	July.
Labor,	.72	.77	.71	.69	.69	.719	.707
Supplies,	.16	.20	.18	.16	.19	.167	.127
Repairs of M. & E.	.06	.11	.06	.03	.09	.109	.127
Building Repairs,		.01				.014	.001
Electro. Replacement,	.31	.08	.24	.69	.20	.249	.286
Preparing Starting Sheets	.05	.04	.04	.04	.04	.045	.053
Slimes,	.19	.09	.07	.05	.07	.056	.032
Tank Repairs,	.15	.12	.09	.12	.15	.098	.106
Handling Slimes,		.03	.03	.02	.03	.028	.056
Steam, Power & Light,	.23	.17	.15	.09	.08	.080	.077
Making Starting Sheets,	.17	.15	.15	.17	.17	.167	.190
Recasting Anode Scrap,	.15	.45	.24	.18	.23	.139	.158
Electrolytic Current,	.66	.81	.71	.66	.79	.721	.784
By-Prod. Treatment,					.04	.039	
Experimental Work,						.016	.005
Filter Press Reprs,		.05	.04	.14	.02	.017	.043
	<u>2.65</u>	<u>3.05</u>	<u>2.71</u>	<u>3.04</u>	<u>2.79</u>	<u>2.665</u>	<u>2.723</u>
Tons Cathodes produced,	2512	2054	2354	2450	2505	2404	2601
K.W.H. Electrolyte,	262734	257481	288398	295062	312078	286044	325067
Deposit per K.W.H.	17.60	15.71	16.31	15.26	12.46	11.90	16.00
Ave. Efficiency for Mo.	87%	88.5	90.5	90.4	86.8	88.4	88.9
Acid Loss by Inventory,					42,170 $\frac{1}{2}$	83,794	83230

Grasselli Plant. \$ per ton.	POWER PLANT COSTS (Engine Room) 1909. (B-1)						
	January.	February.	March.	April.	May.	June.	July.
Labor,	.11	.10	.10	.09	.09	.095	.080
Supplies,	.02	.02	.02		.01	.013	.012
Steam,	.46	.43	.43	.43	.53	.504	.517
Repairs of E. & E. Building Repairs,	.01	.06	.01	.01		.007	.019
	<u>.60</u>	<u>.66</u>	<u>.61</u>	<u>.53</u>	<u>.64</u>	<u>.619</u>	<u>.630</u>
K.W.H. Generated,	328433	314004	349165	359030	382380	357075	410509
Less used in Engine Room	8123	7847	8705	7770	8439	7754	8586
Total Cost Basis.	320310	306157	340400	351260	373941	349321	401923
<u>DISTRIBUTION & CONTROL</u>							
Distributed:- Electro.	256235	251203	281434	288846	310388	284494	323360
L. & P.	64075	54954	58966	62414	63553	64827	78573
K.W.H. Cost Electrolytic, Light & Power,	.59 .62	.66	.60	.56	.64	.50	.634 .625

Grasselli Plant. \$ per ton	Power Plant Cost (Engine Room)				
	August.	September.	October.	November.	December.
Labor,	.081	.087			
Supplies,	.015	.018			
Steam,	.493	.518			
Repairs to L. & P.,	.009	.136			
Building Repairs,					
	<u>.596</u>	<u>.759</u>			
K. W. H. Generated,	421632	386649			
Less Used in Eng. Room	9537	13713			
Total Cost Basis,	<u>412095</u>	<u>373136</u>			
Distributed: Electro.,	329052	307383			
L. & P.,	83043	65803			
K. W. H. Cost., Electrolyt.	.597	.744			
Light & Power,	.597	.833			

(B-1)

An example of a mixture adapted for treatment by the process of this invention SPECIFICATION.

refining process having approximately the following composition:-

To all whom it may concern:-

Silver, 90 per cent or more,
Gold, 1 per cent,
Bismuth, 5 to 10 per cent,
Copper, 5 to 7 per cent,
Antimony, 32-1/2 per cent,

Be it known that I, AUGUSTUS E. KNORR, a Citizen of the United States, residing at Elizabeth, in the County of Union and State of New Jersey, have invented certain new and useful improvements in the

Process of Treating Metallic Mixtures,

A variable small quantity of (2 or 3 per centages) of which the following is a specification.

of Sulphur, and fractional percentages of Arsenic and Silica.

In the manufacture or refining of lead electrolytically, there occurs a slime or mud which forms at the bottom of the electrolyte and remains separate therefrom. This slime, while varying widely in composition, usually contains metals of three classes, namely (1) one or more bullion-forming metals (that is forming a bullion or mixture of pure metals under the conditions hereinafter referred to) such as gold, silver, bismuth platinum, which are comparatively difficult of oxidation, (2) one or more matte-forming metals such as silver and copper, which form a matte with sulphur, and (3) one or more comparatively easily oxidizable metals, such as lead and antimony. Similar mixtures are found in the flue dust occurring in other processes of manufacturing or refining lead, in the slime resulting from the electrolytic manufacture or refining of copper, and in connection with other metallurgical processes. Similar mixtures may be completed artificially from materials lacking one or more of the desired metals or classes of metals, or lacking the desired percentages of such metals or classes.

The present invention provides a process of separating such classes of metals or the individual metals or desirable mixtures or alloys thereof from the original mixtures containing all or less than all the classes stated, and containing in each class a greater or less number of metals than those specified.

green slime the bismuth and copper which were oxidized in the

slimes & drosses of less value of metals than those described
all of less than all the classes stated, and containing in each
mixture of alloys thereof from the ordinary mixtures containing
such classes of metals of the industrial metals of lead, zinc

The present invention provides a process of separating
such metals of classes.

metals of classes of metals, or treating the treated drosses of
materially from materials treating one or more of the treated
metallic processes. Similar mixtures may be composed
mainly of refining of copper, and in connection with other
refining lead, in the same resulting from the electrolytic
the line and occurring in other processes of manufacturing of
metals, such as lead and antimony. Similar mixtures are found in
antimony, and (2) one or more comparatively easily oxidizable
forming metals such as silver and copper, which form a waste with
which are comparatively difficult of oxidation, (3) one or more waste-
materials referred to) and as gold, silver, platinum, bismuth,
forming a portion of mixture of blue metals under the conditions
classes, namely (1) one or more portion-forming metals (such as
valuable metals in composition, namely containing metals of three
electrolytic and remains separate therefrom. This same, while
there occurs a waste of such metal forms at the bottom of the
in the manufacture of refining of lead electrolytically,
of which the following is a specification.

Process of treating metallic mixtures.

metals in the
state of new metal, have invented certain new and useful improve-
ments relating to the treatment of such metals in the country of Union and
be it known that I, AUGUSTUS E. KNOWLTON, a citizen of the
do all upon it may concern:-

SPECIFICATION

8970

(N/A)

Roasting An example of a mixture adapted for treatment by the
process of this invention is a slime from an electrolytic lead
refining process having approximately the following composition:-

- Silver, 20 per cent or more,
- Gold, 1 per cent,
- Bismuth, 6 to 10 per cent,
- Copper, 6 to 7 per cent,
- Antimony, 32-1/2 per cent,
- Lead, 10 to 12 per cent.

A variable small quantity of (2 or 3 per cent) of
Sulphur, and fractional percentages of Selenium, Tellurium, Arsenic and Silica.

Such mixtures have been previously treated by fusion
processes, additional fluxes such as soda, nitre and silica being
added to cause the entry of all metals except the silver and gold
into the slag. The operation of such fluxes commences after the
heating of the mixture, and their effect is only slowly distributed
throughout the mass, until which distribution the separation of
the silver and gold from the other metals is incomplete. The
metals other than silver and gold and copper are usually lost with
the slag.

According to the present process not only the gold and
silver, but also the bismuth (constituting a considerable "value" in
the mixture), copper antimony and lead are recovered, and any
selenium or tellurium present. The mixture is made self-fluxing so
that the smelting requires but a fraction of the time needed with
the old method of adding fluxes. Selective slagging is made
possible, the lead and antimony being preferable slagged out of the
mixture substantially to the exclusion of other metals.

The mixture is reduced to suitable condition for this
ready self-fluxing and selective slagging by oxidizing the lead and
antimony, as by roasting the slimes, or by mixing suitable
percentages of roasted and green (or raw, unroasted) slime or by
addition of suitable oxidizing agents. By the addition of the
green slime the bismuth and copper which were oxidized in the
subsequent treatment we may consider the lowest zone to be a bullion

green slime the platinum and copper anode were oxidized in the addition of suitable oxidizing agents. By the addition of the percentages of roasted and green (or lead) anode or by anode as by roasting the anode, or by mixing suitable lead self-fluxing and selective agents by oxidizing the lead and the mixture is reduced to suitable condition for final treatment approximately to the exclusion of other metals. Possible, the lead and antimony being preferable stages out of the the old method of adding fluxes. Selective fluxing is made that the anode is reduced but a fraction of the time needed with adjustment of temperature present. The mixture is made self-fluxing as the mixture, copper antimony and lead are reduced, and the anode, and also the platinum (constituting a considerable amount) in accordance with the present process not only the gold and the silver, but also of more valuable metals other than silver and gold and copper are partially lost with the anode and gold from the other metals is incomplete. The throughout the mass, must upon distribution and separation of heating of the mixture, and their effect is only slowly distributed into the slag. The operation of anode fluxes commences after the added to cause the entry of all metals except the silver and gold processes, adjustment fluxes such as soda, nitre and other being used. Some mixtures have been developed for the purpose of separating, bismuth, arsenic and stibic.

- of antimony and bismuth percentages of
- Antimony 22-1/2 per cent
 - Copper 5 to 10 per cent
 - Platinum 5 to 10 per cent
 - Gold 1 per cent
 - Silver 30 per cent or more

The process of this invention is a white from an electrolytic lead process of this invention is a white from an electrolytic lead an example of a mixture adapted for treatment by the

Roasting operation are reduced back to the metallic state by the metallic antimony and lead of the green slime, which of course are thereby oxidized. The slime as it comes from the electrolytic apparatus is preferably filter-pressed and dried and washed to recover the K_2O portion of the electrolyte carried therein. In the mixing of the slimes it is advisable to secure a thorough blending by first powdering as in a pebble mill and then mixing in a similar mill. A good mixture of such roasted and green slimes contains by weight from 50 to 70 per cent of the roasted slime. I have, however, obtained successful results using only the roasted slime, the extent of roasting being less than where green slime is to be added.

each zone is as follows:-
The mixture thus oxidized and made self-fluxing is next melted down in a non-reducing, preferably a neutral, atmosphere as for example in a closed crucible furnace, or in any furnace with a covering of slag adapted to protect the contents from the reducing atmosphere or from the flame, or in a closed electric furnace of the induction or other suitable type. There is volatilization of considerable quantities of antimony and lead and less volatilization of bismuth, copper and silver. These pass off in the fumes and may be largely recovered in the flue dust by collecting the latter and adding it to the charge with substantially the same effect as an addition of roasted slime. But such volatilization should be minimized by stopping the operation as soon as fusion is complete, and the ready fluxing and fusion of the mass is an advantage in this respect.

The fused mass ~~now~~ separates into three zones. The lower is a bullion or mixture of substantially pure metals, containing nearly all the silver and bismuth and substantially all the gold, and usually negligible or very small quantities of the other metals. The next zone is a matte containing a small portion of the silver and nearly all the copper. The top zone is a slag containing substantially all the antimony and lead and small quantities of bismuth, copper and silver. For the purpose of subsequent treatment we may consider the lowest zone to be a bullion

spreadment the element we may consider the lower zone to be a solution
 consisting of platinum, copper and silver. For the purpose of
 containing apparatusly all the platinum and lead and silver
 the silver and nickel all the copper. The top zone is a slag
 matte. The next zone is a matte consisting of a small portion of
 and mainly consisting of very small quantities of the other
 metals all the silver and platinum and apparatusly all the gold
 is a solution of mixture of apparatusly pure metals, consisting
 of platinum, gold, silver and copper. The lower
 zone is lead.

The lead may be separated into three zones. The lower
 zone is lead. The middle zone is a mixture of lead and silver
 and the upper zone is a mixture of lead and copper. The lead
 may be separated by stopping the operation as soon as a solution is complete
 in solution of roasted matte. But such a solution should be
 and adding it to the charge with apparatusly the same effect as
 may be largely recovered in the time that is collecting the latter
 of platinum, copper and silver. These are all in the lower and
 considerable quantities of platinum and lead and lead apparatusly
 in solution of other metals. There is apparatusly of
 apparatusly of flow the time, or in a closed electrostatic furnace of the
 a covering of slag adapted to protect the contents from the reducing
 as for example in a closed electrostatic furnace, or in any furnace with
 melted down in a non-reducing atmosphere, hydrogen, nitrogen, apparatusly
 the mixture thus obtained and made self-heating is next
 added.

the extent of roasting being less than where green matte is to be
 however, obtained successive results being only the roasted matte
 weight from 20 to 40 per cent of the roasted matte. I have
 with a good mixture of any roasted and green matte contains by
 by that powdering as in a beddie with and then mixing in a similar
 mixing of the matte is a suitable to secure a thoroughly blending
 recover the full portion of the electrostatic carried therein. In the
 apparatus is preferably better-dressed and dried and washed to
 metallic platinum and lead of the green matte, which of course are
 metallic platinum and lead of the green matte, which of course are
 roasting operation are reduced back to the metallic state by the

of gold, silver and bismuth, the next a matte of silver and
 metals from a button in the bottom of the vessel.
 copper, and the third a slag of antimony and lead. A table
 showing an actual analysis of a melt, giving the percentages
 of each of the principal metals in the different zones, is as
 follows:-

Table of Zone Contents for each metal.

	Ag.	Au.	Bi.	Cu.	Pb.	Sb.	Lead
Bullions--	93.3	100	87.6	2	0.6	1.2	
Matte----	6.4	0	0	93	0.7	0.2	
Slag-----	0.2	0	12.4	5	98.7	98.6	
	100.00:	100:	100:	100:	100:	100:	

Another table showing approximately the contents of
 each zone is as follows:-

Table of Contents for Each Zone.

	Bullion.	Matte.	Slag.
Ag	70	30.68	23.73
Au	1.13	43.90	44.90
Cu	2.60	1.08	0.16
Pb	0.26	trace	0.30
Bi	22.52		0.30
Sb	1.43	1.13	
Te	0.68	2.57	
	trace		

The three zones or classes of metal may be separated
 by tapping or pouring them separately. Or the entire charge
 may be poured and allowed to solidify when the three zones by
 reason of their widely differing specific gravities and their
 different solidifying temperatures and different coefficients
 of contracting in solidifying, may be readily separated.

From the three zones or classes of metals thus
 segregated, the individual metals or desired mixtures thereof
 may be obtained by various known or suitable methods.

The slag may be smelted to commercial antimonial lead,
 after previous removal of the more valuable metals, by maintain-
 ing it molten in a reducing atmosphere until the more valuable

The present invention is a method of separating the noble metals from the base metals of a mixture of the same. The method consists in heating the mixture in a crucible until the base metals are fused and the noble metals are in a molten state. The mixture is then poured into a retort and the noble metals are collected in a separate vessel. The process is repeated until the noble metals are completely separated.

The present invention is a method of separating the noble metals from the base metals of a mixture of the same. The method consists in heating the mixture in a crucible until the base metals are fused and the noble metals are in a molten state. The mixture is then poured into a retort and the noble metals are collected in a separate vessel. The process is repeated until the noble metals are completely separated.

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The present invention is a method of separating the noble metals from the base metals of a mixture of the same. The method consists in heating the mixture in a crucible until the base metals are fused and the noble metals are in a molten state. The mixture is then poured into a retort and the noble metals are collected in a separate vessel. The process is repeated until the noble metals are completely separated.

bismuth, the same style of apparatus being used for collecting selenium. The invention includes broadly the forming of a substantially self-fluxing mixture of bullion-forming metals and oxides of more readily oxidizable metals, the mixture being such as will when fused readily separate and from such metals may be recovered. Within this broad invention there may be many variations in the individual metals and in the zones into which they separate.

A valuable method of obtaining a suitable mixture for fusing is by the addition of copper slime (obtained from the electrolytic refining of copper) to the above described lead slime. Such copper slime differs from the lead slime usually in having all the metals except silver and gold and part of the copper in a more or less oxidized state and in carrying substantial quantities of selenium and tellurium and more sulphur than lead slime. By reason of this difference in composition a mixture of green copper slime and green lead slime serves to diminish the amount of roasting needed or to eliminate the roasting step from the process; although the copper slime alone is not readily self-fluxing by reason of its high copper content.

A particular advantage of the use of such a mixture is that the selenium and tellurium may be recovered by a converting process in which they have a similar effect to sulphur and pass off in fumes. The selenium sometimes occurs in considerable percentage and is a valuable product, and I believe that this invention provides the first means by which it has been practical to recover it and the tellurium.

Various apparatus may be used for carrying out the process.

Referring to the accompanying drawings, Fig. 1 shows more or less diagrammatically the melting of the mixture in a crucible, and Fig. 2 the blowing of the bullion in an externally heated retort to collect the

of the solution in an externally heated vessel to collect the
 solution of the mixture in a crucible, and the process
 of separating the solution of lead from the solution of
 bismuth, copper, antimony, lead and sulphur, which
 consists in roasting a portion of such slimes, forming a
 mixture of such roasted portion with a green portion of slime,
 the roasted portion constituting approximately 50 to 70 per
 cent of the mixture, and the slimes being finely ground and
 intimately blended, melting such mixture in a neutral
 atmosphere whereby a melt is obtained in three zones, the
 lowest a bullion of gold, silver and bismuth, the next a matte
 of silver and copper, and the third a slag of antimony and
 lead, separating said zones and treating them further to
 recover the metals therein.

The process of treating slime from the electrolytic
 production of lead, copper, &c., and consisting of a bullion-
 forming metals, matte-forming metals, and more readily oxidiz-
 able metals, which consists in roasting a portion of such slime,
 mixing the same with a portion of green slime, and fusing said
 mixture in a non-reducing atmosphere, whereby the mixture is
 readily fused and separates into three zones of bullion, matte
 and slag respectively, which are readily separable.

bismuth, the same style of apparatus being used for collecting
 selenium or tellurium.
 Though I have described with great particularity
 of detail certain specific processes embodying the invention,
 yet it is not to be understood therefrom that the invention is
 restricted to the particular processes described. For
 a chamber, closed except for an opening at the top of the
 modifications thereof may be made by those skilled in the art,
 exit of the flame gases, and into which flame from an oil
 burner C is introduced. The mixture melts and settles in
 about the position shown, the slag being indicated at D, the
 matte at E, and the bullion at F.

For the converting of the bullion, the same furnace
 B and oil burner C may be used. The bullion is carried in a
 retort C, into which air is blown by means of a pipe H, and
 the fumes are collected in a condenser J.

1. The process of treating lead slime containing silver, gold, bismuth, copper, antimony, lead and sulphur, which consists in roasting a portion of such slimes, forming a mixture of such roasted portion with a green portion of slime, the roasted portion constituting approximately 50 to 70 per cent of the mixture, and the slimes being finely ground and intimately blended, melting such mixture in a neutral atmosphere whereby a melt is obtained in three zones, the lowest a bullion of gold, silver and bismuth, the next a matte of silver and copper, and the third a slag of antimony and lead, separating said zones and treating them further to recover the metals therein.
2. The process of treating slime from the electrolytic production of lead, copper, &c., and consisting of a bullion-forming metals, matte-forming metals, and more readily oxidizable metals, which consists in roasting a portion of such slime, mixing the same with a portion of green slime, and fusing said mixture in a non-reducing atmosphere, whereby the mixture is readily fused and separates into three zones of bullion, matte and slag respectively, which are readily separable.

...the process of treating lead slime containing silver, gold, bismuth, copper, antimony, lead and sulphur, which consists in roasting a portion of such slime, forming a mixture of such roasted portion with a green portion of slime, the roasted portion constituting approximately 50 to 70 per cent of the mixture, and the slimes being finely ground and intimately blended, melting such mixture in a neutral atmosphere whereby a melt is obtained in three zones, the lowest a bullion of gold, silver and bismuth, the next a matte of silver and copper, and the third a slag of antimony and lead, separating said zones and treating them further to recover the metals therein.

...the process of treating slime from the electrolytic production of lead, copper, &c., and consisting of a bullion-forming metals, matte-forming metals, and more readily oxidizable metals, which consists in roasting a portion of such slime, mixing the same with a portion of green slime, and fusing said mixture in a nonreducing atmosphere, whereby the mixture is readily fused and separates into three zones of bullion, matte and slag respectively, which are readily separable

Though I have described with great particularity of detail certain specific processes embodying the invention, yet it is not to be understood therefrom that the invention is restricted to the particular processes described. Various modifications thereof may be made by those skilled in the art, without departure from the invention.

What I claim is:-

1. The process of treating lead slime containing silver, gold, bismuth, copper, antimony, lead and sulphur, which consists in roasting a portion of such slime, forming a mixture of such roasted portion with a green portion of slime, the roasted portion constituting approximately 50 to 70 per cent of the mixture, and the slimes being finely ground and intimately blended, melting such mixture in a neutral atmosphere whereby a melt is obtained in three zones, the lowest a bullion of gold, silver and bismuth, the next a matte of silver and copper, and the third a slag of antimony and lead, separating said zones and treating them further to recover the metals therein.
2. The process of treating slime from the electrolytic production of lead, copper, &c., and consisting of a bullion-forming metals, matte-forming metals, and more readily oxidizable metals, which consists in roasting a portion of such slime, mixing the same with a portion of green slime, and fusing said mixture in a nonreducing atmosphere, whereby the mixture is readily fused and separates into three zones of bullion, matte and slag respectively, which are readily separable

and the lead-bearing portion of the matte
 is separated into three zones of bullion, matte
 and the same with a portion of green matte, and the same
 matte, which consists in separating a portion of alloy matte,
 containing matte-forming metals, and more readily oxidizable
 portion of lead, copper, etc., and consisting of a portion-
 8. The process of separating the lead from the electrolytic
 recover the metals therein.
 lead, separating into zones and treating them further so
 of silver and copper, and the third a zone of antimony and
 bismuth and portion of lead, silver and platinum, the next a matte
 antimony, arsenic and lead is obtained in three zones, the
 matte-forming metals, and the same with a portion of
 cent of the matte, and the residue being finely ground and
 the treated portion containing approximately 20 to 40 per
 cent of alloy matte with a green portion of matte,
 containing a portion of alloy matte, forming a
 portion, silver, copper, antimony, lead and platinum, which
 9. The process of separating lead from the electrolytic
 matte is as follows:-
 and I claim as:-
 the process of separating lead from the electrolytic
 matte in three zones, the first a zone of lead, silver and
 bismuth, the second a zone of antimony, arsenic and lead,
 and the third a zone of copper, nickel, iron and manganese,
 which consists in separating the lead from the electrolytic
 matte and separating the lead into three zones, the first a
 zone of lead, silver and bismuth, the second a zone of antimony,
 arsenic and lead, and the third a zone of copper, nickel, iron
 and manganese, the process which consists in separating the
 lead from the electrolytic matte and separating the lead into
 three zones, the first a zone of lead, silver and bismuth,
 the second a zone of antimony, arsenic and lead, and the
 third a zone of copper, nickel, iron and manganese, the
 process which consists in separating the lead from the
 electrolytic matte and separating the lead into three zones,
 the first a zone of lead, silver and bismuth, the second a
 zone of antimony, arsenic and lead, and the third a zone of
 copper, nickel, iron and manganese, the process which consists
 in separating the lead from the electrolytic matte and separating
 the lead into three zones, the first a zone of lead, silver and
 bismuth, the second a zone of antimony, arsenic and lead, and
 the third a zone of copper, nickel, iron and manganese, the
 process which consists in separating the lead from the electrolytic
 matte and separating the lead into three zones, the first a
 zone of lead, silver and bismuth, the second a zone of antimony,
 arsenic and lead, and the third a zone of copper, nickel, iron
 and manganese.

7. The process of treating a mixture of bismuth and other bullion-forming metals and more readily oxidizable metals, which consists in separating the bullion-forming metals, sulphur, matte forming metals, and externally heated retort reducing fumes of bismuth oxide, and collecting the fumes, separate zones when fused, and fusing said mixture under non-reducing conditions to form a melt in three zones containing respectively the classes of metals referred to in the form of bullion, matte, and slag, which are readily separable from each other.
8. The method of extracting bismuth from bullion, which consists in separating the bullion in an externally heated retort producing fumes of bismuth oxide, and collecting said fumes.
9. The process of separating bullion-forming metals; matte-forming metals, and more readily oxidizable metals, which consists in recovering selenium or tellurium and therefrom by converting the mixture so that these elements pass off in fumes, three classes of metals occupy three separate zones in the melt, and separating the zones from each other.
10. The process of treating a mixture of metals of the described class, which consists in melting the bullion-forming metals, sulphur, the matte-forming metals, and oxides of the more readily oxidizable metals, with a covering of slag sufficient to protect the contents, fusing said mixture whereby the different classes of metals occupy different zones in the melt, and separating said zones.
11. The process of treating a bullion of silver, gold and bismuth containing also copper and lead which consists in introducing sulphur into the bullion while the latter is in the molten condition, so that the copper and lead separate from the bullion in the form of a matte, the process which consists in forming a substantially self-fluxing mixture of bullion-forming metals and the oxides of comparatively readily oxidizable metals, fusing said mixture whereby the different classes of metals occupy different zones in the melt, and separating the zones from each other.

- 8 -

1. The process of treating a mixture of metals, which consists in separating the bullion-forming metals, and bassemerizing the bullion formed thereby in an externally heated retort reducing fumes of bismuth oxide, and collecting the fumes.

2. The process of extracting bismuth from bullion containing it, which consists in bessemerizing the bullion in an externally heated retort producing fumes of bismuth oxide, and collecting said fumes.

3. The process of treating copper slime or a similar mixture, which consists in recovering selenium or tellurium therefrom by converting the mixture so that these elements pass off in fumes.

4. The process of treating a mixture of metals of the described class, which consists in melting the bullion-forming metals, sulphur, the matte-forming metals, and oxides of the readily oxidizable metals, with a covering of slag adapted to protect the contents.

5. The process of treating a bullion of silver, gold and bismuth containing also copper and lead which consists in introducing sulphur into the bullion while the latter is in the molten condition, so as to separate the copper or lead from the remainder of the bullion in the form of a matte.

7. The process of treating a mixture of bismuth and other bullion-forming metals and more readily oxidizable metals, which consists in ~~separating~~ separating the bullion-forming metals, and bassemerizing the bullion formed thereby in an externally heated retort reducing fumes of bismuth oxide, and collecting the fumes.
8. The method of extracting bismuth from bullion containing it, which consists in bessemerizing the bullion in an externally heated retort producing fumes of bismuth oxide, and collecting said fumes.
9. The process of treating copper slime or a similar mixture, which consists in recovering selenium or tellurium therefrom by converting the mixture so that these elements pass off in fumes.
10. The process of treating a mixture of metals of the described class, which consists in melting the bullion-forming metals, sulphur, the matte-forming metals, and oxides of the readily oxidizable metals, with a covering of slag adapted to protect the contents.
11. The process of treating a bullion of silver, gold and bismuth containing also copper and lead which consists in introducing sulphur into the bullion while the latter is in the molten condition, so as to separate the copper or lead from the remainder of the bullion in the form of a matte.

1. The process of separating the copper and lead from the solution in the form of a precipitate.
2. The process of separating the copper and lead from the solution in the form of a precipitate.
3. The process of separating the copper and lead from the solution in the form of a precipitate.
4. The process of separating the copper and lead from the solution in the form of a precipitate.
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9. The process of separating the copper and lead from the solution in the form of a precipitate.
10. The process of separating the copper and lead from the solution in the form of a precipitate.
11. The process of separating the copper and lead from the solution in the form of a precipitate.

Lead Bullion Cross-section of pig.

M. Adcock

John W. Root.
Mining Reporter 2-8-06.

No. 1.

260.2	260.6	264.1	259.6	260.9
264.6	250.0	249.0	249.0	264.6
263.5	257.0	242.0	258.0	263.5
261.0	259.5	261.5		

No. 2.

260.8	259.4	258.6	259.3	262.8
260.6	258.8	257.3	258.6	260.6
261.5	256.4	245.0	256.4	261.5
260.3	258.0	260.6		

D. R. W. Laboratory 12-13-05.

1.78 104.6	1.78 102.4	1.74 104.4	1.74 102.1	1.78 104.7
1.78 104.8	1.76 103.1	1.78 103.9	1.76 103.9	1.78 104.9
1.78 104.9	1.76 103.1	1.76 103.1	1.78 104.3	
1.78 103.7	1.78 104.3	1.78 104.7	1.78 103.9	

Sample bars.

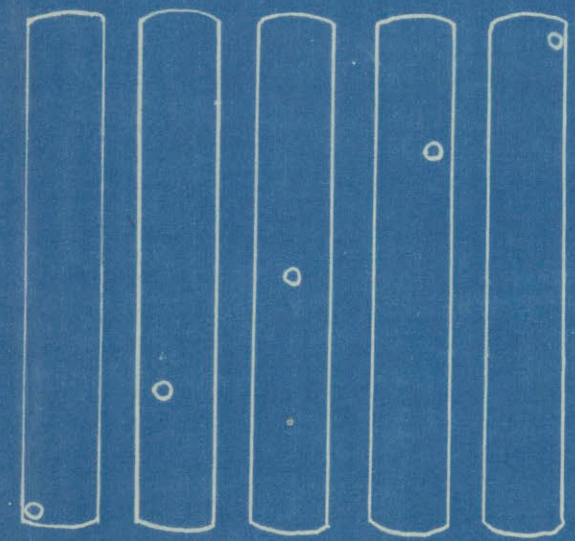
259.8	259.6	263.4	262.0	258.4
-------	-------	-------	-------	-------

259.3	258.5	264.9	261.1	259.3
-------	-------	-------	-------	-------

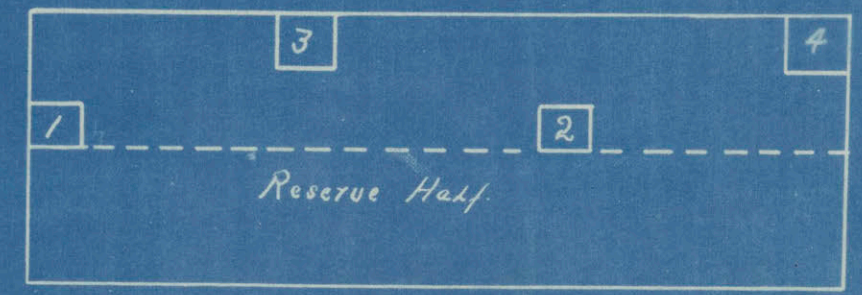
262.0	259.2	259.0	259.6	264.4
-------	-------	-------	-------	-------

260.1	259.2	258.3	259.1	261.0
-------	-------	-------	-------	-------

U.S.S. Bullion.



Sampling.



Sample Assay bar.

Lead Bullion
Plate 3 3-26-06.
Hg
56.

August 23rd, 1906.

Mr. Addicks:-

I have carefully gone over Mr. Betts letter of July 25th, and in compliance with Mr. Prosser's request submit the following commentary:

It may be that the presence of glue had something to do with polarization in case of Mr. Betts' experiments, but the trouble we had was not attributed ^{able} to that cause, since we worked with an electrolyte prepared from straight iron sulphate, copper sulphate, and sulphuric acid. In fact it seems to me instead of giving rise to polarization, glue should act as a depolarizer like any other readily oxidizable organic ~~substance~~ or inorganic substance. However that may be, the slimes should be properly washed and filtered before treatment in order to prevent fouling of the ferric solution with lead electrolyte, which at the same time would effectually remove this cause of trouble if such it be.

Mr. Betts then speaks of 3 further methods of treating slimes.

1. ROASTING WITH SULPHURIC ACID:

This is not new to us from our experience with copper slimes, where we know that drying them with adhering electrolyte (free sulphuric acid) will cause soluble silver ^{sulfate} to be formed. It is perfectly feasible ^a to thus convert all silver into soluble sulphate. An objection to consider in this mode of extracting silver is the very low solubility of silver sulphate in water, which is about 1 in 100. In the practice of parting this is overcome by working with concentrated sulphuric acid in very large excess, in which silver sulphate readily dissolves; and treating slimes direct in this way would hardly be practicable. Mr. Betts, by following the H_2SO_4 treatment with hydrofluoric acid, removes antimony and more silver, while bismuth and still another portion of silver

remains in the residue. Thus he scatters his silver in three different portions, each again to be separately treated for its recovery, which seems to me an unnecessary complication.

II. FUSING SLIMES TO AN ALLOY:

In this we have also anticipated Mr. Betts. It might be worth while, however, when we have exhausted other more direct methods in view, to try Mr. Betts' scheme of electrolyzing this alloy.

III. TREATMENT WITH ANTIMONY PENTA-FLUORIDE:

This scheme looks very good and is along the same lines I have experimented on in finding a solvent which will attack all of the base metals at once, (silver also, which is then cemented out) thus effecting at the first step a separation of the values from the other heavy metals. Mr. Betts' ~~antimony~~ Sb.F₅ solution acts very much like my ferric fluo silicate mentioned in resumé in Betts' process August 22nd, 1906. In fact I have come to the conclusion that the solvent power of that solution for antimony is due to the presence of ferric fluoride derived from decomposition of the fluo silicate.

The obstacle in the way of using this antimony penta-fluoride is again the same as I found; namely, the excessive corrosion of the carbon electrodes in regenerating the oxidizing solution.

Regarding Mr. McNab's theory that the acid loss in lead refining is due to deposition of silica and escape of hydro fluoric acid into the air, I cannot agree, since hydro fluoric acid, one of the most powerful acids known, would certainly not escape, but would merely displace fluo-silicic acid from its lead salt, forming lead fluoride.

That this should take place seems quite plausible, in view of my recent experiments in oxidizing ferric fluo silicate solution either chemically or by electrolysis. The fact that in these conditions dissociation of fluo silicic acid might take place, throws an entirely new light on the question of acid loss, which accordingly would be due to local anodic action, caused perhaps by stagnation of the electrolyte in the interstices of the slimes adhering to the anodes.

Mr. Betts' remarks on oxidation by nitric acid calls for no further comment as that question was considered by us some time ago.

In conclusion I would point out that Mr. Betts no longer appears to have entire confidence in his ferric sulphate process as originally laid down by him, which he virtually acknowledges by submitting a number of altogether different propositions. As long as he does not "stand pat", therefore, it seems fair to me that any further propositions he may have to make should not take ^{precedence} ~~precedure~~ over over new lines suggested by our own work and which are the result of our own experience.

If Mr. Betts had consistently stood by his first process I would feel inclined to attribute its apparent short-comings to a lack of comprehension on my part of its proper working conditions. But, under the circumstances, I feel that I have done my best with it, and that I could do better with one or the other of the more direct methods suggested.

August 22nd, 1906.

Mr. Addicks:-

RESUME ON BETTS' CHAIN OF OPERATIONS FOR TREATING SLIMES
WITH FERRIC SULPHATE.

1 PRELIMINARY: Slimes must be carefully washed from lead electrolyte in order to prevent precipitation of lead sulphate with the slimes. (Also to remove glue according to Betts.)

2 FERRIC SULPHATE BOILING: To obtain the full benefit of ferric sulphate oxidation an excess must be employed, taking silver into solution. I find also that much antimony and some bismuth go into solution, partly separating out on standing and cooling.

Solution after standing: Sb. .066%
Bi. .019%

Cement Silver:
from ferric sulphate boiling:-

Ag.	87.29
Pb.	1.95
Sb.	1.33
Bi.	.54
Cu.	Trace
Se.	"

3. ELECTROLYSIS/ TO FERRIC SULPHATE: Our experiments show that this, with proper attention, is ~~xxxx~~ a working proposition, with recovery of copper and regeneration of the electrolyte. The antimony and bismuth accumulating in the solution appear to have no influence on the oxidation of ferrous salt. The electrolysis can be safely run at a density of 10 amperes per square foot at from 1.4 to 1.8 volts.

4. FILTERING AND WASHING OF SLIMES: Filtering and washing of slimes from ~~xxxx~~ ferric sulphate treatment must be very thorough in order to prevent fouling of the ensuing antimony solution. At this stage I find that the slimes and solution filter and wash very badly indeed.

5. HYDROFLUORIC ACID TREATMENT:

This is done cold and in concentrated solution and seems to present no special difficulty

BISMUTH: again, however, goes ~~gas~~ into solution along with the antimony.

<u>ELECTROLYTE:</u>	Sb.	9.65%
	Bi.	.52%
	Cu.	.67%

The presence of copper can be largely avoided by a more careful washing. For depositing antimony a high current density can be employed, say 100 to 200 amperes per square foot.

6. FILTERING AND WASHING:

The slimes from this treatment are easily filtered and washed.

7. FINAL SLIMES:

The final slimes show a great concentration of lead, while all other constituents (except gold) have been eliminated in part at least, or entirely, as shown by comparison with the original slimes.

<u>ORIGINAL SLIMES:</u>	<u>FINAL SLIMES.</u>
Ag.	21.94
Cu.	5.47
Pb.	9.43
Bi.	9.71
Sb.	32.67
	12.36

	53.37
	3.22
	1.14

According to these results the process does not appear to work out as intended by Mr. Betts. The weak point seems to be that instead of successively eliminating the different constituents and concentrating the values in a final product, there is a scattering of all the elements at each succeeding step and the final result is a concentration of all the lead with the gold and only part of the silver. To be sure a preliminary treatment with an oxidizing fluo silicate such as copper or ferric fluo silicate would remove most of the lead, but at the same time taking out

some bismuth and antimony, thus further adding to the complication.

I have indicated how a dry process would effect a concentration of values on the one hand, and a separation of base metals into non-volatile slag, (copper and lead) and flue dust, (volatile oxides of bismuth and antimony).

Under the impression that a wet process would prove more acceptable to the management, I have made some experiments with a view to separating the slimes constituents, by means of a solvent, into a solution containing all or most of the base metals, and a residue containing the values.

I found a powerful solvent in ferric fluo silicate, but it unfortunately turned out that regeneration of ferrous to ferric fluo silicate by chemical means was accompanied by partial decomposition into fluoride and silica, ^{while} ~~which~~ by electrolysis the same difficulty was encountered, the carbons being very severely attacked at the same time.

I then turned my attention to perchloride solutions, such as, for instance: perchloride of antimony. I found that with a strong solution of $Sb Cl_5$ I could practically completely dissolve the slimes, and by cautious ^{addition} ~~attention~~ of more slimes ~~reprecipitated~~ reprecipitated the silver, by which means I obtained a residue of the following composition:

Ag	53.08	
Pb.	16.08	
Sb.	.40	
Bi., Se., Te.,		Trace.

The slimes residue from this treatment contains practically nothing but lead in the form of sulphate, in addition to the values, and smelting the same to dore would be a very simple proposition.

On cooling lead chloride crystallizes out of the solution of mixed chlorides, and if desired can be more completely precipitated by

addition of sulphuric acid

This leaves a solution of the chlorides of copper, bismuth and lead, from which copper can be cemented out with met. antimony, or plated out by electrolysis, by which latter means at the same time $SbCl_5$ is regenerated to be used over again. Bismuth may be allowed to remain in the solution until it has concentrated sufficiently to warrant separation from antimony. This separation might be effected in several different ways; for instance, by selective electrolysis; or by taking advantage of the different volatility of the chlorides: $SbCl_5$ boils at $220^\circ C.$, and $BiCl_3$ at $450^\circ C.$, just the same as I separate antimony and arsenic analytically. Or they might be separated according to Borchers by electrolysis in an alkaline sulfide electrolyte in which bismuth is not soluble

Another line along which the problem of slimes treatment could be attacked, is based on the observation that slimes, when they are dried in bulk, undergo spontaneous oxidation, the heat thereby developed being sufficient to bring the mass to the point of ignition. It then burns very much like tinder, the combustion gradually proceeding through the entire mass with the formation of very little fumes. Analysis of such oxidized slimes is now being made. I find that these slimes are soluble to the extent of nearly 60% in hydro-chloric acid, 200 grams of such oxidized slimes was digested with 200 c.c. of strong hydro-chloric acid filtered and washed with hot acidulated water. The dry residue weighed 82 grams and showed the following composition:

Ag	45.44%	Sb.	1.95%
Cu:	Trace	Se.	Trace
Pb:	19.29%	Te.	"
Bi.	Trace		

These slimes residue was fused with one-half its weight of a mixture of 3 parts sodium carbonate and 1 part niter.

A silver button was obtained weighing 38 grams and assaying 94.41% fine Ag.

Silver in slag from above fusion:

Ag in Slag	2.285 grams
Ag in Button (38 x 94.41)	35.826 "
Total Ag recovered	38.111 "

Another method which would at once give a separation of antimony and bismuth, (and arsenic if present) on the one hand, from gold, silver, lead, and copper on the other hand, is by direct dry chlorination at a temperature above the boiling point of bismuth chloride (450° C.) which is away below a red heat and perfectly safe as regards silver chloride.

Altogether I would like very much to turn my entire attention to following some of the lines above indicated, and would, therefore, like to be relieved, for the present at least, from further work on the Betts process, which thus far does not appear to offer a complete and satisfactory solution of the problem in hand.

CHEMICAL BEHAVIOR OF CHLORIDES UNDER VARIOUS CONDITIONS:

	COLD WATER	HOT WATER	DIL. HCL	Acid solutions of Chlorides	Boiling point.
Ag.Cl	Insol.	Insol.	Insol.	Sol.	Volatile above a red heat.
Pb.Cl ₂	Nearly Insol.	Sol.	Same as in water.	Same as in water.	"
REXXKX Cu.Cl ₂	Very Sol.	Very Sol.	Very Sol.	Very Sol.	"
Cu ₂ Cl ₂	Insol.	Insol.	Very Sol.	Very Sol.	"
Sb.Cl ₃	Prec. as Oxy.-Chloride	Prec. as Oxy.-Chloride	Sol.	Sol.	212° C.
Sb.Cl ₅	"	"	Sol.	Sol.	"
Bi.Cl ₃	"	"	Sol.	Sol.	450° C.

NOTES ON TRAIL SLIMES TREATMENT.

June 4th, 1907.

(See Graphic Scheme). Slimes slush is baled out of tanks with pails and scoops directly into copper cars provided with a drop bottom rubber plug.

Six (6) of these cars do the work, and 15 to 16 filled cars constitute a day's product. They are raised about 17 feet above tank room floor and run on track over slimes washing tanks and contents dropped. Wash tanks are filled one-half full with a slush and left to settle for one hour; the solution is then decanted into settling tanks previous to being put back into the electrolyte.

6" of water or dilute wash water from previous washings are added and the operation repeated. Then 12" of water are added and the solution boiled and agitated for one-half hour with live steam, introduced into the tank through a 1-1/2" copper pipe perforated with 1/8" holes, then let settle and decant into settling tank, provided for the different routes the solution is to take. These are as follows:

Solution above 20° Be to electrolyte
Solution below 20° Be and above 5° Be to evaporators
5° to 0 back to first slimes washing.

Washed slimes are dropped into wooden filter tank, provided with cotton duck cloth filters and arranged in the manner of sucking tubs. Suction is produced by a small belt driven Rand Air Compressor with double pistons 4-1/2" in diameter. In these tanks the acidity of the slimes is ascertained

and a sufficient amount of caustic soda solution is added for neutralization. This is done apparently to prevent the formation of H_2S on the subsequent Na_2S treatment. The next step is to shovel slimes into the iron leaching tanks, where they are treated with a solution of sulphide of soda, freshly made up, or from electrolyte returned in an improved state from the subsequent circular process. The amount of fresh solution that has to be made up contains approximately 2400# of commercial sodium sulphide, strengthened by the addition of 25# of brim-stone. Presumably this is the amount to compensate for the sulphur loss by forming copper, silver, lead, bismuth, sulphides, etc.

The leaching tanks are filled with 3" to 4" of slimes to each batch and boiled for two hours with a closed iron pipe steam coil, the slimes being agitated with a wooden paddle. Sulphur is added again during this operation. When solution shows to be 19 degrees Be, it is dropped in storage tank, from which it is drawn into circulation through the electrolytic tanks as required.

Mr. McNabb states that the solution contains about 9 grams of Na_2S and three grams of antimony per 100 cc. The temperature of the solution in the electrolyte tanks runs from 45 to 60 degrees C. At the end of the run the degree Be is about 16, the drop being 3 degrees Be.

Ten (10) electrolyte tanks are provided, arranged in two cascades of 5 tanks each, 19 electrodes per tank, approximately of dimensions of 25" X 36". The cathodes are of iron or steel 1/8" thick; the anodes are of sheet lead, made on starting sheet table, weighing approximately 40#

each.

The voltage per cell is about 1.5 volt, varying according to amperage, which runs from 2200 to 3000 amperes. The ampere density per square foot is 10 to 14 amperes. Impoverished solution from electrolyte tanks goes into evaporator and is boiled up again to 19 degrees Be, from there part of it goes back ~~into~~ to the circulation and part of it to the leaching tanks.

Mr. McNabb stated they had circulated the solution as much as 35 times, but he was not quite certain as to how to dispose finally of the foul liquors, or how to regenerate them. I understand that the recovery of antimony was about 70% of that contained in the bullion, which would amount to 700# per day on a basis of 50 tons of bullion at 1% Sb. The deposit is brittle and hard; a thickness of 1/8" to 3/16" is deposited in four or five days. The cathodes are drawn out of the tanks periodically and the deposit knocked off by slight taps with a mallet. I understand that the deposit contains a few percent of arsenic, which is to be expected, and so far the Trail people have not been melting any of their antimony into commercial bars, and are accumulating their material for a clean up. They were experimenting at the time I was there, to get the star on their ingots that the trade demands. I understand that the efficiency is extraordinarily low, which is apparent when 800 to 1100 KW hours are spent daily in the recovery of 700# of antimony.

The slimes residue from the sodium sulphide leaching is ~~the~~ then transmitted to two iron filter tanks, arranged as sucking tubs, and is from there put into drying pans arranged over a return flue of the slimes roasting furnace. From there it is directly dropped through a hopper on the roaster hearth.

The inside dimensions of the roasting furnace are about 40' long by 6' wide. The fire flue is constructed under the hearth and the roasting is therefore altogether done with heat radiating from the arch under the hearth. As by the time they reach the roaster; the slimes should have lost about 25% of their original weight, the total quantities treated in roaster daily should amount to little more than a ton. The material is worked by hand from one side of the furnace only, towards the fire place and attains a cherry red heat near that point.

The roasting operation is, of course, oxidizing, having for its purpose a change of the sulphides into sulphates. The finished material is hoed into large iron buckets set under roaster doors, and has the appearance of calcined anhydrous sulphate scale. The roasted slimes are then leached in dilute sulphuric acid (?) (Do not remember whether water is first added and acid afterwards), which dissolves the sulphates of copper and silver and any Cu. oxides present. The success of this leaching depends largely on the successful previous roasting operation. The Trail people have been widely varying results as to the percentage of the copper and silver extraction, and I could not procure any definite data on this matter. While they expect to reach a very high percentage of copper extraction, only rather a small percentage of the silver is recovered as sulphate. As a proof of the very different results obtained, I may mention the Silver House Foreman's statement, that it took anywhere from 3 to 7 days to blow a charge of dore'.

The solution from the last leaching operation goes

to the silver cementing tanks with the bulk of the silver sulphate solution from the dore' kettle, while the slimes are charged in the dore furnace after being accumulated on an inclined drying bench.

To the slimes in the dore' furnace, coal dust and silica are added; the coal dust apparently for the purpose of reducing the lead sulphate. Silica is added particularly to slag the lead. Blowing with high pressure air at different stages of the melt, is also resorted to, furnished by a little Rand Compressor, which takes its air mostly from the space in bottom of the filter tanks. The blow pipes do not enter the metal bath. The first slag has the appearance of litharge and silicate of lead, said to be very low in values. The second slag produced, after the metal bath shows a clean face, resembling very much the type of slag we used to produce at Chrome, and which we called silver matte, disclosing its copper contents by a pink, reddish hue. This slag is said to run from 80 to 100 ounces of silver, and all slags from the reverberatory, if low enough in values, are delivered to the Smelter.

The dore' bars cast are extraordinarily thick, being about 2", with a few holes cast in them.

The parting of the dore' has no unusual features. The siphoning arrangement is very neat, the siphon being started with steam, connection being made to steam pipe with a rubber hose. I believe this apparatus was introduced at Chrome. Time for settling in settling pot, about one hour.

The silver is melted in a water-jacketed furnace, consisting of five pieces, four sides and bottom, being held

together by buck stays and clamps. This furnace is exactly the same size and construction as the dore' furnace bath, having their outlets into the same flue. The gases are conducted through a system of chambers, which are separated, the cooling being effected by up and down takes from one chamber to another; these are made of sheet iron pipe, approximately 5' in diameter and running about 20' above the roof of the chamber. The flue is approximately 300 feet in length and about 14' wide, but I understand this has been short circuited to about one-half, as the draft was insufficient.

WT/EJ

(Signed) Th.

C O P Y.

Grasselli, Ind., June 15, 1907.

WT-2084

L.R.S.R
TRAIL.

Mr. H. A. Prosser, Gen. Manager,
508 Dooly Block,
Salt Lake City, Utah.

Dear Sir:-

Trail Slimes Treatment.

Enclosed please find analyses on samples taken at different stages of Trail Slimes Process. The samples, of course, do not represent the same batch of slimes and are for that reason only to be considered as an approximate guide. However, the percentage of antimony extraction is about as stated by the Trail people. Assuming that the original slimes ran from 25% to 30% antimony, it would in this case be somewhere about 75%. The Copper extraction is apparently not as good, being only about 55%, but as the roasting operation seems to be a hit or miss affair, better results may be frequently obtained, similar to my experiments with roasted Cu. slimes at Chrome, which gave a nearly complete extraction.

If it takes them on an average of four days to blow the stuff to dore, it strikes me that cupelling with additional lead could be used to advantage at that stage of the process. I must ask Mr. LeBarthe if he ever considered this. A good deal of litharge would, of course, be produced, but the process is so much faster that it ought to receive some consideration.

Some silver extraction is noticeable, comparing the variation of silver contents with the increasing ratios of gold and lead as a guide.

Sample of electrolytic antimony showed 20.22 ounces of silver, and .235 ounces of gold per ton. I could not venture any opinion as to how this happened, but that probably the solution from the leaching vat contained a good deal of suspended slimes. However, the gold is twice as high as its ratio to silver in the slimes.

We will run a test on a batch of slimes at Grasselli as soon as possible.

Very truly yours,

WT/EJ

(Signed) Wm. Thum.

C O P Y.

Grasselli, Ind., June 7, 1907.

WT-2063

Mr. H. A. Prosser,

508 Dooly Block,

Salt Lake City, Utah.

Dear Sir:-

Trail Slimes Treatment.

Referring to your 4410, May 29th, Enclosed please find notes on Trail Slimes Treatment, also graphic scheme of movements of slimes.

I also procured the following samples of process products, on which determinations are as yet not made. The results should throw some light on the present work done at Trail:

1. Slimes after sodium sulphide leach,
2. Slimes after coming from roaster,
3. Slimes after leaching Cu. and Ag. sulphates,
ready for Dore' Furnace,
4. Metallic Electro-deposit of antimony.

The Trail slimes process is working quite satisfactorily now, as far as I could observe, and should be successful if all difficulties involved by the numerous detail have been overcome.

The most favorable feature of the process, in my opinion, is the separation of the Sb and also the As as the first operation; any process that accomplishes this besides recovering the Sb in a fairly pure state should be a good one.

As weak points, may be mentioned that As, Te, Se and Sn will be dissolved with the antimony and to what extent it will go into the deposit will be disclosed by the analysis of the sample of electrolytic Sb. However, I consider the

method superior to ours, unless the 7% Ag (Mr. Addicks letter #2555 of June 3rd, addressed to you) can be recovered as a button from the slags.

The cost of chemicals for sodium sulphide, brim-stone, etc., is quite high, and in our case should amount to

3000# Na ₂ S	at 1.4¢ per lb.	\$42.00
50# Brim-stone	at 1.2¢ per lb	.75

or approximately \$28.50 per ton of slimes. There is also some expense for caustic soda to neutralize the acid in ~~six~~ slimes previous to leaching with Na₂S, but this depends largely on the way the slimes have been washed, and I understand that they frequently run in the neighborhood of 1% of acid.

I do not know, and I do not believe that they know fully at Trail to just what extent any such metals as Bi, Pb, etc. enter into combination as sulphide when leaching with Na₂S. I assume, for instance, that lead will be covered with a film of sulphide. Copper, of course, will be acted upon best of all, and the separation of this metal should be the most satisfactory.

The roasting ~~work~~ needs most skillful handling, and I do not believe that the three men who divide the days shifts between them, are turning out a very uniform product.

The solution from the Cu and Ag leaching tanks has the greenish blue color, a mixture of Cu and Ag would have, but this may be due to other impurities.

The dore' furnace work varies very much according to the success of the preceding treatment, and it is of course evident that if the slimes contained only Pb, Bi, Ag and Au, the operation of bringing same to dore' would be a very simple one, which apparently it is not.

Unfortunately I did not bring along with me a line of samples of the different stages of the dore' furnace slag, but I was assured that they are low in silver. I could not ascertain whether this low silver is due to a greater amount of slags, but the accumulation for a charge that was nearly finished, looked to be a reasonably small amount. I intend to take this subject up with Mr. LeBarthe by letter.

In this connection I would like to be advised just what policy you would suggest to be followed with Trail regarding some further information on slimes treatment. They have a fairly good process, which they will no doubt protect by patents as soon as they have brought it on a basis satisfactory to them. We also have a process, which I assume will be protected by patents. It really appears to me that our process is more suitable for them than it is for us, and vice versa.

We produce slags and other furnace products, which would be more easily handled in connection with a lead smelter, while they with their leaching process, are under heavy expense for chemicals, which cost them nearly twice as much as they would cost us. From the data so far obtained, we could experiment on similar lines as trail, and probably find a combination which would be an improvement on their methods, and would not be considered an infringement, which the Trail people would no doubt regard as such, if they have furnished us with all the detail of their process.

Kindly advise me whether you think it advisable that I should ask for further information from them on this subject.

WT/Ej.
Encl.

Very truly yours,
(Signed) Wm. Thum.

June 14th, 1907.

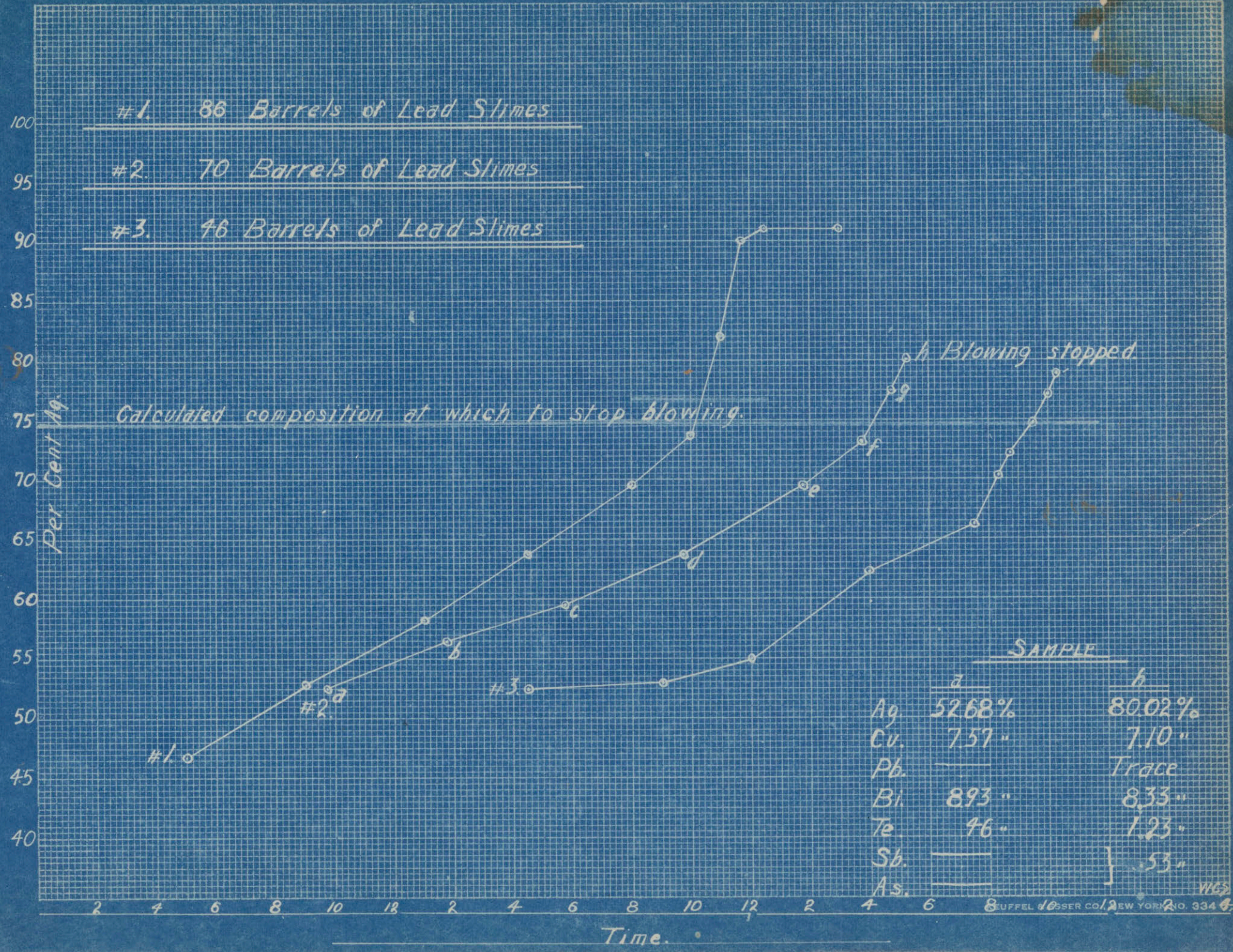
Mr. Thum:-

Analyses as per your directions of Slimes Samples
from Trail, B. C.

	<u>Ag.</u> <u>Oz.Ton</u>	<u>Au.</u> <u>Oz.Ton</u>	<u>Sb.</u> <u>%</u>	<u>Bi.</u> <u>%</u>	<u>Cu.</u> <u>%</u>	<u>Pb.</u> <u>%</u>
1. After leaching with Na ₂ S and S	+ 8469.55	+ 40.95	6.58	1.00	6.32	19.14
2. After going through com- bination dryers and roasters.	+ 6244.15	+ 33.00	5.30	.83	4.53	18.86
3 Ready for Dore' Furnace.	+ 8792.40	+ 47.05	8.90	.70	2.84	24.36

+ Fire Assay Uncorrected.

(Signed) F. C. Ryan.



	SAMPLE	
	a	b
Ag	52.68%	80.02%
Cu	7.57	7.10
Pb	—	Trace
Bi	893	833
Te	46	1.23
Sb	—	.53
As	—	—

Mr Chas. T. Bender
7014 E Jacob

LEAD SLIMES.

	Au.3	Ag.	Cu.	Sb.	As.	Pb.	Bi	Fe.	Si O2
Trail		28.15	8.83	27.10	12.42	17.05	No	1.27	
"		23.05	22.36	21.16	5.40	10.62	"	1.12	
Monteroy Mex	29.1	32.11	1.90	29.51	9.14	9.05	Tr	.49	
Mexican		4.7	9.30	25.32	44.58	10.30	.52		
"		3.9	6.38	50.16	15.23	5.30	19.74	No	
Trail	180.33	31.62	1.40	35.71	4.91	9.57	1		
"	81.99	32.21	6.60	24.60	2.20	12.60			
Rich Pb. Parkes.		78.45	12.56	4.12		3.00	.88		
Trail		29.20	7.10	30.50	6.10	10.20			
"		31.90	7.70	37.60	2.80	12.60			
E. L. Docter		2.44	7.82	75.34	.24	12.23	1.95		
D. R. W.	51.05	15.35	6.45	29.82	.15	14.95	8.52		6.50

ELECTROLYTIC REFINING OF LEAD IN
FLUOSILICATE SOLUTION

By HANS SENN

IN view of the interest which has been aroused by the successful refining of lead as carried out by Mr. A. G. Betts, in fluosilicate solution, the author conducted a series of experiments in order to ascertain whether the improvement in the quality of the lead deposit is due to the lead fluosilicate electrolyte employed by Mr. Betts, or whether it is only to be attributed to the influence of the small addition of gelatine which is made to the solution. He further endeavored to find out the best conditions in regard to current density, temperature and concentration of solution, at which to carry out electrolysis. An interesting description of the experiments, with statement of the author's conclusions, appeared in a late issue of the *Zeitschrift für Elektrochemie*. He conducted his experiments in glass vessels with about 450 cubic centimeters of electrolyte, a cathode being located between and parallel to two anodes in a vertical position. Of 100 grams, the electrolyte contained 8.3 grams of lead and 11.3 grams of free acid.

He found that without gelatine, with a current density of 1.07 amperes per square decimeter, a very irregular deposit was obtained; while under the same conditions, with an addition of 0.1 gram of gelatine to the liter, the deposit was quite smooth and of metallic luster. Further experiments also showed that the addition of gelatine had quite a decisive influence on the quality of the deposit. The author next endeavored to study the influence of varying the current density, the temperature and the concentration of the electrolyte. He found that differences of temperature had no action on the character of the deposit, while the influence of the gelatine was quite marked.

He also endeavored to ascertain whether an equally beneficial result could be obtained by the addition of gelatine to a lead nitrate and a lead acetate solution. Under ordinary conditions, without the addition of gelatine, the lead deposit from a lead nitrate solution is very irregular. It was shown that the quality of the deposit was decidedly improved by the

addition of the gelatine to the electrolyte. The quality of the deposit, however, is not at all comparable to that obtained from lead fluosilicate solutions, since the material is brittle, falls easily apart and does not show the property of bending easily, which is typical of lead. The addition of gelatine also acted favorably in the case of the lead acetate solution, but the quality of the deposited lead was still worse than that obtained from the lead nitrate solution. The author concludes that while the addition of gelatine prevents the formation of larger single crystals in the deposit of lead, yet, in the first place, the use of a lead fluosilicate electrolyte is responsible for the excellent condition in which the lead is deposited by the Betts method. The gelatine in the latter case only prevents the last tendencies for the formation of single crystals which also exist in this electrolyte.

A further series of experiments made by the author bore upon the subject of electrolytic refining of cadmium in fluosilicate solution, during which he obtained fine-grained deposits of the metal. He comes to the following conclusions: (1), It is possible to electrolytically refine lead and cadmium in fluosilicate solutions, so that coherent metallic deposits are obtained; (2), an addition of gelatine to the electrolyte prevents the formation of solitary crystals or trees at the cathode; (3), with increasing dilution of the electrolyte or increasing current density, the good properties of the deposit decrease, but even here an addition of gelatine improves matters; (4), the conditions to be chosen for obtaining a good lead or cadmium deposit are as follows: Electrolyte: contents of free hydrofluosilicic acid, about 11 per cent.; contents of metal, 4 to 8 per cent.; of lead, 2.5 per cent. of cadmium; current density, 0.5 to 1 ampere per square decimeter; addition of gelatine for lead, 0.1 grams; for cadmium, 0.3 gram per liter of electrolyte.

The second part of the author's investigations bore upon the refining in lead fluosilicate solution of lead and copper, lead and bismuth, lead and antimony and lead and

platinum alloys, and upon the changes of a lead fluosilicate solution during use. He comes to the following conclusions:

(1). In fluosilicate solutions, it is possible to refine anodes consisting of a copper-lead alloy, which contains the maximum of copper soluble in lead, with current densities of 0.5 to 1.5 amperes per square decimeter, with the production of a deposit of pure lead at the cathodes, while a slime remains at the anode which still contains about 10 per cent. of lead.

(2). An alloy of lead and bismuth, which contains 12 per cent. of bismuth, can be electrolyzed with the production of pure lead at the cathode and a slime at the anode which still contains about 20 per cent. of lead, the operation taking place at a current density 0.5 to 1.5 amperes per square decimeter. If the contents of bismuth are increased to 26 per cent., the electrolytic refining only succeeds with current densities of 0.5 to 1 ampere per square decimeter. If a current density higher than the above is used, bismuth is also dissolved at the anode and deposits together with the lead at the cathode.

(3). It is only possible to work antimony lead alloys with 10 per cent. antimony for

pure lead at current densities of 0.5 to 1 ampere per square decimeter. In this operation about 30 per cent. of lead remains in the antimony slime. When a current density of 1.5 amperes per square decimeter is used, antimony goes into solution at the anode and is deposited with the lead at the cathode.

(4). Alloys of lead and 10 per cent. platinum furnish during refining in fluosilicate solution at the cathode pure lead, at the anode a slime which consists of the crystalline compound $PtPb_2$. From the latter no considerable quantities of lead are dissolved anodically.

(5). Hand in hand with the electrolytic refining of lead alloys, there goes a gradual decomposition of the electrolyte, inasmuch as $PbSiF_6$ decomposes at the anode with the formation of SiO_2 and PbF_2 , which are present in the anodic slime.

(6). The yield at the cathode is on an average 98 per cent., but the quantities of lead which go into solution at the anode are always somewhat more than 100 per cent. when pure lead is used, which fact occasions a slow consumption of the free hydrofluosilicic acid in the electrolyte.

ZINC MINING AND SMELTING IN SOUTHWESTERN VIRGINIA

By EDWIN HIGGINS, JR.

MR. HIGGINS, in a recent article, running through two issues of the *Eng. and Mg. J.*, gives a concise description of the present condition of the zinc industry of Virginia, with short notes on its history. Lead mining was begun at Austinville by Colonel Chiswell some time between 1750 and 1776. The Bertha mines were opened in 1879 and the Delton in 1902. Of these three important properties, only the oldest is now producing. The Delton mines have produced in all about 8,000 tons of carbonate and silicate of zinc. The mine run at first was concentrated four into one, but later six into one. The finished concentrate ran 40 per cent. zinc, 0.5 per cent. lead and 5 per cent. iron.

The Bertha mines, which were long worked by the Bertha Zinc Company, have been often described. They are not now worked except by open pit for limonite, but the methods de-

veloped may be of interest. The ore occurred under a heavy cover of residual clay in pits and around pinnacles produced by the erosion of the Lower Silurian limestone. The ores were carbonates and silicates of zinc. The deposits ranged from 1 to 25 feet in thickness and averaged per acre 15,000 tons of unwashed ore, containing 26 per cent. zinc.

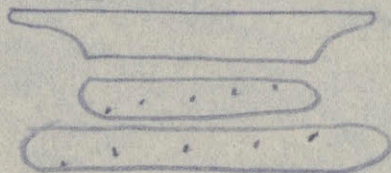
The mine was first worked by open-cut after the removal of the clay. A system of underground mining was then introduced, the ore being reached through shafts of $3\frac{1}{2}$ feet diameter, sunk in the clay. They were fitted with removable iron castings, bolted together in sections. Plank shafts 38 inches square, inside measurement, were also used, being sunk to the limestone bottom at its deepest points. Timbered shafts were then driven in the ore, following it around the chimneys, and when the latter were encircled, the second drift was

- 1 -

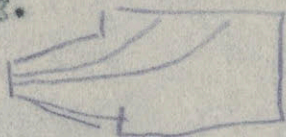
SAMPLING AND ASSAYING LEAD BULLION.
D. R. W. Laboratory.

A SAMPLING:

The lead pigs as received by the refinery are shaped thus:

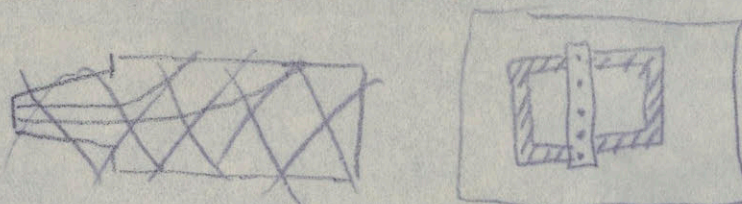


They are placed five in a row, face up, and samples are to be taken out of each bar by punches indicated in the diagram, taking No. 1 for the first bar, No. 2 for the second and so on. The punch is driven half way through the bar. This punch is made to take out the core $3/16$ " square and half the depth of the pig.



The bars are now turned over and punched the same as before, using the template of the lower face.

The cores are now ready to be cast into a sample bar, whose dimensions must be $9" \times 3" \times 5/8"$. The cores of a car load of pig lead is enough to make two such bars, and is cut down as follows: A graphite crucible big enough to hold all the cores is heated red hot in the furnace, taken out and the lead cores put in, which soon melt, stirred quickly with a graphite stick and cast into a double mould, dimensions $9" \times 3"$.



The bars are now sawed lengthwise and half of each put back into a red hot crucible again, stirred and cast into a bar which will be $9" \times 3" \times 5/8"$. This bar is sawed exactly in half, lengthwise, and one half sent to the seller and buyer.

The melting of the cores in a red hot crucible does away with the making of any dross, except what is actually contained in the cores, and heats the lead hot enough to pour so can get a compact sample bar. The bar must be made by pouring fast enough to fill the mould and making no over lap. All the contents of the crucible must be poured out, and if there is any dross, must be sampled separately and calculated.

If the lead cores are very dirty another method of sampling can be resorted to. Weigh the lead and melt under a covering of finely powdered charcoal, and when completely melted slow off the fine charcoal with air pressure (takes only half a minute) and pour the contents into the mould, making sure all the shot is emptied out. The fine carbon reduces all lead slag to oxide. The lead bar must be weighed and assays calculated to the weight of the cores.

B Assaying:

From one of the halves four assay pieces of one half assay ton each are to be cut at the points, and of the general dimensions indicated in

the diagram, the cuts to be parallel and to go through the entire bar.



These four pieces are pared down to exactly one-half assay ton, and are to be cupelled at as low a temperature as will give a lead free from lead and copper. The brightening is to be done at a rising, not a falling temperature. The average of the four beads obtained from the assay is to be the assay of the lot. Should one of the buttons be lost, it will not be advisable to take the average of the remaining three, or to run an assay to replace the one lost, but four new assay pieces, similar to the original ones, must be cut out and assayed, and the average of the four buttons obtained will determine the assay of the lot. If the lead bullion is unclean, the four assay pieces must be scorified with 40 gr. test lead and resulting buttons cupelled.

After weighing the buttons, the four are placed in test tubes and 10 cc of HNO 3.1/2 added, place in beaker of water and bring to a boil, continue boiling until solution is clear, decant off solution, wash once with distilled water, add 10 cc of HNO 3.5l. 42, and boil slowly for 80 minutes, again decant off and wash gold three times with distilled water, fill test tube with distilled water. Invert test tube into a porcelain crucible, allowing gold to settle down into crucible. When gold has all settled take out test tube very carefully so as not to disturb the gold, decant off water from crucible, and allow same to become thoroughly dry; then anneal crucible and gold by heating to a red heat. After same has cooled, carefully put ~~XXXXXXXX~~ gold on scale pan and weigh. Deduct weight of gold from weight of beads, and difference represents the silver.

For the determination of lead, saw the sample in many pieces to obtain an average of the bar, and mix the sawings together.

No. 1. Weigh out 6.831 gr. of the sawed sample into a 400 cc beaker and add 25 cc hot water, 5 cc tartaric acid (50 Sol) and 12 cc Conc. HNO 3. Set on a hot plate, and when dissolved fill beaker to about half full with boiling water, and add 5 cc dilute H 2 S O 4 (1-1). Boil for a minute decant into a 600 cc ~~inflammable~~ flask, and wash twice by decantation using about 75 cc boiling water each time. Set the beaker aside and render the solution in the flask alkaline with Na O H; add 35 cc Na 2 Solution (5 lbs. Na 2 in two gallons). Digest on the bath until the precipitate has settled about an hour, filter and wash without delay with boiling water (In this way it is possible to wash the precipitate well down into the apex of the filter so that when it is dried it can be removed very clearly and, therefore, will be no need of incinerating filter). Transfer the precipitate after drying to the beaker containing the PbSO 4 to which it belongs and add 5 c c conc HNO 3 and 5 cc conc H 2 SO 4. Set on a plate and ~~and~~ run to ~~XXXXXX~~ dense white fumes. Remove; let cool; add 75 cc cold water, boil for a minute, let stand for a few hours and filter into a tared gouch crucible using suction; bring the precipitate into the crucible and wash once with water containing 10 Hn SD4. Wash two or three times with ~~dilute XXXXXXXX~~ alcohol, and once with absolute alcohol. Dry on hot plates and heat finally over a ~~XXXXXX~~ Bunsen burner with wire gauge and thin sheet of asbestos paper, enterpose, cool in a dessicator and weigh each gram of Pb SO4 - 10% Pb.

No. 2 - Five gr. of the sawed bar is introduced into a 250 cc beaker and dissolved in 25 cc H 2 O 5 cc of 50% solution of tartaric acid 10 cc HNO 3 free from Sulphuric acid. This clear solution is diluted with hot water (50 cc) and precipitated by addition of 10 cc H 2 SO 4 diluted to 50 cc, boil a few minutes. The solution containing As. Sb. Ag. and a little lead is decanted on a 12-1/2 cm filters, the insoluble lead

sulphate is then stirred with 50 cc hot H₂O containing a little sulphuric acid (25 cc per liter) Let it settle a few minutes and again decant filter and so three times. The washings united are neutralized by Na OH and digested until clear with 25 cc Sodium Sulphide.

The lead sulphide is collected on the filter used for the filtration of the washings, washed with cold water till hardly acid reaction can be detected in the washings and finally washed with dilute alcohol and dried in the hot water oven.

The alkaline solution of As Sb is separated from the lead, copper and silver sulphides by filtration, the filter used being reinforced by a small cloth filter at the apex, and if As and Sb are to be determined they will be found in the filtrate. The sulphides collected in the filter are dried in a porcelain dish, burned at low temperature, treated by Nitric acid, Sulphuric acids and the small amount of lead sulphite thus obtained is collected in a small filter. The porcelain crucible having been tared, the filter containing that small amount of lead sulphite is gently heated till it chars, then the crucible is brought to a higher temperature on the Bunsen Burner till the combustion of the carbon is complete. When crucible has cooled, treat by few drops of dilute HNO₃, evaporate on hot plate, add one drop of sulphuric acid and evaporate to dryness. The main portion of the lead sulphate, which has dried in the meantime, is separated carefully from ~~the~~ its filter, either on glazed paper or in a well dried watch glass. The paper containing some lead sulphite is introduced in ~~xxxxxxx~~ the crucible and treated as was told for the small filter. An alternate way is burn both filters together and make only one acid treatment; but, of course, there is more ash and carbonate to be taken care of. At any rate we have now the crucible with a small percentage of lead sulphate, and in this crucible we introduce with the aid of ^{a small} the main portion of lead sulphate Calcine at a low heat for about 5 minutes, cool, weigh, and calculate as Pb.

The separation of As Sb in solution in the alkaline solution is made in the ordinary way, viz. acidulate filtrate, let sulphides separate, filter, treat sulphides of As and Sb according to what method preferred. Generally use the separation of As and Sb by digestion with ammonium carbonate.