Dross

CERRO DE PASCO CORPORATION - LA OROYA

Correspondencia Interdepartamental

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Materia: Copper Dross Treatment - Economics
- Recirculation vs. Separate Smelting -

Please refer to R.P. Koenig's cablegram of September 11 on the above subject. The basic information on dross-furnace operation versus the recirculation of copper dross through the blast furnaces was furnished by H.W. Higgs - see his memo of September 20, attached hereto.

The accumulated stock of copper dross is now being treated in the blast furnaces along with the current dross, and it is expected that the copper-dross stock will be consumed by the end of the year.

This economic analysis disregards the proposed construction of new blast furnaces. Reason: By treating copper dross in a separate smelting unit, new blast furnaces would not be essential to smelt the known lead ores & concentrates that will be available during the next ten years.

Also disregarded herein are the capital expenditures for the necessary expansion of mining, milling, sintering, refining, and coking facilities; expansion of these facilities, required in any case, would have no bearing on the warrantability of a dross-smelting furnace.

The justification of constructing additional by-product coke ovens, instead of producing bee-hive coke, was covered by H.W. Higgs in his memo of September 25.

(I regret the delay in submitting this analysis; I misjudged the amount of work and time necessary to complete the job.)

Summary of Findings

Since the economic study involved many factors, this detailed report is quite long. In consequence, a summary of the findings is advisable, viz.:

1) Based on 100% availability of custom lead ores & concentrates from the regions tributary to Oroya, and taking into account the known plans for increased production of lead concentrates, the lead-plant intake during the next ten years is not likely to exceed 190,000,000 pounds of lead per year. (See page 7)

2) This new-feed intake, available in 1958, would yield 180,000,000 pounds of refined lead (net recovery: 95%, approx.)

3) The corresponding quantity of lead bullion could be produced by the existing blast furnaces if the copper dross is eliminated from the blast-furnace feed. This is the considered opinion of H.W. Higgs and other members of the smelter staff. It is based on a blast-furnace capacity of 700 short tons of sinter per day (presupposing adequate production of good sinter). (See page 3)

4) Based on the availability of lead ores & concentrates and the limiting capacity of the blast furnaces, the potential intake from 1955 onward is estimated as follows:

	Mean Monthly Intake, Ores & Concxntrates	
	1955-1957 DST/mo.	1958 et seq. DST/mo.
Copper dross treated separately	13,050	15,500
Copper dross circulated	12,050	12,900
Difference	1,000	2,600

(See pages 4 to 7)

5) The comparative unit costs of lead smelting - dross circulation vs. separate treatment - would not vary to a marked degree.

	Mean Unit Cost of Lead Smelting	
	Per DST Ore-Con. 1955-1957	Per DST Ore-Con. 1958 et seq.
Copper dross treated separately	$16.04	$15.08
Copper dross circulated	$16.23	$15.85

(See pages 9 to 10)

6) In relation to the copper assay of lead ores & concentrates, and with separate treatment of copper dross, the unit smelting cost may be expressed as follows:

1955-'57 (mean divisor: 13,050 DST/mo.): $15.00 + $0.95x%Cu, per DST.
1958 (mean divisor: 15,500 DST/mo.): $14.18 + $0.95x%Cu, per DST.
1955-'58 (mean divisor: 13,660 DST/mo.): $14.76 + $0.95x%Cu, per DST.

These expressions include the cost of treating dross-furnace matte and speiss in the copper-plant converters. (See page11)

7) The potential increased throughput of metals made possible by a dross-furnace installation, as estimated for a 4-year period beginning with 1955, is as follows:

	Copper, Lbs.	Lead, Lbs.	Silver, Ozs.	Gold, Ozs.
1955	1,560,000	10,900,000	1,224,000	1,800
1956	1,560,000	10,900,000	1,224,000	1,800
1957	1,560,000	10,900,000	1,224,000	1,800
1958 (et seq.)	1,735,000	28,400,000	1,663,000	2,000
1955-1958	6,415,000	61,100,000	5,335,000	7,400

For the 1955-'57 period, the increased throughput would derive from the additional custom lead-copper concentrates; for 1958, the throughput would be further increased by additional Paragsha lead concentrate (plus some extra Matagente ore). (See page12)

8) A dross furnace would improve the recoveries of copper and silver from lead ores & concentrates. Interms of feed-assay units, the loss of copper would probably be 0.26 %Cu (versus 0.37 %Cu at present) and the loss of silver would be cut to 0.6 ozAg (versus 0.8 ozAg). For example, from a feed component assaying 1.30 %Cu and 10.0 ozAg, the percentage recoveries would be: copper, 80.0% (vs. 71.5%); silver, 94.0% (vs. 92.0%). The effect on the assays of the blast-furnace slag would be more striking, viz.: 0.25 %Cu (vs. 0.66 %Cu); 0.3 ozAg (vs. 0.8 ozAg). (See page 13)

9) Regarding the transference of metals to the copper plant in the form of matte, the comparative situation would be as follows:

	Annual Transference of Metals Lead Plant to Copper Converters		
	Copper, Lbs.	Lead, Lbs.	Silver, Ozs.
Blast-furnace matte & speiss	2,299,000	1,437,000	115,000
Dross-furnace matte & speiss	4,421,000	884,000	221,000
Net change with dross furnace	+2,122,000	-553,000	+106,000

(See page 14)

10) When the dross furnace is in operation, and we actively bid for custom lead-copper concentrates, the 'copper' terms of our present tariff will be revised. Compared with Hochschild's current tariff for lead-copper concentrates, our revised tariff terms would result in a substantially equal net liquidation. But on the basis of the custom shippers' net returns, the results in most cases would favor sale to CdeP because of the freight differential (freight to Oroya vs. freight to Callao) and other factors. (See page 15)

11) The additional intake made possible by the installation of a dross furnace would yield a contribution in excess of $2,000,000 over a 4-year period (1955-1958). After deducting "load" and income taxes, the net would be more than $1,000,000. (See page 20)

This outcome is based on lower metal prices than obtain at present (see page 16). Depending upon the actual metal prices and the actual operating costs during 1955-1958, the actual net after taxes may be somewhat less or somewhat more than $1,250,000.

In any case, the capital expenditure for a dross=furnace installation would be returned within 3 to 5 years after the furnace is in operation.

12) Item "F" (Foxtrot) in R.P. Koenig's cablegram of Sept. 11 - secondary drossing of copper by the addition of sulphur - cannot be properly evaluated at this time. Experimentation will be necessary to determine the disposition of the various elements - particularly the minor elements: tin and indium. However, certain aspects and benefits of secondary drossing with sulphur are discussed by D.A. Ricketts in his memo of Sept. 18 and by H.W. Higgs in his memo of Sept. 20. Both memos are attached hereto.

Lead Smelting Capacity

By treating the lead-plant copper dross in a separate furnace, instead of circulating the dross through the lead blast furnaces (current practice), the existing blast furnaces would have sufficient capacity to smelt 700 short tons of sinter per day (350 tons per furnace-day), or 21,000 short tons per month. Of course, this is predicated on adequate sintering facilities for the production of first-class sinter.

Currently, with only a small amount of lead-copper concentrate in the smelter intake, the normal monthly tonnage of copper dross circulated through the blast furnaces (i.e., new dross plus circulating load) is approximately equivalent to: T x 0.25 + 200, T being the monthly tonnage of lead ores & concentrates.

Note: This formula applies to continuous treatment of copper dross (i.e., no storage of dross).

The monthly tonnage of sinter is approximately equivalent to: T x 1.32 + 550. Consequently, the tonnage of sinter plus copper dross equals: (T x 1.32 + 550) + (T x 0.25 + 200) = T x 1.57 + 750.

(Note: If anyone concerned desires an explanation of the formulas given herein, this office will be pleased to submit the derivations.)

As pointed out in H.W. Higgs' memo of September 20 (attached), every ton of copper dross fed to the blast furnaces displaces a potential ton of sinter. With good sinter, and with sinter-plant bottlenecks eliminated, the present blast furnaces could therefore treat 21,000 short tons per month of combined sinter and copper dross. Thus, with a revamped sinter plant but with no additional copper in the lead-plant intake, the monthly capacity of the lead plant in terms of lead ores & concentrates would be: T = (21,000 - 750)/1.57 = 20,250/1.57 = 12,900 DST.

But with the copper dross treated in a separate furnace, T = (21,000 - 550)/1.32 = 20,450/1.32 = 15,500 DST per month. In other words, the elimination of copper dross from the blast-furnace feed would permit the smelting of an additional 2,600 tons per month of lead ores & concentrates (15,500 - 12,900 = 2,600). Moreover, this additional tonnage could include the custom lead-copper concentrates which, at present, we are forced to exclude from the intake.

A monthly intake of 15,500 DST of lead ores & concentrates having a mean grade of 50.0 %Pb would yield 176,700, 000 pounds of refined lead annually. Adding the lead recovered from copper ores & concentrates, the total refined lead production per year would be 180,000,000 pounds.

Lead-Plant Intake

The dross-furnace installation (in conjunction with a revamped sinter plant) could probably be completed by the end of 1954. The potentially available intake of new feed for the lead plant (lead ores & concentrates plus leady dusts from the copper circuit) is estimated at 158,000,000 pounds of lead for 1955 and 1956; for 1957, 173,000,000 pounds; for 1958 et seq., 190,000,000 pounds.

Inasmuch as the yearly intake of lead will not be constant from 1955 to 1958, the data pertaining to future operations (with separate treatment of copper dross) are based on a) the estimated mean annual intake for the years 1955, '56 & '57; b) the estimated intake for 1958. Note: The intake for 1958 is considered to be close to the limit, both in respect to availability and to blast-furnace capacity.

<u>For the 1955-'57 period</u>, the estimated mean annual intake of "new" lead would be 163 million pounds (i.e., one-third of 158 + 158 + 173 = 163). The composition of the intake would be as follows:

Mean Annual Intake, 1955-1957
(Separate treatment of copper dross)

Company & Leased:		DST	% Pb	Lead Content, Lbs.
Matagente ore	1/	10,000	18.5	3,700,000
Paragsha con.	2/	37,000	50.0	37,000,000
Casapalca con.	3/	16,000	60.0	19,200,000
San Cristobal & East Moro.con.	4/	5,000	60.0	6,000,000
West Moro. con.	5/	3,000	60.0	3,600,000
Total Co. & Leased		71,000	48.9	69,500,000

Mean Annual Intake, 1955-1957
(Separate treatment of copper dross)

		DST	% Pb	Lead Content, Lbs.
Total Co. & Leased		71,000	48.9	69,500,000
Custom:				
Atacocha con.	6/	21,000	65.0	27,300,000
Cercapuquio con.	7/	2,200	66.0	2,900,000
Colquijirca con.	8/	4,800	53.0	5,100,000
Huarón concentrate	9/	18,000	64.5	23,200,000
Pichita Caluga ore	10/	18,000	32.0	11,500,000
Yauli concentrate	11/	3,600	50.0	3,600,000
Other Current	12/	6,000	45.0	5,400,000
Additional lead-copper cons.	13/	12,000	45.5	10,900,000
Total Custom		85,600	52.5	89,900,000
Total Lead Ores & Concentrates		156,600	50.9	159,400,000
From Copper Ores, Roasted Dusts		6,000	30.0	3,600,000
Total New Feed to Lead Plant		162,600	50.1	163,000,000

With the addition of limerock plus the circulating load of lead-plant dusts (i.e., lead-plant dusts roasted with pyrite), the mean annual lead-bed mix for the 1955-'57 period would total about 218,000 DST and the grade of the bed mix would be 41 %Pb - close to the permissible maximum for sinter-plant feed.

1/ Matagente ore, containing 0.15 %Cu, 8.0 ozAg, 0.005 ozAu. In view of the anticipated increased intake of lead concentrates, the indicated tonnage of Matagente ore would consitute necessary siliceous flux to obtain a proper iron-to-insol ratio and to dilute the grade of the bed mix to the proper lead assay.

2/ Paragsha lead con., containing 0.50 %Cu, 25.0 ozAg, 0.015 ozAu. The mean annual tonnage for 1955-'57 is based on the following: 32,000 DST in 1955 and in 1956; 47,000 DST in 1947.

3/ Casapalca lead con., containing 1.75 %Cu, 56.0 ozAg, 0.005 ozAu. Concentrate tonnage based on milling of 25,000 tons per month of 4%Pb ore.

4/ San Cristobal & East Morococha lead con., containing 1.00 %Cu, 25.0 ozAg, 0.050 ozAu. Tonnage of concentrate based on milling 18,000 tons per month of mixed San Cristobal and East Morococha ores at the Mahr concentrator; average composite mill feed: 2.0 %Pb.

5/ West Morococha lead con., containing 1.50 %Cu, 25.0 ozAg, 0.005 ozAu. Tonnage of concentrate based on treating a total of 60,000 tons per year of 4%Pb ore in a series of milling campaigns at the Morococha concentrator.

6/ Atacocha lead con., estimated intake for 1953; 1.00 %Cu, 47.0 ozAg, 0.125 ozAu.

7/ Cercapuquio lead con., estimated intake for 1953; 0.05 %Cu, 4.0 ozAg.

8/ Colquijirca lead con., estimated intake for 1953; 3.00 %Cu, 73.0 ozAg, 0.005 ozAu.

9/ Huarón lead con., 100% of production; 1.50 %Cu, 58.0 ozAg, 0.125 ozAu.

10/ Pichita Caluga lead ore, estimated intake for 1953; 0.15 %Cu, 5.0 ozAg.

11/ Yauli lead con., estimated intake for 1953; 2.75 %Cu, 62.0 ozAg, 0.050 ozAu.

12/ "Other Current", same as "Other Purchased" estimated for 1953; 3.00 %Cu, 65.0 ozAg, 0.100 ozAu.

13/ Additional lead-copper concentrates; based on former receipts, viz.:

Custom lead-copper concentrates, additional. By the end of 1949, the Corporation had largely discontinued the purchase of lead-copper concentrates due to the lack of adequate facilities for smelting such material. The following table lists the additional custom lead-copper concentrates which presumably will be available for purchase. With certain exceptions (as noted below), the tonnages and grades are comparable to former receipts.

Additional Lead-Copper Concentrates, Custom

	DST Per Year	% Weight	Assays % Cu	% Pb	% Zn	oz Ag	oz Au
Rio Pallanga	4,200	35.00	6.0	47.0	7.0	110.0	0.075
Castrovirreyna	1,800	15.00	4.0	45.0	9.0	105.0	0.550
Obradovich	1,700	14.17	4.0	50.0	6.0	35.0	0.010
Proano, L.A.	1,200	10.00	7.5	50.0	6.0	35.0	0.140
Caudalosa Grande	1,300	10.83	16.0	25.0	7.0	220.0	0.140
Enano (Gubbins)	600	5.00	5.0	60.0	8.0	220.0	0.120
Other "Banco Minero" Lead-Copper Cons.	1,200	10.00	5.0	45.0	6.0	45.0	0.050
Total per year	12,000	100.00	6.5	45.5	7.0	102.0	0.150

Chungar concentrate is not included because we are advised that Sr. Mateo Galjuf plans to discontinue treatment of his lead-copper ore and, in its place, to mill a straight lead-zinc ore.

Rio Pallanga is producing closer to 4,500 tons per year than the 4,200 tons indicated, but we are now receiving token shipments of about 25 tons per month.

Since Neg. Min. Reynaldo Gubbins plans to build a concentrator, the tonnage of "Gubbins" concentrate will doubtless be greater than the 600 tons per year produced at the Banco Minero concentrator (Sacracancha).

The potential tonnage of "Other 'Banco Minero' Lead-Copper Cons." - i.e., from concentrators at Sacracancha, Huachocolpa and La Virreyna - may be somewhat greater than indicated; in the past, annual tonnages of miscellaneous types of lead-copper concentrates were quite variable.

For the 156,600 DST of lead ores & concentrates estimated as the mean annual intake for 1955-'57, the grade would be as follows:

% Cu	% Pb	oz Ag	oz Au
1.46	50.9	39.8	0.053

Intake for 1958. Except for Matagente ore and Paragsha lead concentrate, the estimated intake for 1958 would be composed of the same tonnages and grades of ores & concentrates as listed for 1955-'57. The changes in intake for 1958 would be as follows:

	DST Per Year (1958)	% Pb	Lead Content Lbs.
Matagente ore	12,400	15.0	3,700,000
Paragsha con.	64,000	50.0	64,000,000
Total	76,400	44.3	67,700,000

As compared with the mean annual intake for 1955-'57, the above totals represent increases of 29,400 DST ore-con. and 27,000,000 pounds of lead.

The tonnage of Paragsha lead concentrate corresponds to the milling of 2,000 tons per day of Cerro lead-zinc ore.

Owing to the increase in the tonnage of pyritic Paragsha lead concentrate in 1958, an increased tonnage of siliceous Matagente ore (but at a lower grade) will be required as flux to adjust the iron-to-insol ratio and lead grade of the lead-bed mix.

The estimated intake of new feed for the lead plant in 1958 is as follows:

	DST	% Pb	Lead Content, Lbs.
Lead ores & concentrates			
Mean annual intake 1955-'57	156,600	50.9	159,400,000
Increase in 1958	29,400	45.9	27,000,000
Ores & concentrates, 1958	186,000	50.1	186,400,000
Copper-plant dusts (roasted)	6,000	30.0	3,600,000
Total new feed for 1958	192,000	49.5	190,000,000

Of the total lead intake for 1958, 50.8% would derive from Corporation-mined ores; 47.3% from custom ores & concentrates; 1.9% from copper-plant dusts.

The 186,000-ton intake of lead ores & concentrates for 1958 would have a mean grade as follows:

% Cu	% Pb	oz Ag	oz Au
1.30	50.1	38.2	0.048

Lead Smelting Costs

For comparative purposes, the current unit cost of lead smelting (whether on a monthly or year-to-date basis) is inapplicable because the divisor is inapplicable. In comparing smelting costs (present dross treatment vs. proposed treatment), the ore & concentrate divisor to be used in relation to the present smelting procedure should, theoretically, be the potential maximum of 12,900 DST per month - see page 4, second paragraph. But since the 12,000 DST of additional lead-copper concentrates included in the mean annual intake for the 1955-'57 period could not be treated by the present smelting procedure, the annual tonnage of treatable ores & concentrates would be: 156,600 - 12,000 = 144,600, or 12,050 DST per month.

Under normal circumstances (no unusual operating expenses) certain components of the total cost of smelting are directly proportional to the tonnage of ore & concentrate treated. On a unit basis, such cost items may therefore be considered as substantially constant, viz.:

Feed preparation	$2.25 per DST ore-con.
Sintering	2.50 " " "
Operating supplies other than fuel & fluxes (present operating procedure)	1.00 " " "
Fuel & fluxes for ores & cons.	3.75 " " "
sub-total costs	$9.50 per DST ore-con.

Note: The year-to-date costs are distorted by unusual and heavy expenditures in the feed-preparation section (receiving, crushing, bedding & reclaiming) and in the sinter plant. Conversely, the costs for the month of August appear too low to represent average costs. The above figures are consequently higher than the corresponding figures for August, but lower than the year-to-date figures; they are thought to be reasonable approximations of the mean costs.

Since the indicated cost for fuel & fluxes may seem out of line, an explanation is in order. Because coke must be provided to smelt the circulating copper dross as well as the sinter, the effective divisor for the fuel & flux cost is the tonnage of ore & concentrate plus the tonnage of circulated copper dross. For August, the cost of fuel & flux was $50,226.73 and the combined tonnage of ore & concentrate and dross was 15,013 DST, giving a unit cost with this divisor of $3.34 per ton. (The corresponding year-to-date cost for fuel & flux is $4.22 per ton.) As a mean figure for the cost of fuel & flux, $3.75 is taken as applicable per ton of ore & concentrate treated or per ton of copper dross circulated.

Other lead smelting costs are more or less fixed and largely independent of the tonnage smelted. These costs may be taken as follows:

	Cost Per Month
Blast furnace operation:	
Labor & supervision	$13,000
Maintenance & repair	14,000
Cottrelling proportion, 60%*	10,000
Indirect operating expense	32,000
	$69,000

*Cottrelling cost proportion: 40% now charged to lead smelting and 60% to copper smelting; but in view of the future increase in lead-plant intake and decrease in copper-plant intake, it would be more realistic to charge 60% of the central cottrell costs to lead smelting and 40% to copper smelting.

The monthly fuel & flux cost for smelting the circulated copper dross equals $3.75 times the tonnage of circulated dross, or:
$\$3.75 \times (T \times 0.25 + 200) = \$0.9375 \times T + \$750.$

Under the present method of operation, the total cost of lead smelting per month would equal: $\$9.50 \times T + \$69,000 + \$0.9375 \times T + \$750 = \$10.4375 \times T + \$69,750$, and the mean unit cost therefore equals: <u>$10.44 + $69,750/T</u>, where <u>T</u> is the monthly tonnage of lead ores & concentrates.

Substituting in the unit-cost formula the limiting ore & concentrate tonnage for the present method of operation (i.e., 12,050 DST per month - see page 7), the result would be: $10.44 + $69,750/12,050 = 10.44 + 5.79 = <u>$16.23 per DST ore & concentrate</u> = lead smelting cost at the limiting throughput under the present operating procedure.

<u>Smelting costs, proposed treatment.</u> For the 1955-'57 period, of course, the mean component costs of lead smelting may be quite different from those that now obtain. But in establishing the warrantability of treating copper dross in a separate furnace, the component costs common to the present and proposed smelting procedures must be held constant. In respect to the separate treatment of copper dross, the costs would be as follows:

a) Sub-total smelting costs as above: $9.50 per DST ore-con. Note: The cost of supplies for the copper-dross furnace are included in the operating cost for this unit - see below.

b) Fixed costs as above: $69,000 per month. In his memo of Sept. 20, Higgs estimates that 20 men would be required to operate the copper-dross furnace, but that these men would replace 36 others now required to handle copper dross through the lead blast furnaces, thereby resulting in a net saving of $634 per month in labor cost. It is Higgs' opinion that no extra shift bosses need be engaged to supervise dross-furnace operations. To be conservative, however, it is thought advisable to consider the potential saving in over-all labor cost as being offset by an increase in supervision cost.

c) Exclusive of labor cost, Higgs estimates the probable operating cost of the dross-treatment unit at $10,707 per 1,110 tons of original copper dross, which is equivalent to $9.65 per DST of original copper dross.

d) In this cost summary, an amortization charge for the dross furnace installation is not included because the time required to pay for the installation is indicated in the final section of this memo.

The mean annual content of copper in the copper dross for the 1955-'57 period is theoretically estimated as follows:

	DST	% Cu	Copper Content, Lbs.
Blast furnace feed:			
Lead ores & concentrates	156,600	1.46	4,572,000
Roasted cottrell dusts	37,300	2.00	1,492,000
(Other feed components disregarded)	---	--	----
Total	193,900	1.56	6,064,000
Blast furnace slag	78,300*	0.25**	392,000
Lead-plant copper to dross furnace (theoretical)			5,672,000

* BF slag tonnage at 50% of ore & concentrate tonnage.
** BF slag copper assay as per H.W. Higgs' prognostication when treating copper dross in separate furnace (see HWH memo of Sept. 20).

At a dross assay of 14 %Cu, the tonnage of copper dross would be: 5,672,000/14 x 20 = 20,260 DST per year, or 1,688 DST per month. The cost of treating the copper dross (exclusive of labor & supervision) would therefore be: $9.65 x 1,688 = $16,300 per month.

As noted, the mean annual tonnage of ores & concentrates for 1955-'57 is estimated at 156,600 DST, or 13,050 DST per month. Consequently, combining the component costs given above, the mean monthly cost of lead smelting for the 3-year period would be: $9.50 x 13,050 + $69,000 + $16,300 = 123,975 + 69,000 + 16,300 = $209,275. The mean unit cost would therefore be: $209,275/13,050 = $16.04 per DST ore & concentrate.

This unit cost is $0.19 under the comparative unit cost for operations on an "as is" basis, but the total amount of the smelting cost per month would exceed the potential amount on an "as is" basis by $13,700, viz.:

1955-1957	Ore & Con. Divisor DST/mo.	Unit Cost Per DST	Smelting Cost Per Month
Copper dross treated separately	13,050	$16.04	$209,275
Copper dross circulated	12,050	16.23	195,575
Difference	1,000		$ 13,700

During the 1955-'57 period, the tonnage differential favoring separate treatment of copper dross would pertain solely to the additional intake of custom lead-copper concentrates. Over the 3-year period, the additional 36,000 tons of lead-copper concentrate would yield a fair contribution. But considering the source of the extra revenue (custom business), the net return for the first 3 years would probably be insufficient to recover the high capital cost of the dross-treatment installation.

Smelting costs for 1958. As indicated on page 7, the intake of lead ores & concentrates in 1958 would total 186,000 DST, or 15,500 DST per month. But if the copper dross were circulated through the blast furnaces, the tonnage of ores & concentrates would be limited to 12,900 DST per month (see page 4), and the unit smelting cost would be: $10.44 + $69,750/12,900 = 10.44 + 5.41 = <u>$15.85 per DST ore-con.</u>

For 1958, the content of copper in the dross is estimated as follows:

	DST	% Cu	Copper Content, Lbs.
Blast furnace feed:			
Lead ores & concentrates	186,000	1.30	4,836,000
Roasted cottrell dusts	43,200	2.00	1,728,000
(Other feed components disregarded)	---	--	----
Total	229,200	1.43	6,564,000
Blast furnace slag	93,000	0.25	465,000
Lead-plant copper to dross furnace (theoretical)			6,099,000

The tonnage of dross, at an assay of 14% Cu, would be: 6,099,000/14 x 20 = 21,780 DST per year, or 1,815 DST per month. The cost of treating the copper dross (exclusive of labor & supervision) would then be: $9.65 x 1,815 = $17,500 per month.

The total monthly smelting cost, with copper dross treated separately (but exclusive of amortization), is therefore equal to: $9.50 x 15,500 + $69,000 + $17,500 = 147,250 + 69,000 + 17,500 = $233,750. The unit cost: $233,750/15,500 = <u>$15.08 per DST ore-con.</u>

This unit cost is $0.77 under the comparative unit cost for retreatment of dross in the blast furnace. The comparative situation in respect to operations for 1958 is as follows:

1958	Ore & Con. Divisor DST/mo.	Unit Cost Per DST	Smelting Cost Per Month
Copper dross treated separately	15,500	$15.08	$233,750
Copper dross circulated	12,900	15.85	204,450
Difference	2,600		$ 29,300

Aside from the principal fact that a dross furnace will make it possible to smelt an extra 2,600 tons per month of lead concentrates (in 1958), it is worth noting that the extra tonnage can be treated with an increase of only $29,300 in smelting costs.

<u>Smelting cost in relation to copper content of feed.</u> With separate treatment of copper dross, the unit cost of lead smelting would be equivalent to: $9.50 x T + $69,000 + $9.65x(Tx%Cu + Tx0.40 + 1,000 - Tx0.125)/14 divided by <u>T</u> = $9.69 + $0.69 x %Cu + $69,690/T, in which <u>%Cu</u> is the copper assay of one or more feed components and <u>T</u> is the monthly divisor of lead ores & concentrates.

For the 1955-'57 period, with a mean monthly divisor (<u>T</u>) of 13,050 DST, the unit cost of lead smelting would be: $9.69 + $0.69 x %Cu + $5.34 = <u>$15.03 + $0.69 x %Cu</u> (1955-'57).

For 1958, with a mean monthly divisor of 15,500 DST, the unit cost of lead smelting would be: $9.69 + $0.69 x %Cu + $4.49 = <u>$14.18 + $0.69x%Cu.</u>

To obtain the total cost of smelting lead-copper material, it is necessary to add to the lead smelting costs the cost of converting the dross-furnace matte. (Note: The reported unit cost of converter operation per ton of matte cannot be used because the divisor includes the circulating load of matte.) The cost of converter operation is approximately $26 per ton of "new" copper in the intake to the copper plant. Consequently, per DST of lead-plant intake, the cost of converting the dross-furnace matte & speiss would be: \$0.26 x (%Cu - 0.125) = \$0.26 x %Cu - \$0.03.

Thus the smelting cost per ton of copper-bearing lead ore or concentrate would be as follows:

1955-'57 (mean divisor:13,050 DST/mo): \$15.03 + \$0.69x%Cu + \$0.26x%Cu - \$0.03 = \$15.00 + \$0.95x%Cu, per DST.

1958 (mean divisor:15.500 DST/mo): \$14.18 + \$0.69x%Cu + \$0.26x%Cu - \$0.03 = \$14.15 + \$0.95x%Cu, per DST.

1955-'58 (mean divisor:13,660 DST/mo): \$9.69 + \$0.69x%Cu + \$69,690/13,660 + \$0.26x%Cu - \$0.03 = \$14.76 + \$0.95x%Cu, per DST.

Costs Other Than Smelting

Since a subsequent section of this analysis deals with the contribution from the additional intake, it is necessary to establish all relevant costs.

Purchasing of custom concentrate - see page 17.

Mining costs for Cerro lead ores. In August, the total tonnage mined at Cerro was approximately 54,000 DST. By 1958, the mean monthly mining rate for all Cerro ores will exceed the rate for August. This may result in a lower unit cost for mining Cerro lead-zinc ore. To be conservative, however, the costs for August 1952 are considered to apply, viz.:

Mining cost per DST Paragsha lead con.	\$18.91
Mining cost per DST Matagente ore	\$ 2.04

Milling cost, Paragsha lead con. Including I.O.E., the reported year-to-date cost for milling is \$8.02 per DST lead concentrate produced; for August, \$9.23 per DST. Certain of the unit-cost components will decrease when the mill feed is doubled (in 1958). But to be conservative, the applicable milling cost is assumed to be \$8.50 per DST lead concentrate produced.

Freight cost, Cerro to Oroya. On Paragsha lead concentrate, the freight cost is taken at \$1.40 per DST concentrate; on Matagente ore, \$1.30 per DST.

Refining & shipping costs. Since the production of lead and silver will increase, the respective unit costs for refining may decrease somewhat. However, the effect of a greater divisor on the unit costs of refining is disregarded. Except in the case of silver, the current (reported) unit costs are used herein.

The reported unit cost for silver refining is considered unrealistic. Reason: Owing to accounting instructions issued by the New York Office in 1949, silver bears 100% of the cost of anode-residue smelting. In our opinion, the actual refining cost per ounce of silver produced is closer to 3.00¢ than the reported cost of 4.85¢. A silver refining cost of 3.00¢ per ounce is therefore used in computing contribution.

(Note: Bismuth is not taken into account in this study; the disposition and over-all recovery of bismuth will be affected, but to an unknown degree, by proposed changes in operating procedure. See memos by D.A. Ricketts and W.H. Higgs, attached hereto.)

	Copper ¢/lb. f.a.s. Callao	Lead ¢/lb. f.a.s. Callao	Silver ¢/oz. c.i.f. New York	Gold ¢/oz. c.i.f. Lima
Refining	1.250	0.450	3.000	40.000
Shipping	0.505	0.380	1.325*	10.000
R & S	1.755	0.830	4.325	50.000

* Freight & insurance on silver based on price of 75 cents per ounce.

The ex-Callao costs against copper and lead are taken as follows:

	Copper ¢/lb.	Lead ¢/lb.
Ocean freight to U.S.	0.603	0.603
Insurance, marine	0.057	0.034
U.S. import duty	---	1.063
Inland freight (U.S.)	0.240	0.240
Selling commission, 1% of quotation	0.250**	0.140**
Total	1.150	2.080

** See section on metal prices, page 16.

Increased Throughput of Metals

By treating the copper dross in a separate furnace, the throughput of metals in the lead plant would increase materially. As noted above, the increase in tonnage of concentrates for the 1955-'57 period would constitute the additional intake of custom lead-copper concentrates, the lead content of which would increase the annual throughput of lead by 10,900,000 pounds (see table: "Mean Annual Intake, 1955-'57", page 5).

In 1958, the increased intake made possible by separate treatment of copper dross would be 2,600 DST per month, consisting of 1,000 DST of additional custom lead-copper concentrates, 1,400 DST of Paragsha lead concentrate, and 200 DST of Matagente ore. The increased throughput of lead for 1958 would therefore be:

	DST Increase Per Month	DST Increase 1958	% Pb	1958 Increased Throughput Lead, Pounds
Custom lead-copper con.	1,000	12,000	45.5	10,900,000
Paragsha lead con.	1,400	16,800	50.0	16,800,000
Matagente ore	200	2,400	15.0	700,000
Total	2,600	31,200	45.5	28,400,000

Note: this increased throughput would also apply to 1959 et seq.

The estimated increase in the throughput of copper, lead, silver and gold, beginning with 1955, is as follows:

Year	Copper, Lbs.	Lead, Lbs.	Silver, Ozs.	Gold, Ozs.
1955	1,560,000	10,900,000	1,224,000	1,800
1956	1,560,000	10,900,000	1,224,000	1,800
1957	1,560,000	10,900,000	1,224,000	1,800
1958	1,735,000	28,400,000	1,663,000	2,000
1955-1958	6,415,000	61,100,000	5,335,000	7,400

Recoveries & Transference of Metals to Copper Plant

As pointed out in Higgs' memo of September 20, experience indicates that separate treatment of copper dross versus the present procedure would decrease the blast-furnace slag assays from 0.66 %Cu and 0.8 ozAg to 0.25 %Cu and 0.3 ozAg. Furthermore, it is Higgs' opinion (based on experience) that the dross furnace would yield a matte-speiss product having a copper-to-lead ratio of 5:1 as compared with the present low ratio of 1.6:1.

Copper recovery. Under "Lead-Plant Intake", it was estimated that the mean annual intake of new feed for 1955-'57 (lead ores & concentrates plus copper-plant roasted dusts) would be 162,600 DST assaying 1.48 %Cu. But in relation to current practice, the 12,000 DST of additional custom lead-copper concentrate (at 6.50 %Cu) could not be treated. Deducting the additional custom lead-copper concentrate, the net applicable to present practice would be 150,600 DST of new feed at 1.08 %Cu, or a mean annual new-feed copper content (for 1955-'57) of 3,253,000 pounds.

With copper dross circulated, the recovery of copper would therefore be determined as follows:

Copper Dross Circulated	Mean Annual Copper Content, Lbs. 1955-1957
New copper in lead-plant feed	3,253,000
Copper in blast-furnace slag: 144,600 x 0.50 x 0.66 x 20	954,000
Lead-plant copper in BF matte, etc.	2,299,000
Losses on re-treatment, etc., 7%	161,000
Net copper recovered	2,138,000

Resultant net recovery of copper: 100 x 2,138/3,253,000 = 65.7%

Copper loss (c) as assay units of new feed: (1.08 - c)/1.08 = 0.657; c = 0.37 %Cu.

With copper dross treated separately, the mean annual intake of new feed (1955-'57) would be 162,600 DST at 1.48 %Cu, equivalent to 4,813,000 pounds of copper, and the recovery would be:

Copper Dross Treated Separately	Mean Annual Copper Content, Lbs. 1955-1957
New copper in lead-plant feed	4,813,000
Copper in blast-furnace slag: 156,600 x 0.50 x 0.25 x 20	392,000
Lead-plant copper to dross furnace, etc.	4,421,000
Losses on re-treatment, etc., 10%	442,000
Net copper recovered	3,979,000

Resultant net recovery of copper: 100 x 3,979,000/4,813,000 = 82.7%

Copper loss (c) as assay units of new feed: (1.48 - c)/1.48 = 0.827; c = 0.26 %Cu.

Note: Copper losses, expressed in terms of assay units of new feed, would be the same if computed on the basis of the estimated 1958 intake.

Matte production & transference of metals. The production of matte from lead-plant feed would be substantially the same whether figured on the mean annual intake for 1955-'57 or on the intake for 1958. Using the data for 1955-'57, the theoretical production of matte would be as follows:

	DST Per Year	Assays % Cu	% Pb	ozAg
Copper Dross Circulated:				
Blast-furnace matte & speiss (2,299,000 lbs. copper available)	5,750	20.0	12.5	20.0
Copper Dross Treated Separately:				
Dross-furnace matte & speiss (4,421,000 lbs. copper available)	5,525	40.0	8.0	40.0

The theoretical quantities of metals transferred in matte & speiss from the lead plant to the copper converters would then be:

	Annual Transference of Metals Lead Plant to Copper Converters		
Alternative Products:	Copper, Lbs.	Lead, Lbs.	Silver, Ozs.
Blast-furnace matte & speiss	2,299,000	1,437,000	115,000
Dross-furnace matte & speiss	4,421,000	884,000	221,000
Net change with dross furnace	+2,122,000	-553,000	+106,000

Note: Although the respective quantities transferred annually are theoretical (and therefore somewhat higher than actual), the difference figures, or net change in transference, should be fairly accurate.

Lead recovery. In respect to the lead transferred to the copper converters via lead-plant matte & speiss, the ultimate recovery would be rather low. For normal lead concentrates, i.e., low in copper (under 2% Cu), the loss of lead in terms of feed-assay units may be taken at 2.5 %Pb. In the case of lead-copper concentrates, however, part of the lead reporting in the matte is lost in the copper circuit, and the lead losses in terms of feed-assay units would be higher. Omitting the arithmetic, the lead lost from lead-copper concentrates is found to be approximately as follows:

Lead-Copper Concentrates:	Lead Losses Feed-Assay Units
Copper dross circulated	3.5 %Pb
Copper dross treated separately	3.0 %Pb

Example: Of two concentrates, both of which assay 45.0 %Pb, one is low in copper and the other is a typical lead-copper product. The recovery of lead from the low-copper concentrate would be: 100 x (45.0 - 2.5)/45.0 = 94.5%; from the high-copper concentrate, and with copper dross circulated; 100 x (45.0 - 3.5)/45.0 = 92.2%; from the high-copper concentrate, but with copper dross treated separately: 100 x (45.0 - 3.0)/45.0 = 93.3%.

Silver recovery. In the case of silver, recoveries would be at high levels in both the lead and copper circuits. Consequently, it is not necessary to differentiate between low-copper and high-copper concentrates as in the case of lead. The recovery of silver from lead ores & concentrates is determined as follows:

(see next page)

		Mean Annual Silver Content, Ozs. 1955-1957
Copper Dross Circulated:		
New silver in lead-plant feed (150,600 DST at 34.9 ozAg)		5,254,400
Silver losses:		
Blast-furnace slag: 144,600x0.50x0.8 =	57,850	
Treatment of matte: 5,750 x 0.6 =	3,450	
Unaccounted-for: 1% of new feed =	52,500	113,800
Net silver recovered		5,140,600

Percentage recovery of silver: 100 x 5,140,600/5,254,400 = 97.8%

Silver loss (s) as assay units of new feed: (34.9 - s)/34.9 = 0.978; s = 0.8 oz/DST.

Copper Dross Treated Separately:		
New silver in lead-plant feed (162,600 DST at 39.8 ozAg)		6,478,400
Silver losses:		
Blast-furnace slag: 156,600x0.50x0.3 =	23,490	
Treatment of matte: 5,525 x 0.6 =	3,320	
Unaccounted-for: 1% of new feed =	64,780	91,590
Net silver recovered		6,386,810

Percentage recovery of silver: 100 x 6,386,810/6,478,400 = 98.6%

Silver loss (s) as assy units of new feed: (39.8 - s)/39.8 = 0.986; s = 0.6 oz/DST.

Custom Lead-Copper Concentrates - Competitive Position

At the present time, purchase of lead-copper concentrates is not warranted because increased intake of copper in the lead plant would increase the circulating load of copper dross through the blast furnaces, thereby decreasing the sinter throughput.

CdeP smelting tariff. By treating the copper dross in a separate furnace, we could again enter the market for lead=copper concentrates. Our present smelting tariff could remain in force except in respect to the terms for copper. It is suggested that we pay for copper on the following basis: Up to 13.0 %Cu, deduct 1.3 units from the assay and pay for 100% of the remainder; over 13.0 %Cu, pay for 90% of content. The R&D charge on copper would be revised to a flat 6¢ per pound liquidated. In addition to the treatment charge as specified in our current tariff (i.e., $24.50 per DST less $0.10 per unit of lead), it is suggested that we impose a surcharge of $0.25 per unit of copper.

Competitive tariff (Hochschild). The Lima ore buyers' terms vary somewhat, but their respective net liquidations are substantially the same. Our competitors may try to establish a competitive edge when we enter the market for lead-copper concentrates. But in the absence of premium prices, Hochschild et al do not have much leeway for competition.

At the present time, our competitors terms result in a net return to the shipper which is comparable to the amount he would receive under our current tariff terms. Example: At current metal prices, and taking into account the cost of trucking concentrate to Callao, Sin. Min. Rio Pallanga's net return from sale to Hochschild is computed at $182.71 per DST; but our present tariff terms would give Rio Pallanga a net of $182.76 per DST.

Hochschild's current terms for lead-copper concentrates are as follows:

Payment:

Copper: Deduct 1.3 units from assay and pay for 100% of remainder at the E&MJ quotation for foreign copper less 5.00¢/lb.

Lead: Deduct 1.5 units from assay and pay for 95% of remainder at the E&MJ quotation for domestic lead (New York) less 2.80¢/lb.

Silver (over 100 oz/DST): Pay for 96% at the E&MJ quotation for foreign silver less 2.00¢/oz. (Not over 100 oz/DST, pay for 95%.)

Gold: Pay for 96.75% at $34.9125/oz.

"Merma": 2%.

"Maquila" (treatment & export charges) per DST delivered at Callao, equivalent to: $30.25 - $0.25 x (%Pb - 40).

Penalties:

Zinc:	10% free,	excess at 35¢	per unit.
Arsenic:	1% free	" " 50¢	" "
Antimony:	1% free	" " 50¢	" "
Bismuth:	0.03% free	" " 50¢	per pound.

Sacks furnished free (except for freight charges).

(Note: All tariffs except ours impose a high penalty on bismuth.)

Future metal prices. To avoid overstating the increased revenue which will derive from the future increased intake of lead and lead-copper concentrates, the assumed average metal prices for the 1955-1958 period should be lower than current prices. But the forecasts of long-term prices used for ore inventories (i.e., 18.5¢ copper, 12¢ lead, 70¢ silver) seem too low as 4-year averages for the period indicated. For the purpose of this memo, the mean E&MJ quotations for 1955-1958 are assumed to be as follows: copper (foreign), 25.00¢/lb.; lead (domestic), 14.00¢/lb.; silver (foreign), 75.00¢/oz.; gold, $34.9125/oz.

In relation to the assumed quotations for copper and lead, the corresponding f.a.s.-Callao prices would be: copper, 23.850¢/lb.; lead, 11.920 ¢/lb.

Based on the foregoing E&MJ quotations, our net metal prices for custom purchases would be as follows: copper, 18.885¢(25.000 - 6.115); lead, 11.272¢/lb. (14.000 - 2.728); silver, 70.275¢/oz. (75.000 - 4.725); gold, $33.4125 (34.9125 - 1.5000).

Custom shippers' comparative returns - sale to Hochschild versus sale to CdeP. Instead of computing the comparative returns for each individual lead-copper concentrate, it will suffice here to apply the Hochschild tariff and the proposed CdeP tariff to the weighted composite of the additional lead-copper concentrates on which we shall bid in 1955 (see table: "Additional Lead-Copper Concentrates", page 6), viz.: 1,000 DST per month of lead-copper concentrate having a mean grade of 6.50% Cu, 45.5 %Pb, 7.0 %Zn, 102.0 ozAg, 0.150 ozAu.

In relation to the assumed mean prices of metals for 1955-'58 (see previous page), the shippers' comparative returns would be as follows:

	Hochschild (current tariff)	CdeP (current tariff)	CdeP (proposed tariff)
Payment per DST concentrate:			
Copper	$ 20.800	$ 14.820	$ 19.640
Lead	93.632	89.049	89.049
Silver	71.482	65.946	65.946
Gold	5.067	4.511	4.511
Total payment	$190.981	$174.326	$179.146
Deductions per DST concentrate dlvd.:			
Merma, 2% of payment	$ 3.820	---	---
Treatment charge	28.875	$ 19.950	$ 19.950
Treatment surcharge (for copper)	---	---	1.625
Penalties: nil (assumed)	---	---	---
Total deductions	$ 32.695	$ 19.950	$ 21.575
Liquidation before 4% levy	$158.286	$154.376	$157.571
4% levy, law #11357	6.331	6.175	6.303
Net liquidation	$151.955	$148.201	$151.268
Freight differential, assumed average	4.000	---	---
Shippers' comparative returns per DST dlvd.	$147.955	$148.201	$151.268

Under the terms of our proposed tariff (increased payment for copper), a competitive advantage of $3.30 per DST over Hochschild's current terms should be sufficient to get the business.

Even if the comparative returns were at a stand-off, the general advantages in selling to CdeP - prompt settlement, shipment in bulk, mining-supply privileges, technical assistance - would, in most cases, tip the scales in our favor.

Return on Investment

The justification for a copper-dross furnace rests on the amount of additional revenue to be derived from the installation.

Additional intake. As indicated on page 12, in the section on increased throughput, the additional intake for the 1955-1958 period - i.e., additional intake made possible by the installation of a dross furnace - would constitute the following:

	Additional Intake DST Per Month		DST 1955-'58	Assays % Cu	% Pb	oz Ag	oz Au
Custom lead-copper con.	1,000	(1955-'58)	48,000	6.50	45.5	102.0	0.150
Paragsha lead con.	1,400	(1958)	16,800	0.50	50.0	25.0	0.015
Matagente lead ore.	200	(1958)	2,400	0.15	15.0	8.0	0.005
Total additional intake	2,600	(1958)	67,200	4.77	45.5	79.4	0.111

Value of production. Based on the factors established in the foregoing sections of this memo, the value of production in respect to the components of the additional intake would be as follows:

Custom lead-copper concentrates:			Per DST
Copper (Callao):	(6.5 - 0.26)x20x	0.23850 =	$ 29.765
Lead (Callao) :	(45.5 - 3.0) x20x	0.11920 =	101.320
Silver (New York):	(102.0 - 0.6) x	0.75000 =	76.050
Gold (Lima):	(0.150 - 0.002) x	34.91250 =	5.167
Vale of production per DST custom con.			$212.302

Value of production for 48,000 DST custom con. (1955-1958): $10,190,500

Paragsha lead concentrate:

		Per DST
Copper (Callao):	(0.50 - 0.26) x20x 0.23850 =	$ 1.145
Lead (Callao):	(50.0 - 2.5) x20x 0.11920 =	113.240
Silver (New York):	(25.0 - 0.6) x 0.75000 =	18.300
Gold (Lima):	(0.015 - 0.002) x 34.91250 =	0.454
Value of production per DST Paragsha con.		$133.139

Value of production for 16,800 DST Paragsha con. (1958): 2,236,700

Matagente ore:

		Per DST
Copper (Callao): nil		
Lead (Callao):	(15.0 - 2.5) x20x 0.11920 =	$ 29.800
Silver (New York):	(8.0 - 0.6) x 0.75000 =	5.550
Gold (Lima):	(0.005 - 0.002) x 34.91250 =	0.105
Value of production per DST Matagente ore		$ 35.455

Value of production for 2,400 DST Matagente ore (1958): 85,000

Total value of production from additional intake for 1955-1958: $12,512,200

Costs against additional intake. The allocable costs against the respective components would be as follows:

Custom lead-copper concentrates:

		Per DST
Purchasing, dlvd. Oroya (before 4% levy)	=	$157.268
Smelting: $14.761 + $6.175	=	20.936
sub-total costs		$178.204
Refining & shipping:		
Copper (Callao): 124.8 lbs. x 0.01755	=	2.190
Lead (Callao): 850.0 lbs. x 0.00830	=	7.055
Silver (New York): 101.4 ozs. x 0.04325	=	4.386
Gold (Lima): 0.148 ozs. x 0.50000	=	0.074
Allocable costs per DST custom concentrate		$191.909

Allocable costs on 48,000 DST custom con. (1955-1958): $ 9,211,600

Paragsha lead concentrate:

		Per DST
Mining	=	$ 18.910
Milling	=	8.500
Freight to Oroya	=	1.400
Smelting: $14.150 + $0.475	=	14.625
sub-total costs		$ 43.435
Refining & shipping:		
Copper (Callao): 4.8 lbs. x 0.01755	=	0.084
Lead (Callao): 950.0 lbs. x 0.00830	=	7.885
Silver (New York): 24.4 ozs. x 0.04325	=	1.055
Gold (Lima): 0.013 ozs. x 0.50000	=	0.006
Allocable costs per DST Paragsha con.		$ 52.465

Allocable costs on 16,800 DST Paragsha lead con. (1958): 881,400

Matagente ore:

(see next page)

Carried forward:

Allocable costs on 48,000 DST custom con. (1955-1958):	$ 9,211,600
Allocable costs on 16,800 DST Paragsha lead con. (1958):	881,400

Matagente lead ore:			Per DST	
Mining		=	$ 2.040	
Freight to Oroya		=	1.300	
Smelting: $14.150 + $0.143		=	14.293*	
sub-total costs			$ 17.633	
Refining & shipping:				
Copper (Callao): nil				
Lead (Callao):	250.0 lbs. x 0.00830	=	2.075	
Silver (New York):	7.4 ozs. x 0.04325	=	0.320	
Gold (Lima):	0.003 ozs. x 0.50000	=	0.002	
Allocable costs per DST Matagente lead ore			$ 20.030	
Allocable costs on 2,400 DST Matagente lead ore (1958):				48,000

Total allocable costs on additional intake for 1955-1958:	$10,141,000

*Matagente ore, smelting cost: Owing to the increased tonnage of pyritic Paragsha lead concentrate, the Matagente ore would constitute necessary siliceous flux for the Paragsha concentrate; thus the Matagente ore would be charged with a normal smelting cost (i.e., no extra pyrite required; hence no extra smelting cost).

Contribution from additional intake.

Custom lead-copper concentrates:		Per DST	
Value of production	=	$212.302	
Allocable costs -	=	191.909	
Contribution per DST custom con.		$ 20.393	
Contribution on 48,000 DST custom con. (1955-1958):			$ 978,900

Paragsha lead concentrate:		Per DST	
Value of production	=	$133.139	
Allocable costs	=	52.465	
Contribution per DST Paragsga con.		$ 80.674	
Contribution on 16,800 DST Paragsha lead con. (1958):			1,355,300

Matagente lead ore:		Per DST	
Value of production	=	$ 35.455	
Allocable costs	=	20.030	
Contribution per DST Matagente ore		$ 15.425	
Contribution on 2,400 DST Matagente lead ore (1958):			37,000

Total contribution from additional intake for 1955-1958:	$ 2,371,200

Gross margin & net return on additional intake for 1955-1958.

"Lead" computation:

Total costs on additional intake:	$10,141,000
Purchasing cost of custom lead-copper concentrates: $157.268 x 48,000 DST =	7,548,900
Allocable costs subject to lead factor	$ 2,592,100
Lead, at 10%:	$ 259,200

Note: Since the total amount of operating costs for all mining, milling, smelting and refining activities will increase with the increased production of lead and zinc, the lead percentage for the 1955-1958 period should be less that at present. Hence the 10% lead factor.

Total contribution from additional intake, 1955-1958:	$ 2,371,200
Lead	259,200
Gross margin from additional intake:	$ 2,112,000
Income taxes etc., at 40% of gross margin:	844,800
Estimated net return during 1955-1958, excluding amortization of dross-furnace installation:	$ 1,267,200

Depending upon the actual metal prices and the actual operating costs during 1955-1958, the actual net return within the 4-year period may be somewhat more or somewhat less than the estimated net return.

Amortization charges will depend upon the type of dross furnace used. In any case, it is evident that the capital expenditure for the dross-furnace installation will be returned within 3 to 5 years after the furnace is in operation.

Theo. R.W.

cc: AHE (5)
RPK (10) New York
RPK Lima
WCS N.Y.
GR
JWH
ILB
JMM
HWH
DAR
RRvE

CERRO DE PASCO CORPORATION - LA OROYA

Correspondencia Interdepartamental

Fecha: Sept. 18, 1952

A : T. R. Wright

De : D. A. Ricketts

Materia: COPPER DROSSING OF LEAD BULLION
Possible Effects on Refineries

Drossing of lead bullion, with the method in use, results in a bullion running from .05 to .15 % Cu with .06 to .07% Cu as normal.

Cold drossing with addition of sulphur should result in a bullion with max. copper not over .01 %.

Lead Refinery

Through the many years of lead refining here it has been noted that if the bullion consistently carries over .08 to .09 % Cu a spotty or patchy deposit appears on the cathode which is both loose and quite rough. When this takes place efficiency drops through direct shorts and lead is lost from the cathode to the bottom of the cell. The condition was never allowed to continue for extended time so further detrimental effects are not known.

The results to be expected if the bullion were to run .01 % Cu or under are not known but we would assume they would be beneficial to refining and at the same time hard to evaluate.

Anode Residue Plant

The bulk of the copper intake, in this section follows the bismuth through to the kettle stage where it is drossed off. This dross is then treated in a reverb, with pyrite, and copper eliminated as a matte for return to the smelter copper circuit.

Matte shipped from the plant during the first seven months of this year amounted to 208.191 dry s.tons and carried the following values:

Cu	lbs.	42,362
Pb	"	96,800
Bi	"	38,340
Ag	ozs.	246,578.8
Au	"	180.46

Intake of copper for the same period was 14,198 lbs. with copper slimes and 41,181 lbs. with lead slimes for a total of 55,379 lbs. On this basis for each pound of copper intake the drag out in matte for smelter treatment is:

Pb per lb. Cu			1.75 lbs.	
Bi	"	"	"	.69 "
Ag	"	"	"	4.45 ozs
Au	"	"	"	.0033 "

Assuming copper in bullion to have been .01% instead of .06% then during this seven months period the copper intake from lead slimes would have been 6864 lbs. or a reduction of 34,317 lbs. which in turn would have increased direct metal production by:

Bi	lbs.	23,679
Ag	ozs.	152,711
Au	"	113.2

The copper in following the bismuth is removed, in the greater part, fromthe silver during the cupel operation. Some improvement would be forthcoming with a lighter copper load at this point and would probably show as more capacity due to less time required for copper removal. This in turn should show some saving on furnace linings, fuel oil and nitre.

On kettle operations the copper dross load is now 50 to 60% of feed from bismuth reduction reverb. This causes a heavy recirculation load in the plant and a tie-up of bismuth while retreating. Reducing copper intake would definitely cut down this load and allow for more direct production of bismuth, less work on kettles & less operation of reduction reverb. Recoveries would be improved.

General

In an overall discussion with Higgs, he feels that our present matte production might better be treated through the lead circuit. I agree in that this would cut any losses on bismuth which would be present between the copper and the lead circuits. The shorter tie-up on Ag and Au is apparent.

Experimental work should be carried out as regards copper removal before tin drossing. On the face of it this seems possible because of the low temperature and if true would eliminate the time and costs required for cooling and reheating of bullion if tin is removed before final copper by sulphur.

D A Ricketts

D. A. Ricketts

cc: AHE - 2
JWH
TRW - 12
ILB
JMM
HWH
DAR

CERRO DE PASCO CORPORATION

Correspondencia Interdepartamental

Fecha: September 20, 1952

To: J. M. Mortimer

From: H. W. Higgs

Subject: DROSS FURNACE JUSTIFICATION
(Cable R.P.K. to A.H.E. Sept.11, 1952)

The following information describes the metallurgical improvements to be expected following the installation of dross smelting facilities. All calculations are based on performance obtained during the first seven months of 1952 as compared with probable performance with an operating dross furnace assuming the same intakes of ores, concentrates and dusts. The data is arranged in the same order as the questions in the above mentioned cable.

1.- Alfa Intake of copper into the lead plant from all sources amounted to 2,338,376 lbs. during the seven month period. Based on an assay of 18 % Cu and not accounting for copper losses this is equivalent to a production of 926 tons of dross per month, yet while accurate figures are not available, at least 3000 tons of dross were treated per month in order to matte the copper and circulating load of ever 200 % is indicated. It is estimated that this 3000 tons per month could easily be replaced by additional sinter plant production if dross smelting facilities were available No additional cost would result at the Blast Furnace since this tonnage is presently being handled and fuel charged against it. Assuming no other improvements to arise while removing the dross and pyrite (source of sulphur for matte formation) from the charge an additional capacity of 28,800,000 lbs. of lead per year would be obtained at the Blast Furnace with a 40 % lead sinter charge.

Of the 2,338,376 lbs. of copper entering the lead circuit it is estimated that 2,182,356 lbs. would be available for treatment in the dross furnace the balance being lost in the Blast Furnace slag. This copper would make 1,110 tons per month of dross at 14 % copper available for treatment in a dross furnace and, assuming a 50 % recirculation of dross from the dross furnace, a total of 1,670 tons would be actually treated.

On the basis of treating 1,670 tons per month the following direct costs for dross furnace operation have been assumed.

Costs per Month

Soda Ash 6 % usage at $ 65/T 1670 x .06 x 65	=	$ 6,513
Coal 1.5 % usage at $ 10/T 1670 x .015 x 10	=	250

Fuel at 19,300 BTU/lb. & $ 18.00/T usage of 2,500,000 BTU/T. Dross 108 T at $ 18.00/T.	=	1,944
Arch repairs 6000 brick every six months Brick at $ 500/1000		500
Other repairs, Bodega charges, etc	=	1,500
Total $		10,707

It is also estimated that twenty men per day will be required to operate this furnace. Four per shift for direct operation tapping, charging etc. for a total of 12, and 8 for loading and handling matte, and repairs. Against this we believe that the present blast furnace and drossing crews can be reduced by 36 men per day due to reduction in men needed to charge dross to furnaces and the fact that furnace operations should improve greatly with the removal of dross from the charge.

Feed floor (charging dross) 4 men/shift	12 men
Operating floor (tapping/settlers etc.) 6 men	18 men
Dross drum crew 2 men/shift	6 men
Total	36

Therefore the net result is the elimination of 16 men per day with a cost of $ 634 per month at $ 1.32/man day.

2.- Bravo Actual receipts of copper for the seven month period were 2,338,376 lbs. Slag losses amounted to 409,286 lbs. with copper in the slag (31,204 T.) assaying 0.66 %. Copper losses would be reduced to 0.25 % with removal of dross and pyrite from the furnace charges resulting in slag losses of 156,020 lbs. A direct saving of 36,181 lbs. of copper per month is indicated.

During the same seven month period 25,083 oz were lost in the slag. Again experince indicates that silver losses could be reduced to .30 oz/T. resulting in a saving of 2,469 oz per month.

3.- Coca Matte transfered to the copper plant during the period carried lead to the converters at a ratio of 1 lead to 1.6 copper. Since 2,338,376-409,286 (slag loss)= 1,929,090 lbs. of copper were available for matting it is indicated that 1,205,680 lbs of lead were transfered. Since a ratio of 1 lead to 5 copper is reasonable for products transfered from the dross furnace, and since 2,338,376 -156,020 (assumed copper loss) = 2,182,356 lbs. of copper would be available, a transfer of 436,471 lbs. of lead is indicated.

It is therefore estimated that the reduction in lead transfered to the converters would be 109,900 lbs/month. The retreatment of this lead is expensive and again increases the recirculating load in the plant resulting in less new tonnage. 4 Delta see # Alfa Above.

5. Fox-trot. The process of decopperizing bullion with sulphur is complicated by the presence of tin in the bullion, and the desire to recover this tin.

(a) One suggested procedure is to make a preliminary copper drossing, followed by reheating, and tin drossing followed by cooling, cold copper drossing and sulphur drossing. This process involves one extra heating of all the lead and would require additional kettle capacity.

(b) The possibility exists, since tin drossing requires a high temperature that tin could be drossed following our present procedure with the insertion of the sulphur drossing procedure after the regular cold copper drossing. This method would be satisfactory if the tin did not come up with the copper dross resulting from the addition of sulphur, and would require less man power, fuel and capital expenditure. At present there is a large circulating load of tin in the dross indicating that most of the tin will enter the dross furnace in the cycle of opertions irregardless of this sequence of drossing. Some possibility exists that this tin could enter the speiss and be lost from the lead circuit, and this fact should be considered as a possible financial loss when operating the dross furnace. However the consensus of opinion is that it will remain with the bullion and return to the kettles

It should be possible to determine the proper sequence of dressing and the tin balance to be expected in the dross furnace by small scale research and I strongly recommend that we do this before deciding on a final procedure.

CC: AHE (2)
JWH
TRW ✓(12)
ILB
DAR
HWH

H. W. Higgs

H. W. Higgs

M. L. Addicks

CERRO DE PASCO CORPORATION - LA OROYA

Correspondencia Interdepartamental

Fecha: November 6, 1952

A: A.H.Engelhardt, Manager of Operations

De: J.W.Hanley, Assistant Manager of Operations

Materia: LEAD PROGRAM - DROSS REVERBS

While in the United States in 1949, I had the occasion to see Mr. E.P.Fleming, Metallurgist for the American Smelting & Refining Company, who has done considerable work with the dross reverb. Now that we are seriously considering a dross reverb operation, at the Oroya Smelter, the information given me by Mr. Fleming at that time may be of some interest to our operating and engineering departments. Attached to this memorandum you will find quite a complete list of specifications on the dross reverb furnaces at Chihuahua, Mexico and El Paso, Texas. These specifications give both furnace construction data and operating data as to fuel, draft, temperature, gas analyses, and capacity.

In discussing dross reverberatory furnaces with Mr. Fleming he estimated that the cost of the El Paso furnace in the United States, in 1949, would run about $35,000 for the furnace proper, not including building, cranes, flue connections, etc. From this figure a furnace of the El Paso type, erected in Oroya in 1949, would have cost approximately $46,000, allowing 30% for freight, duty, consular fees, etc. A furnace of this type erected today in Oroya would probably cost in the neighborhood of $60,000.

Also attached find a drawing of the dross reverb at the Shelby Plant of the American Smelting & Refining Company. While there is no data or specifications available for this furnace the drawing may be of general interest.

CERRO DE PASCO CORP.
MANAGER of OPERATIONS
NOV 7 1952

To	For Action	For Informat'n	Date
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J. W. Hanley

JWH/wgs.

Encl. 2 (Incl. 1 Drwg.)

cc: AHE (6 - with 1 Encl. to each)
GR - 1 Encl.
ILB - 1 Encl.
JMM - 1 Encl.
HWH - 1 Encl.
FILE (JWH)

Original signed by
A. H. Engelhardt

(RP 1-7-52)

DROSS REVERBS

		Chihuahua	El Paso
Arch:	Thickness	13-1/2"	13-1/2"
	Brand & Manuf.	Arco - G.R.Co.	Arco - G.R.Co.
	Material	70% Al_2O_3	70% Al_2O_3
	Mortar	Sillimanite	Sillimanite
	Life	150 days	463 days
Side Walls:	Above Bath-Thickness	Inside 9" Brick Water Jacket	13-1/2"
	Brand & Manuf.	Arco G.R. Co.	Alumex - Mexico, Mo.
	Material	70% Al_2O_3	70% Al_2O_3
	Mortar	Sillimanite	Sillimanite
	Life	150 days	463 days
	At Bath Line-Thickness	Water Jacket+9"Mag.	Water Jacket + 9" Mag.
	Material	86% MgO H&W	86% MgO H&W
	Mortar	Q. Chrome	Grefco
	Life	300 days	463 days
	Below Bath Line-Thick.	18"	18"
	Brand & Manuf.	Hard Burnt MgO H&W	Hard Burnt MgO H&W - G.R. Co.
	Material	86% MgO	86% MgO
	Mortar	Q Chrome	Grefco Thermolith at Bath
	Life	Incomplete	3 - 4 years
Bottom:	Double or Single Thickness	18" Double	18" Double
	Brand & Manuf.	St. Louis-G.R.Co.	Fire Brick-Pueblo S-1
	Material	Fire Brick	Fire Brick
	Mortar	Ojihaya	Hiloset
	Life	Incomplete	3 - 4 years
Uptake:	Thickness	9"	9"
	Brand & Manuf.	Arco-G.R.Co.	ARCo-G.R.Co
	Material	70% Al_2O_3	70% Al_2O_3
	Mortar	Saireset	Sillimanite
	Life	150 days	463 days
Furnace Measurements	Length - Inside	24'0"	27'7"
	Width - Inside	9'6"	8'2"
	Height at Drop Hole	6'6"	(6'2" (7'1"
	Height between Bath and Arch	2'8"	(2'10" (3'9"
	Area of Furnace	228 sq. ft.	225 sq. ft.
	Area Cross Section Combustion Zone	25.3 sq. ft.	21.8 sq. ft.
	Area of Exit of Furnace	14.5 sq. ft.	16.5 sq. ft.
	Depth of Bath	46"	45"

DROSS REVERBS

Combustion Data:

Type of Fuel	Oil	Gas
Type of Burner	Low Pressure D.F.C.	Force Draft
Number of Burners	2	2
Temperature of Fuel	145° F	-
Amt. Fuel burnt. Hr.	50 gal.	11,000 C.F.
Draft in Furnace during smelting	.06-.08	.05-.06
Height of burner above bath	2"	12-3/8"
Inclination of burner to bath	7°	7°
B.T.U. Value of Fuel	14931	1090
Pressure at Burner	44 ozs.	10.0

Gas Analysis:

O_2	1.2	2.5
CO	-	0
CO_2	12.8	10.0

Temperature:

Combustion Zone	2700° F	2300° F
Bath	2500° F	2080° F
Exit Gases	2200° F	1020° F

Capacity:

Dross Approx.	100 T.P.D.	120 T.P.D.

Duplicate enclosure-not received

L. Oddiehs.

CERRO DE PASCO CORPORATION - LA OROYA

Correspondencia Interdepartamental

Fecha: November 3, 1952.

A: A.H. Engelhardt, Manager of Operations

De: T.R. Wright, Assistant to Manager

Materia: Separate Smelting of Copper Dross - Net Returns
Ref.: Cable of Oct. 29, Koenig to Engelhardt.

In my memo of October 21 on the above subject, the net-return calculations did not take into account the potential revenue that would derive from the increases in recoveries of copper, lead and silver from the regular intake of copper- and silver-bearing lead concentrates. This revenue increment was disregarded for two reasons: a) the cabled directive of October 16 was misinterpreted; b) the misinterpretation conformed to my view that economic analyses should be conservative - particularly so in the case of proposed operations for which factual data are lacking. Consequently, the net-return calculations (Oct. 21) were limited to the additional throughput of custom ores & concentrates that would obtain with separate dross-smelting facilities.

By disregarding the revenue increment in respect to the regular intake, the estimated net returns from a dross-furnace installation are protected by an economic safety factor. Since the safety factor was fairly large in this instance, the net returns reported in the memo of October 21 are admittedly on the low side.

In his cablegram of October 29, R.P. Koenig stresses the point that full credit should be given for the potential increases in recoveries as applied to the regular intake. In line with this viewpoint, the net returns are re-amended herewith.

Increases in Recoveries, Regular Intake

Lead recovery is not affected by the method of treating copper dross except in the case of concentrates that assay fairly high ib copper (2% Cu or higher). In relation to the regular intake, the high-copper concentrates are the following:

	DST	Grade	
	Per Year	% Cu	% Pb
Colquijirca	4,800	3.00	53.0
Yauli	3,600	2.75	50.0
"Other Current" custom	6,000	3.00	45.0

(Note: these figures correspond to those on page 5, memo of October 1.)

As given on page 14 of the orginal memo (Oct. 1), the net loss of lead from lead-copper concentrates, in terms of feed-assay units, is:

Copper dross circulated: 3.5 %Pb
Copper dross treated separately: 3.0 %Pb

Thus, with separate treatment of copper dross, the additional lead recovered annually from the above concentrates would be:

(4,800 + 3,600 + 6,000)x(3.5 - 3.0)x20 = 14,400x0.5x20 = 144,000 lbs. lead.

(As stated, the respective lead losses, or feed-assay units, are approximations - to the nearest tenth of a unit. If the actual difference in feed-assay units should be 0.486 instead of 0.5, the additional lead recovered would be 140,000 lbs. per year.)

Since the difference in lead recovery depends upon the amount of lead transferred to the copper plant in the form of lead-plant matte, the additional lead recovered from the regular intake of lead-copper concentrates may be computed by an alternative method. Referring to page 14 of the original memo, the annual transference of lead in the form of blast-furnace matte is 1,437,000 lbs.; in the form of dross-furnace matte, 884,000 lbs. The respective losses of lead on re-treatment of the matte would then be:

	Annual Lead Loss, Lbs.
Blast-furnace matte, 12.5% Pb;	
loss: 100 - 75x(12.5 - 0.6)/12.5 = 28.60%;	
0.2860 x 1,437,000 =	411,000
Dross-furnace matte, 8.0% Pb;	
loss: 100 - 75x(8.0 - 0.6)/8.0 = 30.63%;	
0.3063 x 884,000 =	271,000
Decrease in loss = increase in recovery =	140,000

This alternative method of computation gives substantially the same result as the first method. Consequently, from 1955 onward, the annual increase in lead recovery from the regular intake may be taken at 140,000 pounds.

<u>Copper recovery, lead-plant feed.</u> Without a dross furnace, the net loss of copper in terms of feed-assay units is taken at 0.37% Cu; with a dross furnace, the net loss is assumed to be equivalent to 0.26% Cu. (See page 13, memo of Oct. 1.)

For the 1955-1957 period, the mean annual intake of regular feed (i.e., restricted feed without a dross furnace) is estimated at 150,600 DST; for 1958 and subsequent years, 160,800 DST (see pages 2 & 3, memo of Oct. 21). With separate smelting of copper dross, the increase in recovery of copper from the regular intake would then be:

		Copper Recovered, Annual Increase from Regular Intake
1955-1957:	105,600x(0.37 - 0.26)x20 = 105,600 x 2.2 =	232,000 lbs.
1958 et seq.:	160,800x(0.37 - 0.26)x20 = 160,800 x 2.2 =	354,000 lbs.

<u>Silver recovery, lead-plant feed.</u> Without a dross furnace, the silver loss in terms of feed-assay units is taken at 0.8 ozAg; with a dross furnace, the loss is assumed to be equivalent to 0.6 ozAg (see page 15, memo of Oct. 1). With separate smelting of copper dross, the apparent increase in recovery of silver from the regular intake would be:

		Silver Recovered, Annual Increase from Regular Intake
1955-1957:	105,600 x (0.8 - 0.6) = 105,600 x 0.2 =	21,100 ozs.
1958 et seq.:	160,800 x (0.8 - 0.6) = 160,800 x 0.2 =	32,200 ozs.

Net Returns from Additional Recoveries on Regular Intake

Value of production. Using the same metal prices and ex-Callao deductions as given in the memo of Oct. 1, the value of production in respect to the additional recoveries would be as follows:

		Increase in Annual Value
1955-1957:		
Copper (Callao):	232,000 lbs. x 0.23850 =	$ 55,300
Lead (Callao):	140,000 lbs. x 0.11920 =	16,700
Silver (New York):	21,100 ozs. x 0.75000 =	15,800
Total		$ 87,800
1958 et seq.:		
Copper (Callao):	354,000 lbs. x 0.23850 =	$ 84,400
Lead (Callao):	140,000 lbs. x 0.11920 =	16,700
Silver (New York):	32,200 ozs. x 0.75000 =	24,200
Total		$125,300

Allocable costs. As applied to the additional production of metals from the regular intake, the production costs would constitute the refining & shipping costs plus the cost of converting the additional copper. (Note: The lead-smelting cost would not change significantly; i.e., at the low copper content of the regular intake, the cost per ton of feed would be substantially the same with or without a dross furnace.) The cost of converting the additional copper may be taken at 1.350¢ per pound, which would give a combined cost of 3.105¢ per pound for converting, refining & shipping copper.

		Increase in Annual Costs
1955-1957:		
Copper (Callao):	232,000 lbs. x 0.03105 =	$ 7,200
Lead (Callao):	140,000 lbs. x 0.00830 =	1,200
Silver (New York):	21,100 ozs. x 0.04325 =	900
Total		$ 9,300
1958 et seq.:		
Copper (Callao):	354,000 lbs. x 0.03105 =	$11,000
Lead (Callao):	140,000 lbs. x 0.00830 =	1,200
Silver (New York):	32,200 ozs. x 0.04325 =	1,400
Total		$13,600

Gross margin and net returns on additional production of metals from regular intake:

	Annual Amounts 1955-1957	Annual Amounts 1958 et seq.
Value of production	$ 87,800	$125,300
Allocable costs	9,300	13,600
Contribution	$ 78,500	$111,700
Load, approx. 10% of costs	900	1,400
Gross margin	$ 77,600	$110,300
Income taxes etc., at 40% of margin	31,000	44,100
Net returns	$ 46,600	$ 66,200

Return on Investment

Combining these net-return increments with the net returns on extra custom intake (as given in the memo of Oct. 21), the over-all net returns would be as follows:

Year	Net Returns On Extra Custom Intake (see memo of Oct.21)	Net Returns On Additional Prod. from Regular Intake (see above)	Total	Cumulative
1955	$ 119,700	$ 46,600	$ 166,300	$ 166,300
1956	119,700	46,600	166,300	332,600
1957	119,700	46,600	166,300	498,900
1958	343,700	66,200	409,900	908,800
1959	343,700	66,200	409,900	1,318,700
1960	343,700	66,200	409,900	1,728,600
1955-1960	$1,390,200	$338,400	$1,728,600	

Theo. R.W.

cc: AHE (5)
RPK (10) New York
RPK Lima
WCS N.Y.
GR
JWH
ILB
JMM
HWH
DAR
RRvE

La Oroya,
July 23, 1951.

To : Mr. A. R. Merz,
Manager of Operations,
Oroya.

From : J. W. Hanley,
Superintendent.

Treatment of Copper Dross

Attached find a report by Mr. Higgs on the above, also electric furnace work done, and a report by Mr. Millican on his experiments.

As Mr. Higgs points out, the decision as to a dross furnace is very importand to our over-all lead expansion program, and is definitely indicated. We should reach a decision as quickly as possible as to the type of furnace and process to be used. Should further research work on the project be necessary, and it seems indicated from Millican's work, this should be done.

In connection with the above, we see no reason why the experimental furnace (copper dross) should not be torn down and the materials salvaged since it has not been in use for over six months and we have no plans for its operation in the future.

We would appreciate comments from the New York Staff on the attached reports.

/s/ J. W. Hanley

Superintendent

encls.
cc: ARM - 3
HWH
ILB
f.

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La Oroya, July 23rd, 1951.

To : Mr. J. W. Hanley,
Superintendent,
La Oroya.

From : H. W. Higgs
Asst. Supt., Lead Smelting,
La Oroya.

Subject: Treatment of Copper Dross

1.- The problem of the treatment of copper dross in Oroya has been met in the past by resmelting through the blast furnaces with sufficient sulphur to insure matting of the copper in the charge. The necessity of expanding operations in the future, improving metallurgical work and providing the most economical installation possible make it mandatory, for the following reasons, that separate dross smelting facilities be installed.

a) The large circulating load of dross (several thousand tons per month) displaces similar tonnages of sinter which will become available if contemplated sinter plant expansion proceeds.

b) In order to matte the copper it is necessary to leave considerable amounts of sulphur in the sinter or to add sulphur to the furnaces in the form of pyrite. This has several bad effects on operations as a whole.

1. It is almost impossible to control sulphur in sinter to any set limits. Good operation requires continual effort toward producing low sulphur sinter of excellent physical quality. Deviations from this practide result in production of poor quality sinter with subsequent furnace difficulties.

2. The addition of sulphur in any form may result in the formation of crusts, almost certainly increases copper and silver losses in the slags and is recognized to prohibit fast furnace speeds.

c) Smelting of dross in the blast furnace is not economically justifiable compared with other methods when consideration is given to all factors involved. These include actual cost of smelting, loss of tonnage on the blast furnace, additional metal losses in lead furnace slags and increased lead losses in the copper plant caused by poor ratios of lead to copper in the furnace matte.

2.- Two methods are used extensively at other smelters for treating this dross. Both involve smelting in small gas or oil fired reverberatory furnaces and can be described as follows.

a) Smelting with a silica-lime slag. Using this method the dross is charged into the furnace with sufficient silica and limerock to form a fluid slag with the oxidized portions of the dross. Three products are formed. Slag which is usually returned to the blast

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furnace. Speiss which is shipped to copper plants and bullion which is returned to the lead kettles. This is a fairly high temperature operation and the circulating load of copper between the dross furnaces and the kettles is often of large magnitude. Lead to copper ratios vary considerably in the speisses produced and depend upon the ratios of copper, arsenic, antimony and iron present. This process is often modified by the addition of metallic iron or blast furnace speiss to the furnace charge. Under optimum conditions very excellent ratios (20-1 copper to lead) can be obtained. However the process in practice has been found difficult to control and all American Smelting and Refining Co. Plants have adapted the Fleming Process developed by Mr. Fleming of that company.

b) The Fleming Process utilizes soda ash as a flux and produces three products. A Matte-Slag combination containing most of the iron which is shipped to copper plants. A high copper speiss also shipped and a bullion for return to the lead kettles. This is essentially a low temperature operation and the bullion produced is low in copper.

3.- The choice of processes must be based on economic considerations and the advantages and disadvantages of each are listed below.

a) Advantages of Silica Lime Process.

1. Cheapness of flux.
2. Simple operation to run.

b) Disadvantages of Silica-Lime Process.

1. High fuel cost (high temperature operation)
2. Short furnace life (high temp. operation)
3. Usually poor or erratic copper lead ratios.
4. When ratios are good copper assay is low.

c) Advantages of Soda Ash process.

1. Low fuel cost (low temp. operation)
2. Good furnace life (low temp. operation)
3. Uniformily good copper lead ratios over a wide range of compositions.
4. High assay copper products formed.

d) Disadvantages of Soda Ash process.

1. Cost of flux.
2. Difficult operation (removal of matte necessary at definite intervals or oxidation and magnetite formation takes place. Constant charging necessary to prevent oxidation and clost control of bath temperatures necessary to obtain good copper lead ratios)

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3. Experience of other plants indicates that tin oxidizes readily if present with formation of refractory crusts.

Insufficient data is available at present to enable us to reach a decision as to which process should be used however both can be conducted in the same furnace with nearly identical handling equipment and the decision could be based on actual full scale operational results.

4.- All dross smelting furnaces with which I am familiar have used either gas or oil as fuel. I can see no reason why this type of operation could not be carried on in an electric furnace of a type which heats from the top of the bath. In both operations (slag and soda ash) the melting temperatures of the various products are highest for the light materials and drop as the products get heavier therefore top heating is essential. When consideration is given to the use of electric furnaces vs. oil fired the following factors must be considered.

a) Original cost of installation.
b) Cost of fuel, oil vs. electricity plus electrodes.
c) Life of furnace brickwork.
d) The fact that electric furnaces have not yet been proven for this work and so far as I know are still in the process of experimentation.
e) Other operational costs, handling etc. should be about equal.

5.- The test work so far performed on dross smelting consists of three years trial smelting using the soda ash process and test smelting in a carbide furnace and the Detroit rocking furnace.

Soda ash smelting in the dross reverb was discontinued in November 1950. This operation was never successfull due to the location of the furnace. Reasons for discontinuing this operation are given in my letters to you of April 22, 1950, "Dross Furnace Operation" and October 10, 1950 "Operation of Present Dross Reverb. Furnace. Smelting in the carbide furnace was covered in my report of November 28, 1950, "Copper Dross Smelting". I am attaching a copy of the pertinents parts of that report to this. Mr. H. Millican's full report covering the experimental work performed in the Detroit rocking furnace is also attached.

Both the carbide furnace and Detroit rocking furnace experiments indicated that our dross contains an excess of collector (arsenic plus antimony) and that excellent copper lead ratios could be obtained by the addition of scrap iron. These tests as they were performed gave results similar to those obtained using the silica lime slag process and comparable results could probably be expected when using this process in either oil fired or eceltric furnaces. It is probable that the soda ash process could be used in the electric furnace also.

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It was found impossible to obtain any data on the consumption of electricity per unit of dross smelted in either test. The Detroit rocking furnace used excessive amounts but was lined with only 4" of magnesite brick.

6.- The balance of Mr. Millican's report concerns the work he performed trying chloride fuming and smelting dross with litharge slag to effect the removal of tin. Chloride fuming results were largely negative, however, the experimental work, while admittedly scanty, indicates that tin could be quantitatively removed if high litharge slags can be produced and used.

7.- Smelting dross using the ordinary processes in either an oil or electric furnace will result in loss of most of the tin into the copper circuit. Mr. Millican's litharge slag smelting process apparently offers a solution to this problem and additional experimental work should be done immediately to determine the true possibilities of this method.

The actual installation of dross smelting facilities, I feel, should be considered as part of the overall smelting expansion program. This installation would best be located under the present lead plant craneway where dross could be charged hot directly from the kettles. Construction of new furnaces and the dismantling of # 1 Blast furnace would provide room for this.

If the litharge slag smelting proves unsuccessfull a decision must be reached as to the use of electric furnaces or oil based on the factors mentioned previously.

If plans are made to proceed with smelter expansion in the near future it would be far better to install dross smelting facilities as part of this project than to proceed at present however if this expansion is to be postponed for some time then dross smelting facilities must be installed regardless. In the meantime we will continue to smelt in the blast furnace.

H. W. Higgs
Asst. Supt. - Lead Smelting

cc: JWH-4
ILB
HWH
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HWH/af

Excerpts letter HWH to JWH of Nov. 28, 1950.

Subject: Copper Dross Smelting.

During the latter part of July at Mr. Reinberg's suggestion we tried smelting dross in one of the carbide furnaces with the object in view of determining the possibility of developing a process using this type of furnace. At that time operations were conducted on day-shift only. All products were tapped through a single tap hole into a small transfer ladle with an approximate capacity of 350 pounds. The material was allowed to freeze in the pot which was subsequently dumped and various products separated cold. In practice it was often found necessary to dump the pot before the lead had frozen but this should have had little effect on the analyses of matte, slag and speiss.

We first attempted to smelt dross in the furnace using no flux of any sort. Dross was charged around the electrode and products tapped at frequent intervals. We immediately started producing speiss and lead but also produced a slag which was extremely viscous and of the following assay:-
Cu 5.65%, Pb 33.0%, SiO_2 11.6%, Al_2O_3 3.3%, Fe 12.7%, CaO 2.9%, As 4.5%, Sb 4.3%.
In view of the lead content of this slag and the possibility that this le d might exist as litharge we started to add fine coal but this did not change the characteristics of the slag. At the time assays were not available and later some burnt lime was added to the charge to flux any silica present but this also gave no relief. Speiss and lead produced during this period liquated nicely and froze in distinct layers in the pot. Visual observation disclosed two layers of speiss with the following assays:-
Top layer Cu 50.2%, Pb 27.7%, As 2.67%, Sb 1.45%, Fe 1.5%, S 3.28%.
Bottom layer Cu 48.9%, Pb 19.9%, As 17.6%, Sb 7.01%, Fe 0.2%, S .28%.
The bottom layer is a true speiss but the top layer is a product of questionable origin. The best ratio of copper to lead existed in the bottom layer of speiss with a ratio of 2.46 to 1.

The slag formed was proving almost impossible to remove from the furnace. It was then suggested that granulated reverberatory slag be added to the charge since it easily melts and it was felt that it had sufficient latitude in composition to flux the still unknown slag being formed from the dross. By using comparatively large quantities of this reverberatory slag we obtained a slag which gave little trouble and had the following composition.
Cu 2.75%, Pb 2.4%, SiO_2 35.4%, Al_2O_3 3.8%, Fe 28.3%.
A comparatively minor amount of matte was produced assaying as follows:
Cu 41.95%, Pb 14.8%, Fe 17.9%, S 19.0%.
A Speiss was produced assaying
Cu 34.9%, Pb 22.5%, Fe 20.4%, As 12.1%, Sb 6.1%.

Results from this test indicated that the dross was deficient in iron to form a good slag so it was decided to try the addition of iron to the charge. This was done by adding shavings etc. to the charge. With iron alone the slag trouble returned so the run was finally conducted using slag in addition to the iron. Slag, matte and speiss were obtained with the following compositions.
Slag Cu 0.70%, Pb 0.2%, SiO_2 41.8%, Al_2O_3 0.4%, Fe 23.6%, CaO 7.5%.
Matte Cu 39.00%, Pb 10.6%, Fe 21.3%, S 21.4%.
Speiss Cu 18.00%, Pb 1.2%, Fe 47.8%, As 19.2%, Sb 5.2%.

At the time I started working in East Helena in 1941 the plant was treating drosses in the following fashion. The charge consisted of all dross produced plus all blast furnace speiss (added to the charge hot) plus sufficient silica and limerock for fluxing. At that time East Helena was receiving fairly large quantities of arsenic in gold ores and it was their practice to use large amounts of scrap iron on the blast furnaces (5%). As a result they produced large quantities of irony blast furnace speiss. Metallurgical results in the dross furnace were excellent and a speiss was produced assaying about as follows. Cu 20%, Pb 1% to 2%, Fe 45%, As 24%.

The final speiss produced in the carbide furnace closely resembled the speisses being produced in East Helena in 1941. The condition that brought this about was undoubtedly the addition of metallic scrap iron to the charge along with sufficient arsenic in the dross so that it was possible to duplicate the conditions obtained in East Helena by the addition of blast-furnace speiss to the reverberatory furnace charge. When such conditions can be obtained it is possible to produce speiss with better copper lead ratios than can be obtained with the soda ash process.

The carbide furnace fails for several reasons. A liquid bath is maintained above which charging takes place. Smelting occurs at the face of the bath. Lead and speiss are liquated above the bath and are heated in passing through the slag layer. Consequently they are tapped hot with a large absorption of speiss in lead and lead in speiss. Slow colling in the pot allows the products to separate to some extent but separation of the products cold is very difficult and impossible on a large scale. Also the speiss and lead passing through the slag layer tend to cool it and this effect is probably partially responsible for the trouble we had in removing slag.

HEH/af

La Oroya, May 25, 1951.

To : Mr. E. H. Gates, Director of Research

From : Herman Millican

Subject: Treatment of Copper Drosses

Much of the material covered in this report has been included in previous reports, but will be covered here for the sake of completeness.

The Fleming process for separation of lead and copper in copper dorsses had previously been tried in the smelter, but with generally unsatisfactory results. It was decided to try various other schemes, in the hope of finding a process which could replace the Fleming one. A secondary consideration was recovery of tin and indium from the copper dross. In the Fleming process, all of the tin and indium go to the matte and speiss layers and are eventually lost.

Equipment first used was the Detroit Rocking Furnace usually employed for tin dross reduction, and several small portable casting pots borrowed from the Bismuth Plant. A series of preliminary runs was first made to determine the optium weight of copper dross to charge into the furnace each time. It was found that 200 kg. of dross would fill the furnace fairly well and would yield about three-fourths of a pot of product. All work done at the rocking furnace was with a charge of 200 kg. of dross. All weights of fluxes are here reported as percentages of the weights of the copper dross charged.

When no flux was used with the copper dross, there was a certain amount of floating clinker which did not melt, even at a temperature of 2600°F. It was decided to try varying quantities of limestone and silica to obtain a fusible slag. Test runs were made with 2-1/2% silica and 5% limestone, 3-3/4% silica and 3-3/4% limestone, 5% silica and 2-1/2% limestone, 5% silica and 10% limestone, 7-1/2% silica and 7-1/2 limestone, and 10% silica and 5% limestone. For some reason still not apparent, none of the mixtures produced a fusible slag, but greatly aggravated the problem of the clinker. The accretions had to be chipped out of the furnace after almost every run.

5% iron was added to another fusion of dross, and 2-1/2% charcoal to still another. These, too, failed to give any noticeable improvement in melting efficiency.

It was observed that if heating were continued long enough and at a high enough temperature, a certain amount of the clinker seemed to be absorbed by the matte layer. The matte layer was very thin, so it was decided to add extraneous sulfide to the charge. This would thicken the matte layer, thereby increasing the capacity of the layer to absorb the clinker, and possibly improving the copper-lead separation.

An addition of 10% Paragsha lead concentrate was made to one charge of copper dross, with a resulting great increase in melting efficiency. A molten product, almost devoid of clinker, was obtained at 2200°F. It was then decided to run a series of quantitative tests, using pyrite flux instead of lead concentrates, since weight for weight, pure pyrite contains almost four times as much sulfur as pure galena.

Six melts were made in the tests--bland, 5% FeS2, 10% FeS2, 15% FeS2, 20% FeS2, and 5% FeS2 + 5% Fe. Assays of speisses and mattes are tabulated below.

Product	Cu, %	Pb, %	Cu/Pb	Fe, %	S, %	As, %	Sb, %	SiO2, %	CaO, %
Matte									
Blank	49.0	22.4	2.18	2.3	10.2	6.1	1.5	0.6	0.1
5% FeS2	32.5	24.3	1.33	12.8	11.7	1.5	0.7	6.0	1.2
10% FeS2	38.8	28.7	1.35	7.4	16.5	1.8	0.6	0.8	
15% FeS2	34.3	31.0	1.11	11.2	17.0	1.6	0.7	0.6	0.5
20% FeS2	26.3	33.6	0.78	15.3	15.7	2.0	1.0	2.2	
5% FeS2 + 5% Fe	40.6	18.5	2.19	14.1	19.4	1.1	0.6		
Speiss									
Blank	49.8	17.6	2.83	2.1	2.2	12.2	6.6		
5% FeS2	40.6	21.3	1.91	4.9	1.6	14.1	5.2		
10% FeS2	41.2	24.2	1.69	2.8	1.9	13.8	5.4		
15% FeS2	(No sample taken)								
20% FeS2	14.5	62.1	0.23	2.8	2.5	7.8	4.8		
5% FeS2 + 5% Fe	39.3	25.0	1.57	6.9	2.2	11.1	5.8		

When 15% FeS2 and 20% FeS2 were added to the charge, there were only two layers--a matte layer and another layer which seemed to combine the physical properties of speiss and metallic lead. As can be seen from the assay of "Speiss--20%FeS2" the bottom layer was apparently a speiss and the metal layer had completely disappeared. As might be expected, the quantity of matte produced increased with the quantity of pyrite added, until the fusion with 20% FeS2 contained approximately half matte.

The addition of pyrite, though it made the fusion of the charge much easier, had such an adverse effect upon the copper-lead separation that the experiments were dropped. On the other hand, the metallic iron which was added in the last fusion seemed to increase the separation of copper and lead, so it was decided to investigate more thoroughly the effect of varying percentages of iron upon the copper-lead separation.

Instead of using pyrite or a made-up charge of limestone and silica to flux the charge, it was decided to try granulated reverberatory slag. After several experimental runs, it was found that the addition of a minimum of 20% reverberatory slag would give complete fusion of the charge at 2400°F. All succeeding fusions, blank and with varying amounts of iron, were made with this addition of 20% slag. It is quite possible, that, in a larger furnace, smaller additions of a better-balanced slag could be made. Four melts were made for assay purposes--a blank, 5% Fe, 10% Fe, and 15% Fe. Assays of slags, mattes, speisses, and metals are listed below.

Product	Cu, %	Pb, %	Cu/Pb	Fe, %	S, %	As, %	Sb, %	SiO_2, %	CaO, %	Al_2O_3, %
Slag										
Blank	11.55	4.0		35.8				27.2	4.5	5.1
5% Fe	0.50	1.2		38.7				29.4	4.2	5.5
10% Fe	3.90	0.8		40.3				23.2	3.2	4.4
15% Fe	1.50	0.8		41.2				26.6	3.5	4.8
Matte										
Blank	42.3	24.2	1.75	7.3	16.8	2.4	0.5			
5% Fe	42.3	15.7	2.69	13.5	15.7	3.1	0.6			
10% Fe	39.8	17.5	2.24	15.7	19.2	1.7	0.5			
15% Fe	35.5	9.8	3.62	20.7	19.8	1.3	0.6			
Speiss										
Blank	41.2	29.6	1.51	3.0	2.3	10.8	4.5			
5% Fe	39.6	20.7	1.91	8.2	3.0	12.2	4.4			
10% Fe	34.3	18.6	1.84	16.1	3.8	9.7	4.1			
15% Fe	24.2	4.2	5.76	36.4	3.7	13.5	4.0			
Metal										
Blank	0.35	93.2				0.3	2.4			
5% Fe	0.35	94.3				1.5	2.8			
10% Fe	(No sample taken. Button frozen to pot.)									
15% Fe	1.00	95.3				0.1	0.9			

It can be seen from the assays that, though the percentage of copper in the matte and the speiss decreases when metallic iron is added, the ratio of copper to lead show a steady improvement.

When the encouraging copper-lead ratios were obtained with the addition of metallic iron to the charge, it was decided to have the various products assayed for tin, to see whether the iron had reduced the tin to the metallic state or not. Assays are listed on the next page.

	Blank, % Sn	5% Fe % Sn	10% Fe, % Sn	15% Fe % Sn
Slag	0.99	1.23	0.84	0.60
Matte	0.11	0.25	0.69	0.27
Speiss	0.38	1.44	2.29	1.34
Metal	0.82	0.35		0.31 (3.01)°

° Metal assaying 0.31% Sn was taken from the bottom of the button after it had solidified. Metal assaying 3.01% Sn was taken from the top of the button.

It can be seen, then, that the addition of metallic iron does not have any great effect upon the proportions of tin in the slag, the matte, the speiss, and the metal.

It was then decided that, since stannous chloride is very volatile at the fusion temperature of copper dross, it might be possible to fume the tin from the melted dross with common salt. Runs were made as follows: blank, 2% NaCl, 5% NaCl, and 10% NaCl. Assays are listed below.

	Blank, % Sn	2% NaCl, % Sn	5% NaCl, % Sn	10% NaCl, % Sn
Fume		9.92	5.39	3.47
Slag		8.00	7.44	2.46
Matte		0.31	2.93	0.47
Speiss		0.47	2.39	1.03
Metal	0.64	0.73	1.46	0.33

No weights were obtained, except in the case of the fume. For 2% NaCl the fume weighed 3 Kg., for 5%, 9 kg., and for 10%, 10 kg. The contained tin in the fume was roughly 0.3 kg. for 2% salt, 0.5 kg. for 5% salt, and 0.35 kg. for 10% salt. The percentage of recovery was on the order of 10-20% in each case.

The larger percentages of salt seemed only to produce more fume and lower the percentage of contained tin in the fume without improving the extraction of tin, so the experiments were dropped. It should be noted that temperatures of 2700-2800°F were reached in the fusions, and that it would probably not be practical in a large-scale furnace to go to any higher temperatures.

It was noticed that, when salt was used as a slagging agent, the percentage of tin in the slag increased greatly, possibly because of some oxidation reaction catalyzed by the chloride. It was decided to try a combination of salt and litharge to see if all of the tin could be oxidized and taken into the slag layer. For the preliminary tests, two melts were made--one with 5% NaCl and 10% litharge (fines from Betts Plant hammermill), and the other with 5% salt and 25% litharge. Assays are listed below:

	Sn, %	Cu, %	Pb, %	$\frac{Cu}{Pb}$
5% NaCl, 10% PbO				
Slag	8.87	4.99	19.1	
Matte	2.74	46.0	29.5	1.56
Speiss	2.06	48.4	21.9	2.21
Metal	1.01			
5% NaCl, 25% PbO				
Slag	3.53	3.00	34.4	
Matte	0.68	52.8	25.2	2.10
Speiss	0.10	51.3	22.3	2.30
Metal	0.02			

The assays for copper and lead were made to see if the oxide slag had any effect upon the copper-lead ratio in the matte and the speiss. The tests tended to show that the tin would go into the slag, and that the copper-lead separation would be improved, at least slightly, so it was decided to investigate the possibilities of the scheme more thoroughly. Since it would be very difficult to obtain the litharge needed to treat 40 to 80 tons of copper dross per day, it was decided to use refinery slag from the Bismuth Plant converters. This slag contains roughly 50% lead 20% antimony and small percentages of copper, arsenic, and silver, all as oxides. It was also decided to try to use the refinery slag alone, since salt increased the amount of fume formed.

A preliminary fusion was made with 30% refinery slag. There was so much oxide slag left floating on top of the fusion, however, that it was decided to cut down the amount of slag used. No samples were taken of this preliminary fusion. Four fusions were made for assay purposes--blank, 5% refinery slag, 10% refinery slag, and 15% refinery slag. Assays are listed below and on the next page.

	Sn, %	Cu, %	Pb, %	Cu/Pb
Blank				
Matte	1.72	44.1	30.0	1.47
Speiss	0.07	46.3	26.2	1.77
Metal	0.02			
5% Refinery Slag				
Fume	12.4			
Slag	No slag layer present			
Matte	0.38	48.7	29.6	1.65
Speiss	0.97	49.1	21.9	2.242
Metal	Trace			
10% Refinery Slag				
Fume	10.2			
Slag	3.81	23.7	36.4	
Matte	0.64	46.3	31.3	1.50
Speiss	Trace	50.9	20.0	2.54
Metal	0.01			
15% Refinery Slag				
Fume	8.93			
Slag	2.69	1.73	46.8	
Matte	Trace	48.9	28.7	1.70
Speiss	Trace	47.8	23.0	2.08
Metal	0.01			

The fumes, though containing a high percentage of tin, were of such low weight as to be relatively unimportant. The weights of fume were 2 kg. for 5% refinery slag, 3 kg. for 10% refinery slag, and less than 1 kg. for 15% refinery slag.

Not weights of slags were taken except in the case of the final melt (15% refinery slag), which weighed 23 kg. The refinery slag was apparently more effective in slagging the tin than was pure litharge, probably because any of the antimony oxides contains more oxygen per unit weight than does litharge.

It was then decided to try blowing the copper dross in a converter, to see if the copper-lead ratio could be further improved. A small converter at the antimony plant was renovated and used for this work. The converter was oil-fired rather than electrically heated, so the work was done with somewhat lower temperatures than were used in the rocking furnace. Since the converter had a somewhat larger capacity than the rocking furnace, 500 kg. of dross was charged each time instead of 200 kg.

On the first run, it was decided to turn off the burner while the blow was in progress, since otherwise there was a great deal of fuming. As a result, the furnace became somewhat cold, since the heat of formation of litharge did not completely counteract the cooling of the converter by radiation. There was no matte obtained, but a sample of the speiss was taken and sent to the laboratory. It assayed 57.6% Cu and 9.8% pb, for a Cu/Pb ratio of 5.88.

On the second charge it was decided to ignore the fuming and keep the burner on while the charge was being blown, so that the dross would stay hot. Two sets of samples were taken--one after 15 minutes of blow, and the other after 30 minutes of blow. Assays are listed below.

	Cu, %	Pb, %	$\frac{Cu}{Pb}$	Sn, %
After 15 minutes				
Slag	1.11	47.6		1.30
Matte	52.6	27.4	1.92	0.01
Speiss	52.8	18.2	2.90	0.01
After 30 minutes				
Slag	3.29	50.1		0.80
Matte	59.0	20.8	2.84	0.01
Speiss	56.7	14.3	3.97	0.01
Metal	0.09	99.3		Trace

It was then decided to determine what length of blow was necessary to obtain good results without getting an excessive amount of slag. Results were very puzzling. In one melt, for instance, samples were taken after 15 minutes, 30 minutes, one hour, and an hour and a half. After 15 minutes, the matte assayed 22.6% pb; after 30 minuted 21.9% Pb; after one hour, 22.5% Pb; and after an hour and a half, 21.7% Pb and 58.2% Cu. Two samples of speiss were taken. That taken after 30 minutes assayed 15.3% Pb, and that taken after an hour and a half assayed 19.5% Pb and 56.9% Cu. In other words, the lead content of the matte was more or less constant, and that of the speiss was apparently increasing with continued blowing.

It was then decided that possibly the temperature was the controlling factor. A charge was blown for thirty minutes, a sample of the speiss was taken, and the converter was allowed to chill until the crust started forming on the oxide slag (±1500°F). A sample of both matte and speiss was taken, and all three samples were sent to the laboratory. Assays were as follows: Hot speiss, 15.0% Pb and 60.1% Cu, for a ratio of 4.01; cold speiss, 7.1% Pb and 66.5% Cu, for a ratio of 9.37; and cold matte, 17.9% Pb and 57.3% Cu, for a ratio of 3.20. There was some freezing of the matte when the sample was taken, however, so pouring would probably have to be done at a somewhat higher temperature in practice. Incidentally, a small amount (10 kg. or 2%) of silica was added to this charge, with the result that the slag, instead of becoming viscous, remained fluid right down to its freezing temperature.

It was then decided to try using refinery slag again, without blowing the charge. Apparently the melt was allowed to cool too much before cooling, since no matte was obtained, and there were some solids left in the furnace after pouring. Assays of the products are listed below: (10% refinery slag was used.)

	Cu, %	Pb, %	Cu/Pb	Sn, %	Fe, %	As, %	Sb, %
Slag	3.54	53.1		0.85	6.1	6.74	7.16
Speiss	58.2	12.1	4.81	0.01	0.6	18.0	6.03
Metal	1.14			Tr.			1.55

From the admittedly scanty evidence, then, it would seem that the following are true: That the oxide slag treatment will give copper-lead separations about as good as the Fleming process (Average of the Fleming furnace copper-lead ratios for the period January-August 1950 are as follows: slag, 3.47; speiss, 3.14; and matte, 2.73.); that tin and possibly indium can be extracted quantitatively from the matte and the speiss (An assay for indium of one of the oxide slags has just been finished. The slag contained 0.30% In, which would indicate that probably all of the indium is in the oxide slag.); that the treatment does not need any special fluxes, such as the Fleming process does; that the oxide slag can be furnished either by blowing or by adding refinery slag, though blowing seems to be somewhat more efficient; that an oxide slag of 10% of the weight of the dross charge is sufficient; that probably 75% of the lead present in the dross can be recovered as metallic lead, the rest being in the matte and speiss which go to the copper circuit and in the oxide slag which returns to the blast furnaces; that the matte and the speiss will contain around 50-60% copper; and that a treatment temperature of 1800-1900°F is sufficient, as compared to the metallic-iron treatment temperature of 2200°F.

It is not known whether there would be any eventual problem of either liner corrosion or accretions forming, but there was no evidence of either one in the short time in which the furnace was operating. If it is decided to do further development or operations work on this process, I will gladly answer any questions that I can. Mr. Gates has my home address.

Yours truly,

Herman Millican

Extra copy HWH/af