



CERRO DE PASCO CORPORATION NEW YORK  
INTER-DEPARTMENT MEMORANDUM

Date: July 14, 1952

To: R. P. Koenig

A-51

From: L. Addicks

Subject: Concentrating Practice

1. I should like to see the question put to our Concentrator Department as to what changes would result in the net recoveries of values and the grade of concentrates produced if the mills were permitted to make a foul middlings product. I am not trying to start anything at the moment for we certainly have our hands full enough with new projects but the answer to the question might have a bearing on future policy. I give the reasoning below.

2. We have three major circuits, copper, lead and zinc, and our complex ores result in making us feed leady ores in the copper furnaces, copper-zinc material to the lead plant and so on. Could a better separation be effected at the concentrators, cheaper costs and better recoveries would result at the smelter.

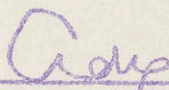
3. We tend to think of metal losses as a smelting problem but really we should consider overall recoveries between the mineshaft and the market. The current May "Report on Operations" shows the following tailing losses:

<u>%</u>	<u>Cu</u>	<u>Pb</u>	<u>Zn</u>	<u>Ag</u>
Paragsha	-	12.8	15.0	27.4
Morococha	7.31	20.00	7.44	9.26 (Bulk concentration)
Casapalca	22.0	8.6	11.3	12.0 (Triple concentration)

Could a lower tailing be made if middlings were permitted?

4. The question follows of what we would do with the middlings, the whole modern trend in concentrating having been toward avoiding intermediates. In the meantime, however, metallurgy has been changing. The new Sterling process will presumably take messes and make a fine end product with good separations of the impurities. The new process sponsored by the Cyanamid people with leaching under heat and pressure and fractional precipitation by hydrogen will presumably do likewise (incidentally ammonium sulphate is a by-product in some cases).

5. So, making any and all assumptions necessary, would there be a financial margin to work with, viewed from this new angle?

  
Lawrence Addicks

LA:JC

A-64

April 10, 1945

Mr G. P. Sawyer, Vice Pres.,  
Cerro de Pasco Copper Corp.,  
40 Wall Street, New York City 5

Dear Mr Sawyer: CENRO EXCELSIOR ORE

Noting Mr Wright's question in his letter of March 10th regarding the effect of altitude upon the oxidation of copper minerals in Excelsior ore, my own view is that altitude may well interpose difficulties. This is the same question that was brought up when we considered flash roasting of zinc concentrates several years ago. Trail argued that altitude would not enter but I never thought they made out a case.

Mr Smith has made the right suggestions in the tests he recommends. Practical tests are the best answer. Preheating the air to the roaster would seem to be indicated plus a longer time in contact.

Very truly yours,

*Ray*

April 9, 1945

American Cyanamid Company  
Stamford, Connecticut

Attention: Mr. S.J. Swainson

Gentlemen:

We are submitting herewith two copies of Mr. Wright's comments relative to your report of January 8, on tests of Excelsior ore.

The second paragraph under "Hydrometallurgical Tests" page 4 of Mr. Wright's letter raises the question of oxygen concentration of air at Oroya and its possible effect on roasting of the copper minerals in the ore. Would it be possible to repeat your experiment under roasting conditions approximating those at Oroya, namely at an oxygen concentration of about 14% by volume at sea level? We estimate that roasting the ore in a mixture of two parts air and one part nitrogen should give this condition.

Meanwhile we are suggesting that Mr. Wright repeat his tests using an air plus oxygen mixture at Oroya, which will be equivalent to sea level oxygen concentration.

The experimental work is for our benefit and the cost of same should be invoiced to us. Kindly give us an estimate of this cost.

We wish to thank you for your cooperation and await your comments.

Yours truly,  
CERRO DE PASCO COPPER CORPORATION

W. C. Smith  
Metallurgist

WCS/RL  
Encs.

Mr. Cox.

See page 4.

La Oroya,

March 10, 1945

Mr. V. L. McCutchan,  
Assistant General Manager,  
La Oroya.

Dear Sir:

- Cerro Excelsior Ore -  
Tests by American Cyanamid Co.

We acknowledge receipt of the American Cyanamid Co's report of January 8, together with covering letters from our New York Office, on the metallurgical testing of the Excelsior dump ore. This report by Messrs. Kentro and Davis is indeed excellent; it is comprehensive, straightforward and concise - and the photomicrographs are impressive.

Flotation Tests. It is unfortunate, however, that the flotation tests were not made at the time the A.C.Co. received the sample, for it is evident that the sample had deteriorated considerably during the intervening months. The flotation results obtained by the A.C.Co. doubtless represent the true amenability of the sample at the time it was tested, but the results are decidedly inferior to those obtained by this laboratory in testing a duplicate sample in December 1943. (Note: The original sample was split, and the A.C.Co.'s portion was shipped from Oroya in December 1943; it was received by the A.C.Co. in Feb., 1944 but their flotation tests were apparently postponed until the end of 1944. Incidentally, it is quite possible that the sample was adversely affected by heat while in transit through the tropics; we have found that samples of Cerro copper ore deteriorate rather rapidly when stored in a warm room.)

For comparison, the following tabulation shows the results of: a) the A.C.Co's best flotation test, Test N<sup>o</sup>2, using copper sulphate and sodium Aerofloat; b) the A.C.Co's test using our reagent combination (i.e., Test N<sup>o</sup>4, with potassium amyl xanthate as collector); and c) our test on a duplicate sample, made in December 1943.

	Ratio	% Recoveries				
	Conct	% Cu	Au	Ag	Cu	Fe
a) A.C. Test N <sup>o</sup> 2, Cu cleaner con.	7.58	9.35	33.7	37.8	79.3	14.6
b) A.C. Test N <sup>o</sup> 4, Cu cleaner con.	5.15	5.44	32.4	36.3	70.3	26.6
c1) CdeP Test N <sup>o</sup> 3, Cu cleaner con.	11.50	15.40	33.0	44.2	87.4	7.8
c2) do Cu rougher con.	5.88	8.14	40.9	51.5	90.2	15.7

The test conditions as applied to b and c were as follows:

	b A.C.Co. test	c C de P test
Grind, % minus 200-mesh	91.5	80.0
pH, flotation pulp	10.3	10.5
Reagents, lb./ton ore:		
Ca(OH) <sub>2</sub> , to ball mill	11.0	6.0
NaCN, to ball mill	0.30	0.30
Reagents, lb./ton ore (cont.):		
PAX(Z-5), to conditioner	0.15	0.050
PAX(Z-5), to rougher flot.	--	0.075
Pine oil, to conditioner	0.025	0.075
Pine oil, to rougher flot.	0.025	--
Weight of sample, grams	600	2000
Flotation time, roughing minutes	8	20
Flotation machine	Fagergren	M.S.-type (CdeP)

The flotation method was substantially the same in both cases (b and c). The degree of grinding was finer in the A.C. test than in our test, but it is not likely that over-grinding was the cause of the inferior results. The fact that twice as much lime had to be used in the A.C. test to obtain the same pH is significant; it substantiates our contention that the poor results of the A.C. Co's tests were due to the deterioration of the ore sample - i.e., a greater consumption of lime indicates greater decomposition (oxidation) of the ore and, consequently, inferior amenability to selective flotation. Our flotation tests, of course, were made on a fresh sample.

We had investigated the efficacy of the various American Cyanamid reagents (Aerofloats, etc.) in the flotation of Cerro copper ores, but amyl xanthate proved to be the most suitable collector; that is, for the same percentage recovery, amyl xanthate produced a higher grade concentrate. Incidentally, the current milling of Excelsior dump ore, employing the lime-cyanide-pine oil-amyl xanthate reagent combination, recovers 85% of the copper in a concentrate assaying 16% Cu - and the mill feed assays only 1.2% Cu.

In tests on fresh samples, the use of copper sulphate as an activator in copper flotation had no beneficial effect. We have not tried such reagents as sodium bisulphite, calcium bisulphite and ammoniacal copper sulphate in conjunction with the Aerofloats. To determine the effect of these reagents combinations on the flotation of fresh samples of ore, we shall repeat the A.C. Co's tests.

Gold. With regard to gold, of particular interest was Mr. Davis' discovery of electrum (Au-Ag alloy) in a paner concentrate of Cerro pyrite. Although we knew from indirect evidence that a portion of the gold was liberated from the pyrite by fine grinding, we have never detected native gold nor electrum in specimens of Cerro copper ores. Since the particles of electrum

are minute - not readily revealed at magnifications below 1000X - and probably too small for determinative tests, and since electrum is isotropic and exhibits a pale yellow color similar to pyrite, it is assumed that identification was based on visual indications of softness, such as relief in polished section. In this assumption correct, or was the identification of electrum confirmed by positive tests?

Our investigation of the distribution of gold in the Excelsior dump ore indicates that, after grinding to 80% minus 200-mesh, 10% of the gold is free, 20% is associated with the copper minerals, and 70% remains locked in pyrite. (Note: this percentage distribution is based on the gold content of a large number of products obtained in superpanning 'infrasized' fractions of flotation concentrates, middlings and tailings; and the distribution of gold has been verified by scores of tests in which the actual gold assays checked closely with the calculated assays. See News Letter for April 1944.)

There are several generations of pyrite in the Cerro copper ores, and it is not likely that each generation is equally auriferous - in fact, some of the pyrite may even be barren. However, in none of the tests made to date has been any positive indication of a separation of high-gold pyrite from low-gold pyrite; i.e., for a particular sample of ore, pyrite concentrates representing high and low ratios of concentration always assay the same in gold. Thus, the lower ratio of concentration the higher the recovery of gold. There has been no exception to this relationship in any of our tests.

Despite the high alkalinity of the flotation pulp, our investigations give strong evidence that practically all of the free gold reports in the copper concentrate - at least in the case of fresh samples.

Contrary to the results obtained by our laboratory in hundreds of tests, the A.C. Co. found (in four tests) that the recovery of iron varied from 11.2% to 26.6% while the recovery of gold remained substantially constant. If this anomaly is not due to errors in assaying, then the only plausible explanation we can offer is that the deterioration of the ore sample impaired the floatability of the free gold. Referring to the tabulation of comparative results on page one of this letter, and applying our figures for the distribution of gold, the predicted recoveries of gold would be:

	a	b	c1	c2	
	A.C. Co.	A.C. Co.	C de P	C de P	
	Test N <sup>o</sup> 2	Test N <sup>o</sup> 4	Test N <sup>o</sup> 3	Test No 3	
	<u>cleaner con.</u>	<u>cleaner con.</u>	<u>cleaner con</u>	<u>rougher con</u>	<u>con</u>
Au, free	10.0%	10.0%	10.0%	10.0	%
Au in Cu minerals	15.9	14.1	17.6	18.0	
Au in pyrite	10.2	18.6	5.5	11.0	
Au recovery, predicted	<u>36.1%</u>	<u>42.7%</u>	<u>33.1%</u>	<u>39.0</u>	%
Au recovery, by assay	<u>33.7%</u>	<u>32.4%</u>	<u>33.0%</u>	<u>40.9</u>	%
Difference: loss of free gold?	2.4	10.3			

Note: Au in Cu minerals = 0.2 x recovery of copper; Au in pyrite = 0.7 x recovery of iron.

But we are inclined to believe that the apparent lack of conformity of the gold recoveries in the A.C. Co's tests is due to assay discrepancies; a difference of 0.005 oz/ton in the gold assays of the flotation products could account for the apparent discrepancies in gold recovery. In this connection, it is noted that the calculated feed in A.C. Test N<sup>o</sup>1, as derived from the assays of the products, is 0.033 oz. Au, whereas the calculated feed in A.C. Test N<sup>o</sup>4 is 0.054 - an apparent difference of 64% in gold content.

Hydrometallurgical Test. The results of the A.C. Co's hydrometallurgical test - roasting & acid-leaching followed by cyanidation - were surprisingly good: high extractions of copper & gold and low consumption of cyanide. Roasting & leaching tests made here on a duplicate sample failed to yield satisfactory results; only 80% of the copper was extracted by an acid leach of the roasted ore, and the residual copper (plus other cyanicides ) caused an excessive consumption of cyanide.

After receiving the American Cyanamid Company's report, we endeavored to repeat their test, but the results were poor: low extractions and high consumption of cyanide. It is obvious that something was amiss. Either our technique was faulty, or the atmospheric conditions prevent adequate oxidation of the copper minerals to CuO and CuSO<sub>4</sub>. We do not like to blame the altitude for our poor results, but it is possible that the low oxygen pressure at Oroya may tend to produce more cuprous oxide (Cu<sub>2</sub>O) in the calcine than would be produced at sea level. Of course, if either cuprous oxide or cupric ferrite are present in the calcine in appreciable amounts, a simple acid leach will result in a low extraction of copper and a correspondingly high consumption of cyanide in the subsequent cyanidation operation.

What are the opinions of Messrs. Addicks and Smith regarding the possible formation of cuprous oxide and cupric ferrite when roasting Excelsior dump ore (65% pyrite, 3% enargite) at an atmospheric pressure of 9.6 psi and at a temperature of 430°C±5° in a well-ventilated muffle furnace with frequent rabbling of the charge? We should also like to ask the American Cyanamid technicians whether they were able to extract 90% of the copper from the Excelsior dump ore consistently and without difficulty in successive roasting & acid-leach tests.

Our experiments along these lines, i.e., low-temperature roasting and hydrometallurgical treatment of the calcine, will continue. If we can duplicate the A.C. Co's results, and if these results can be readily obtained by practicable means, the beneficiation of the low grade Cerro Copper ores would be enhanced considerably.



To effect dissolution of the cuprous oxide (assuming that the presence of  $\text{Cu}_2\text{O}$  is responsible for our poor results), tests are now under way employing a ferric sulphate-sulphuric acid lixiviant and a long leaching period.

We take this opportunity to thank Messrs. Kentro and Davis of the American Cyanamid Company for their excellent report.

Yours truly,

Theo R. Wright

cc: BTC

New York, December 28, 1928

MR. C. V. DREW:

The following is a preliminary estimate of possible saving at Cerro de Pasco properties by milling the copper ores rather than smelting direct as now practiced.

Assuming a similar tonnage of ore as was smelted in 1927 but excluding the Excelsior ore, that due to its auriferous pyrite, can not be concentrated to as high a degree, a 2000 ton a day mill will suffice. This mill could be built at an outside cost of \$2,000,000, or \$1,000 a ton day capacity.

Judging by previous mill operations at Morococha the ratio of concentration should be 4 into 1 on a 5% ore. This with a tailing loss no higher than the present slag loss - e.g. .32% Cu. and .56 oz. in Ag (1927 figures). The tonnage of tailings so produced would be 75% of the total, equaling the tonnage of slag now being produced.

Estimated costs for milling, freight and smelting each reduced to ton of original ore are:

Morococha - 5% Ore Milling:

Concentration	\$	1.00
Freight @ \$.95 per ton		.24
Smelting @ \$2.25 per ton		.57
Total per ton of crude ore	\$	<u>1.81</u>

5% Ore Smelted:

Freight	\$	.95
Smelting		2.25
	\$	<u>3.20</u>
Saving by milling	\$	1.39

Cerro de Pasco 5% Ore Milling:

Concentration	\$	1.00
Freight @ \$1.65		.42
Smelting @ \$2.25		.57
	\$	<u>1.99</u>

Smelting Direct:

Freight	\$	1.65
Smelting		2.25
	\$	<u>3.90</u>
Saving by Milling	\$	1.91

Aside from the above further additional benefits will be felt in the form of

1. Cheaper mining costs
2. Relief from smoke troubles
3. Lower smelting costs due to high grade initial charge
4. Increased smelter capacity
5. Increased ore reserves by making profitable ores now abandoned or used as stope fillings

The estimated savings do not include the slag loss from smelting the higher grade concentrates that in my opinion can be more than offset by reducing the tailing losses from that shown. This has been the experience at Anaconda where their actual poundage of copper lost as been decreased by adoption of selective concentration before smelting.

In reference to increased ore reserves a copper ore of 1.5% Cu. would show a profit provided mining costs could be kept below \$1.00. This would likely be possible at Morococha rather than at Cerro. Assuming copper in the form of 20% Cu. concentrates to be worth 10 cents a pound with copper selling at 15 cents the following estimate would hold:

Milling	\$1.00
Amortization	.42
Total	<u>\$1.42</u>

1.50% ore      Cu. @ 10 cents -

    .90% recovery                      - \$2.70

Total net from which mining

    costs must be deducted is \$1.28

CPL/G

Quiulacocha Mill, Nov. 29, 1928

Mr. Harold Kingsmill, General Manager,  
Cerro de Pasco Copper Corporation,  
Lima, Peru.

CERRO DE PASCO LEAD-ZINC ORES

Dear sir:

Seventy six samples of this class of ore have been tested both singly and as composites by selective flotation for the separation of the values contained.

Without exception all samples showed signs of either one or both of two types of oxidization.

FIRST:- Impregnation with soluble salts, sulfates of zinc and iron predominating.

SECOND:- Decomposition of the gangue minerals with a resulting excess of primary slimes. In the second instance oxides of the metals contained were also shown to be present on microscopic examination indicating high heat decomposition.

The samples in most cases having been taken from old stope faces possibly show a higher degree of decomposition than would normally be expected had they all been taken from actively producing mine. In view of this fact it is reasonable to assume that when the mine is in production most of the difficulty encountered in these tests will be automatically eliminated. The second type of oxidization has proven to be the chief cause of difficulty in treating these ores. On checking the source of examples and correlating the age of exposure of the faces from which samples were taken the results obtained from tests were in respect to recovery and grade in direct ratio to the age of exposure, indicating that the degree of oxidization of the ore both as to sulfides and gangue minerals was directly responsible for metallurgical results obtained. This fact is again bourn out by the deterioration of the samples in the laboratory in that the ore crushed to minus 10 mesh after standing for four weeks shows practically no possibility of separation on samples that have given commercial results when treated within a few days after being taken.

As a concrete example of the above the samples from the 300 level, numbers 55 to 69 inclusive all showed signs of prolonged or high heat decomposition due to this section having been opened for a period of years. A typical test on a sample from this level shows the effect of its resulting high primary slime content that militates against a commercial separation of values viz:-

	A S S A Y S							R E C O V E R I E S					
	%Wt.	%Cu.	Oz.Ag.	%Fe.	Ins.	%Pb.	%Zn.	Cu.	Ag.	Fe.	Ins.	Pb.	Zn.
Heads	100.0	.05	2.50	19.7	26.2	5.7	14.5	100	100	100	100	100	100
Pb. Conc	10.2	.04	10.4	15.9	6.8	33.2	10.2	20.0	42.4	8.2	2.6	59.3	7.2
Zn. Conc	30.2	.08	2.7	13.2	11.8	3.4	37.5	40.0	32.4	20.2	13.6	17.9	78.2
Tails	59.6	.04	1.06	23.6	36.8	2.2	3.6	40.0	25.2	71.6	83.8	22.8	14.6

In direct contrast to the above the freshly opened ore on the 200 level, where the effect of prolonged exposure of the faces has not been felt but where oxidization in place would be expected to a more pronounced degree, gave the following typical separation on the same class of ore:-

	ASSAYS						RECOVERIES						
	% Wt.	% Cu.	Oz. Ag.	% Fe.	Ins.	% Pb.	% Zn.	Cu.	Ag.	Fe.	Ins.	Pb.	Zn.
Heads	100.0	0.06	4.1	24.2	16.3	5.1	11.4	100	100	100	100	100	100
Pb. Conc	9.0	0.10	29.8	16.8	2.0	44.0	5.5	14.5	65.5	6.2	1.1	77.9	4.3
Zn. Conc	21.0	0.12	3.0	11.5	4.8	2.3	25.5	40.5	15.4	10.0	6.2	9.8	84.0
Tails	70.0	0.04	1.14	29.0	21.6	.9	1.8	45.0	19.1	83.8	92.7	12.3	11.7

You will note the difference in recoveries of values as well as the high recovery of insoluble in the first instance indicating that the higher primary slime content had interfered with proper selection and recoveries.

To cite on extreme case of the effect of this decomposed gangue material the following assays from a test on sample number 67 that had been standing for two weeks in the laboratory showed little separation:-

	% Cu.	Oz. Ag.	% Fe.	% Ins.	% Pb.	% Zn.
Pb. Conct.	.24	11.6	24.1	6.8	14.3	13.3
Zn. Conct.	.20	6.5	13.0	21.6	8.6	25.0
Tails	.10	2.0	10.2	63.2	3.4	2.7

Here again you will note the high insoluble content of the concentrates produced, due solely to the primary slimes contained and their effect on the distribution of values.

Briefly the effect of these primary or colloidal slimes is one of absorbing reagents added to the ore pulp and thereby inhibiting their effectiveness in obtaining the separation for which they were added. Due to this fact ores of a high slime content that show little or no decomposition of the sulfides are often of no commercial value because of the impossibility of obtaining a physical separation of the mixed values contained.

In regard to the soluble salts that all Cerro ores appear to be impregnated with suitable reagent combinations have overcome this cause of former difficulty to a large extent. These soluble salts in the ore pulp cause a precipitate of the heavy metal salts formed by the addition of any activating reagents such as Xanthate, Thiocarbonyl, Phosocresylic acid, Sodium Sulfide and the like. Research work has shown that in cases of ores carrying a high sulfate content that a small quantity of sulphuric acid added after the addition of activating reagents again releases the active principle of these reagents and the desired effect is obtained without resorting to an excessive amount of reagents and obtaining the attendant bulk concentrate.

For example of the above, ore sample number 31 showed no separation with the usual reagent combination until treated with 3 pounds per ton of sulfuric acid, after the addition of Xanthate which alone did not prove effective. This change in reagents gave a 91% recovery of lead and 75% recovery of the silver in a rougher concentrate assaying 29.3 Oz in silver, 38.4 % in lead and 6.3% in zinc. This followed by a rougher zinc concentrate assaying 51.5% in zinc and 0.9% in lead, as against a negative commercial recovery of values with out the sulfuric acid.

In all cases where a high sulfate ore was treated the use of sulfuric acid as a re-activating agent was found to be beneficial and to overcome to a large extent the deleterious effect of the contained sulfates.

The removal of these soluble salts by washing was also tried in several cases but with improvement in results so slight, that until this can be tried out under continuous operating conditions in the pilot mill and the economics determined, no recommendation as to this type of treatment can be made.

In classifying the samples and test thereon as to ore bodies there is little difference to be noted as to their origin in this respect. The only relation between metallurgical results and source of the sample has been the time of exposure of the face from which the sample was taken. This can be regarded as a temporary condition and in my opinion the poor results obtained on the samples from long exposed faces discounted. In pilot mill testing on car lots this condition will be, to a large extent, eliminated in that a tonnage sample will involve going into the ore body to a greater depth.

Another condition that has been apparent in testing these ores is the relation of the copper content of the sample to the zinc content of the lead concentrate. The higher the copper content of the heads the higher the zinc content of the lead product. This relation is obvious when the effect of the soluble copper salts on the rate of float of the zinc sulfides is considered. Copper sulfate is added during flotation to increase the rate of float of the zinc sulfides but with the a high copper content of a partially oxidized ore there is contained in the ore itself sufficient copper sulfate to activate the zinc minerals to a degree where, unless an abnormal amount of depressant is used, an unusual amount of zinc is floated with the head. For example sample number 75 assaying .54 % in copper gave a rougher lead concentrate assaying 20.3% in zinc while sample number 73 from the same level with head assay of .05% in copper gave a rougher lead concentrate assaying only 4.4% in zinc. These are of course extreme cases used to illustrate the point. In making up composites it was found that a head assay in copper of under .20% did not materially raise the zinc content of the lead product but any material excrease over that quantity resulted in an increased zinc content. This condition I again regard as temporary, in that on penetration into the orebodies on active production, the decrease in oxidization of the copper minerals will eliminate the high percentage of copper sulfate in this class of ores.

Testing on the 76 samples submitted was carried out under standard conditions E.G. grinding in steel ball mill to 80% -200 mesh and treating in a suba type flotation machine with standard reagent conditions in preliminary tests, and varied reagent conditions on composite samples. Composites were made up as sufficient number of samples were received to avoid long standing and deterioration. All known successful reagent combinations were tried on the composite samples as well as successful processes now in commercial use. Of these tests not all were assayed as the character of products produced evidenced the failure of the treatment employed. A total of some 200 odd tests were made of which some 100 were assayed for record.

This survey of reagents and types of treatment resulted in the adoption of a treatment scheme similar to that employed at Casapalca with the exception of the use of acid as before noted.

Some trouble was experienced in the control of the lime alkalinity of the zinc circuit but with the recommended use of the P.H. control apparatus in the pilot mill this difficulty should be eliminated. The Kraut machine was tried out on several tests but did not show any advantage in small scale testing over the M.S. suba machine used on the major portion of the tests. There has been a Kraut unit ordered for the pilot mill to further investigate the possibilities of this apparatus. My opinion is that this machine could be used to advantage as a cleaner due to its nicety of air control.

With few exceptions all tests were made with only rougher products shown. The reason being that under proper selective conditions a commercial product should be obtained in a rougher concentrate without resorting to several recleanings to prove the effectiveness of a process of this kind. This also eliminates the question of distribution of values in the middlings resulting from retreatment. The treatment of these ores in the pilot mill will, due to retreatment, show a higher grade finished product and eliminate any question as to distribution of middling products. A typical test on a composite sample that in my opinion can be used as a guide to the grade of concentrate that can be expected gave a finished lead concentrate assaying .68% in Cu., 22.8 Oz. in Ag., 8.9% in Fe, 1.2% in Insol., 55.8% in Pb, and 8.3% in Zn. This was followed by a zinc concentrate assaying .28% in Cu., 2.6 Oz. in Ag, 10.8% in Fe, 1.2% in Insol, 1.1% in Pb, and 55.5% in Zn. In both cases the rougher products were retreated once with no excessive values contained in the middling products indicating that no loss in recovery would result from retreatment to obtain the grade products shown. Excluding the excessively oxidized samples, that, unless taken from the Pacos-Sulfide contact zone, only represent a small tonnage (a few feet in depth of the present faces) the freshly mined ore as represented by eighty percent of the samples submitted should under proper treatment show the following metallurgical results:-

	A S S A Y S :						R E C O V E R I E S :				
	% Wt.	% Cu.	Oz. Ag.	% Fe.	% Pb.	% Zn.	Cu.	Ag.	Fe.	Pb.	Zn.
Heads	100.0	.15	5.0	20.0	8.0	16.0	100	100	100	100	100
Pb. Conc	12.5	.60	25.0	10.0	50.0	9.0	30.0	60.0	6.0	80.0	5.0
Zn. Conc	24.0	.25	5.0	10.0	1.5	53.0	35.0	20.0	10.0	7.0	85.0
Tails	63.5	.08	1.3	28.0	1.3	2.0	35.0	20.0	84.0	13.0	10.0

On the basis of the estimated tonnage taken from the 1928 report of ore reserves including both in sight and probable ore there is 1,822,941 tons of ore of the grade shown. Discounting this figure by 20% to exclude the tonnage represented by the badly oxidized samples there remains some 1,450,000 tons of high grade ore that can be assumed to yield the products shown. This would give on treatment 194,000 tons of lead concentrate and 348,000 tons of zinc concentrates of the grades shown. My understanding is that these ore reserves have been considerably increased by the past year's development and that all the present reserves are the result of Copper ore development so that any active campaign for this type of ore would result in a very material increase over these figures.

In regard to treatment costs the fact that no unusual quantity of reagents or method of treatment has been employed I can see no reason for any increase in direct milling cost over \$1.25 per ton when modern milling methods are considered and the tonnage milled maintained at 1,000 tons a day or better. This figure of course does not include the indirect costs such as amortization and the other usual indirect operating expenses.



Considering a 1,000 ton mill and applying costs on similar plants in Mexico, a preliminary figure for a plant of this size, excluding the necessary trackage, houses, etc., would be \$1,200,000 U.S.Cy.

CONCLUSIONS:-

- (1)- The Cerro Lead-Zinc ores offer no difficulty in treatment by proper selective conditions aside from their partially oxidized condition that in all probability is superficial in extent.
- (2)- The effects of the contained soluble salts can be overcome by proper reagents.
- (3)- The primary slimes contained in the highly oxidized ores prevent these ores from yielding commercial products. Such ores should be excluded from ore estimates and be regarded as waste until the extent of this character of oxidization can be determined.
- (4)- Long exposure of ore before treatment results in decreased recovery and grade.
- (5)- Mining methods in view of the above, should be employed that will have a minimum of ore in the stopes at all times.
- (6)- Pilot mill tests on Cerro ores will be required to determine the distribution of intermediate or middling products as well as grade of final products.
- (7)- Tonnage samples for pilot mill tests should be taken with a view of penetration of the ore body to overcome the superficial oxidization of the exposed faces and to determine the depth of such deterioration.
- (8)- The tonnage of ore now developed would warrant consideration of a mill at least 1,000 tons per day.
- (9)- Due to the relatively high rate of decomposition of these ores delay in their treatment results in an ultimate economic loss.

A complete tabulation of the samples tested is attached hereto.

Very truly yours,

Dec. 4, 1928

Mr. Harold Kingsmill,  
General Manager,  
Lima.

Dear sir:

CASAPALCA CONCENTRATES

In referense to the separation of the lead and copper values contained in the above product, a number of tests have been made with this in view and while a separation of the Chalcopryrite and Galena were made, the assays of the ultimate products did not show the segregation of the values desired.

The mixed concentrate now produced at Casapalca does not contain over 40% of its copper as Chalcopryrite. The remaining 60% is present in the form of Complex silver-copper-lead minerals that due to their chemical combination cannot be separated by any physical process.

The ore now being developed from the lower tunnel is showing a higher percentage of copper as Chalcopryrite and at the time this ore is treated there may be a possibility of a separation of the major part of the copper from the lead present as Galena.

The type of separation desired on this concentrate, and as practiced at Utah Apex, is based on the retarding action of cyanide on the iron contained in the copper-iron minerals (Chalcopryrite). Copper present in other chemical combinations is not affected so that this treatment will not apply to sulfides of copper and lead other than Chalcopryrite and Galena.

I would recommend that this treatment be again considered when lower level ores are to be treated.

Very truly yours,

Page No.4,  
July 30th.,1928.

CASAPALCA MILL

This mill was visited during the first and the latter part of the month during both visits suggestions as reagent were made. Sufficient data as to results from these changes is not as yet available for quoting any exact figures as benefits derived. However indications from several shifts assays point to the possibility of obtaining the following separation:

	% Cu.	Oz. Ag.	% Fe.	% Insol.	% Pb.	% Zn.
Heads	.90	22.0	9.0	50.0	6.0	9.0
Pb. Concts.	9.00	230.0	4.0	3.5	50.0	7.0
Zn. Concts.	.60	6.0	2.5	1.5	1.5	60.0
Tailings	.12	1.2	10.0	65.0	0.6	0.8

By the substitution of soluble reagents for progressively soluble or only partially soluble chemical reagents it is quite possible to overcome the effect of and change in temperature of the mill circuit, without resorting to heating of the pulp. Xanthate substituted for This has to date given as good or better results on the 12 to 8 shift as on the other two shifts indicating the loss in effectiveness of. This was due to its decreased solubility in the colder circuits whereas Xanthate being added as a solution suffered little or no ill effects due to drop in temperature.

In regard to the separation of copper and lead contained in the Casapalca concentrates; tests have been made at Quiulacocha but with negative results to date due primarily to lack of equipment to give the proper preagitation period required for this treatment. Suitable apparatus is being made at Oroya and on its completion further investigation along this line will be made. This copper-lead separation or more correctly speaking Chalcopyrite-galena separation as practiced at Utah-Apex was developed by myself in 1925 using a circuit strong in cyanide to depress the chalcopyrite while floating the galena with proper reagents. It occurs to me that the use of a strong cyanide circuit in treating the Casapalca concentrates may result in a solution loss of the contained silver that would more than overcome the advantage gained by the base metal separation. Another point to be determined is the behavior of the complex copper-lead minerals contained on these concentrates. The lead concentrate produced on the 12-8 shift of July 26th. shows 9.30% copper and 3.4% Fe. indicating some 60% of the copper present in minerals other than chalcopyrite. The behavior of the complex copper-lead and copper-silver minerals can only be determined by experimentation and will be reported on as soon as apparatus is available. However I do not expect as complete a separation of base metals as would be possible with a concentrate composed of chalcopyrite and galena alone.

Page No.5.  
July 30th.,1928.

In reference to both mills now in operation I would suggest that recoveries be reported as economic recoveries rather than the present form of recovery reports where grade of product produced is not taken into consideration. Economic recoveries figured on fixed metals prices of values in the ore divided into metals recovered figured at the same base values under terms of smelter contracts will give a better index of mill operation than the present method now in use.

Very truly yours,

C. P. Lewis.

Oreya,  
Oct 3rd 1928.

Mr. Harold Kingsmill, General Manager,  
Cerro de Pasco Copper Corp.  
Lima.

Dear Sir:-

GASAPALCA MILL.

Herewith is my report on the above mill both as to metallurgical progress made to date as well as recommendations for future improvement.

COURSE CRUSHING: The present practice of crushing to approximately two inches for the ball mill feed is to be condemned because of the resulting high fine grinding costs in small diameter mills. Mills of the size used (64-1.2<sup>x</sup> Marcy) in the primary grinding section should have a feed no larger than 1/2" inch. The present duty of this mills can be performed at under half the cost by crushing dry to 3/8 or half inch. To correct this I would recommend that a 3 or 4 foot Symonds Cone Crusher be installed ahead of the present elevators that discharge into the mill fine ore bins. The two gyratory crushers now installed there are not in use and can be removed to provide space for this equipment. A 3 foot cone would give the required capacity but a four foot cone would be recommended to provide the margin of safety in capacity that is to be desired in case of breakdowns and future increase in mill capacity. The cost of this installation would probably be approximately \$10,000.00 and the savings resulting therefrom would return the cost within twelve months.

FINE GRINDING AND CLASSIFICATION : Due to the fact that this mill has been remodelled several times the location as well as types of machines in this department are not all that could be desired.

FIRST: The primary mills using 5 inch balls are doing work more economically accomplished by up to date dry crushing.

SECOND: The rod mill used as a secondary grinder is being run under its normal efficient operating load. This causes metal to metal contact with high steel consumption as well as over sliming.

THIRD: The Allis Chalmers mill section due to small size of the units and the small classifiers are high steel consumers due to lack of circulating load passing through them.

FOURTH: All classifiers on hand and in use lack the depth that is necessary to provide classification without too much dilution. The present overdilution of the classifier overflow has been found necessary to insure proper grinding but in return resulted in a shortening of the treatment period by reason of larger volumes of pulp handled.

As a remedy for the above I would recommend that the second large classifier on hand be installed near and slightly above the rod mill classifier. This second unit to receive the discharge from the two primary ball mills after they have been converted into fine grinding units by reducing the size of the feed to them as well as the use of 3 inch balls rather than 5 inch. The sands from both classifiers to return by gravity to the rod mill and constitute its feed. This will eliminate the entire Allis Chalmers mill section at the present tonnage rate or provide for an increased tonnage of 100 to 150 tons per day should this be desired, as well as reduce steel consumption and overdilution of the greater portion of the pulp as obtained.

LEAD FLOTATION CIRCUIT: The lead circuit consisting of Inspiration type Gallow machines both in the rougher and cleaner sections is producing good results from a metallurgical standpoint but with high costs both as to power and repairs.

No unit costs are available so exact figures can not be quoted. The power consumed is approximately 40% above what would be required by mechanical machines producing equivalent results, altitude considered. Estimated savings here would not justify any change when the age of the mill as well as possibility of a new mill on a different site are considered.

ZINC FLOTATION CIRCUIT: From a metallurgical and operating standpoint there is little fault to be found with this section. Under proper control excellent results are produced with the exception of those periods when water shortage results in lack of air from the turbo blowers.

PILOT TABLES:- Both these tables are now being run on tailings as pilots for the entire operation. I would recommend that one be used as a pilot on zinc concentrates to more closely control the iron content of this product.

GENERAL:- The flotation circuits as a whole are lacking in proper emulsification or contact of reagents. To remedy this I recommended last July that two 2 cell agitators be built to insure proper agitation of the pulp before it entered the flotation circuits. These I understand are now being built and should be installed sometime within the coming month.

As mentioned in my previous report on this mill various reagent changes have been made that are reflected in comparing the following two tabulations of metallurgical results:-

AVERAGE OF JANUARY TO MAY 1928.

	<u>A S S A Y S</u>				<u>R E C O V E R I E S %</u>			
	<u>%Cu</u>	<u>Oz/Ag</u>	<u>%Pb</u>	<u>%Zn</u>	<u>Cu</u>	<u>Ag</u>	<u>Pb</u>	<u>Zn</u>
Heads	.954	20.75	5.06	8.35	100	100	100	100
Pb Cts	8.62	200.96	43.65	10.34	81.1	88.4	84.4	11.4
Zn Conc	.77	10.44	2.40	52.8	10.0	6.3	6.3	78.8
Tails	.11	1.35	.56	1.05	8.9	5.1	9.2	9.8

AVERAGE AUGUST 21st to 31st 1928

	<u>A S S A Y S</u>				<u>R E C O V E R I E S</u>			
	<u>%Cu</u>	<u>Oz Ag</u>	<u>%Pb</u>	<u>%Zn</u>	<u>Cu</u>	<u>Ag</u>	<u>Pb</u>	<u>Zn</u>
Heads	1.076	21.81	6.51	10.2	100	100	100	100
Pb Conc	8.69	180.46	48.56	10.2	84.2	92.7	89.0	11.1
Zn Cts	.60	4.04	.90	59.5	7.2	2.6	2.0	80.3
Tails	.13	1.38	.74	1.2	8.6	4.7	9.0	8.6

You will note in the above that while the final tailing is about the same the distribution of the values is better by several percent.

I find from my visits to this plant that the chief difficulty encountered in maintaining the best results, lies in the fact that very close operation is required and the personal factor plays too prominent a part on that account. Close attention can produce results at this plant second to none but with the present flow sheet more than usual attention is required. The Casapalca ores are remarkably amenable to selective flotation treatment and they should yield products and recoveries unequalled anywhere.

I have hesitated in recommending any mechanical changes until I was sure that the best results had been obtained with equipment and flow sheet in use on my arrival. The recommended changes contained herein can all be made without loss of mill time and with no greater capital outlay than would be justified considering the life of the mill.



(5)

Trusting you will find the foregoing in order

I remain,

Yours very truly,

G. P. Lewis.

Oroya, Perú,  
July 30th., 1928.

Mr. Harold Kingsmill, General Manager,  
Cerro de Pasco Copper Corporation,  
Casilla 2412,  
LIMA.

Dear Sir:

The following is a brief summary of my activities since my arrival in Perú on June 19th.

The interval between June 21st. and July 6th. was spent in visiting the various camps and becoming familiar with the concentration problems presented by the concentrators now in operation and the prospective problems in the ores now being smelted direct as well as these developed incidental to the mining of direct smelting ores.

#### QUIULACOCHA MILL

I visited this mill from the 6th. to the 22nd. incl. as per your verbal request made while in Lima.

During this interval three major changes were made there. Viz:

FIRST: The flow was changed from a counter current concentrate flow to a concurrent flow for the following reasons: -  
1st. a counter current flow entails the lifting of the more refractory sulfides several times before they can be eliminated as concentrates.  
2nd. a counter current flow results in the inclusion of an excess amount of wash water that results in an over-dilution of the pulp to a point where selective flotation at the lower end of the circuit is impossible. As a case in point of my arrival the density at the head of the circuit was 40% solids by weight with the tailings 7% solids - this with an average elimination of only 20% of the total weight of the ore in the form of concentrates.

To overcome this obvious fault all middling froths and cleaner tailings were returned to the classifiers where the contained water was used as make-up water at that point and to provide the more refractory sulfides with another opportunity to come in contact with fresh flotation reagents.

SECOND: From the screen analysis of the tailings and their assays it was apparent that the most of the recoverable losses were in the # 200 mesh material. The - 65 and # 200 mesh product showing on an average of twice the silver content of the - 200 mesh product. Investigation showed that the primary mills were discharging a partially ground product, the major portion of

Page No.2  
 July 30th.,1928.

which was being overflowed from the classifiers without being sufficiently ground to expose fresh sulfides surfaces. To remedy this defect and to insure sufficient brightening of the sulfides the discharge from the primary mills was sent direct to the secondary mills without classification. This apparently not only accomplished the desired result but in addition corrected the under-loaded condition of the secondary mills that has undoubtedly caused excessive steel consumption as well as trouble in maintenance of these units.

THIRD: Reagent changes were made during the first 10 days in an endeavor to both increase the recovery and grade of concentrates as well as reduce the excessive quantities of reagents used. This work was handicapped to some extent by an occasional erroneous sample so that several days were lost before an accurate estimate of reagent functions could be made. It was found that in treating the partially oxidized dump ores from the Colquijirca mine that any material increase in grade of product was obtained at an unwarranted sacrifice in recovery. That is a 60% lead concentrate was accompanied by an 8 to 10 oz. silver tailing.

With the above in mind the reagents were changed to give a concentrate of from 45 to 50% in lead at which point it was apparent no sacrifice in tailings was necessary and recoveries materially increased over previous operations on this same class of ore. The grade quoted can be increased by careful operation under supervision of qualified shift bosses. Results during the month of July can be said to have suffered to some degree due to lack of this class of supervision thru out the 24 hours of the day.

To summarize, the following are the average results for June when treating the dump ore, including five days after my first visit there where a minor reagent change resulted in an approximate increase of 10% over former recovery:-

A S S A Y S				
	% Cu.	Oz Ag.	% Pb.	% Zn
Heads-	.39	17.6	11.1	6.7
Concs.	1.26	70.0	44.2	14.8
Tails	.14	5.9	3.2	3.7

RECOVERIES BY FORMULA

Cu.	72.2%
Ag.	72.7%
Pb.	76.5%
Zn.	59.8%

Page No.3.  
July 30th.,1928.

After the 18th. and until my departure while a definite reagent program was adhered to and after all the changes had been completed the average results for 18 shifts were as follows:-

	% Cu.	A S S Oz. Ag.	A Y S % Pb.	% Zn.
Heads	.33	16.4	11.6	6.4
Concns	1.20	66.6	44.5	13.2
Tails	.05	3.7	1.6	3.2

## RECOVERIES BY FORMULAE

Cu.	88.5%
Ag.	82.0%
Pb.	89.3%
Zn.	60.8%

As far as I have been able to learn since leaving this plant the results to date have of the same order as above with some variation in grade of concentrates due no doubt to changes in ore and also an occasional high tailing due to shut downs because of mechanical difficulties and power interruptions.

In comparing these figures you will note the similar grade of concentrate produced with a reduction of 2.2 Oz. of silver in the tailings as well as a 1.6% reduction in lead.

As to reagents the following table will serve to illustrate the reduction in quantity per ton of ore treated under conditions prevailing on July 7th. and those under which the last quoted results were obtained:

Reagents July 7th.		Reagents July 19th.
Thio-carbanalide	.313.....	None
Zinc sulfate	3.639.....	None
Sodium Cyanide	.847.....	.231
Hardwood Creosote	.113.....	.228
Pine Oil	.165.....	.045
Xanthate	.378.....	.380
Soda Ash	12.564.....	1.50
Black Ash	1.431.....	None
Petroleum	.015.....	0.05

It is to be expected that further familiarity on the part of the operators with this changed style of operation will result in a gradual further betterment both as to grade and recovery and that these changes will also result in improved recoveries and grade when treating the mine ore during the ensuing month.

# CERRO DE PASCO LEAD ZINC ORE SAMPLES ASSAYS.

Nº	ORE BODY	LEVEL	SAMPLE DISTANCE	PLACE	%Cu.	A <sup>oz</sup> .g.	%Fe.	%Pb.	%Zn.
1	Lourdes-Mesapata.	800	49 x 98 Feet	8216-N	.04	10.8	31.2	6.6	23.5
2	Mesapata.	800	187 x 187 "	8205-E	.16	5.0	28.4	4.7	12.5
3	"	800	480 x 480 "	8212-E	.04	3.7	24.9	5.5	9.8
4	Lourdes	600	72 x 72 "	6607-E	.02	1.1	18.9	2.8	10.4
5	"	600	90 x 90 "	6612-N	.01	4.3	25.0	4.3	8.5
6	"	600	189 x 378 "	6711-W	.04	7.2	22.3	5.6	8.4
7	Mesapata.	600	80 x 160 "	6604-N	.06	5.6	39.8	.7	1.5
8	Lourdes	600	210 x 420 "	6674-E	.02	4.9	18.4	9.6	18.8
9	"	600	220 x 440 "	6677-E	.02	.9	26.3	3.5	11.2
10	"	600	160 x 320 "	6708-E	.06	6.7	23.4	5.1	13.2
11	"	600	140 x 280 "	692-N	.24	4.4	28.2	4.8	10.7
12	Mesapata	600	280 x 560 "	6672-N	.04	3.3	28.4	4.7	15.6
13	"	600	150 x 300 "	6645-E	.02	3.2	25.6	6.5	16.4
14	"	600	90 x 180 "	6645-E	.24	1.3	19.7	6.9	13.9
15	"	600	130 x 260 "	6694-N	.40	3.1	16.5	4.5	9.0
16	"	600	150 x 300 "	6682-E	.14	2.0	24.5	9.7	14.3
17	"	600	70 x 70 "	692-N	.06	4.1	20.1	8.4	14.1
18	Lourdes	500	90 x 180 "	519-E	.06	2.0	10.6	8.6	38.5
19	"	500	166 x 332 "	520-N	.08	3.8	14.9	8.6	30.7
20	"	500	170 x 340 "	547-N	.08	6.0	12.2	8.4	26.3
21	"	500	170 x 340 "	540-W	.08	5.8	13.6	2.5	6.7
22	"	500	170 x 340 "	542-N	.10	5.8	11.8	9.5	34.2
23	"	500	80 x 160 "	545-W	.16	5.5	11.1	7.3	35.6
24	"	500	120 x 240 "	546-W	.30	7.1	17.1	5.0	15.4
25	"	400	130 x 260 "	4435-E	.12	7.9	19.4	4.7	19.4
26	"	400	90 x 90 "	404-N	.04	5.1	14.4	9.7	29.5
27	"	400	55 x 55 "	4447-E	.08	0.7	8.5	2.6	4.4
28	"	400	70 x 70 "	4462-E	.30	2.0	6.4	3.8	7.6
29	"	400	80 x 80 "	4430-S	.16	2.3	20.0	3.2	8.0
30	"	400	114 x 114 "	4441-N-S	.12	7.1	24.6	4.7	8.8
31	"	400	80 x 80 "	4440-N	.12	10.6	21.3	11.7	15.5
32	"	400	80 x 80 "	4451-W	.08	5.7	21.2	10.1	14.3
33	"	400	110 x 110 "	4445-N	.10	4.5	25.8	5.6	17.3

Nº	ORE BODY	LEVEL	SAMPLE DISTANCE	PLACE	%Cu	Ag. <sup>oz</sup>	%Fe.	%Pb.	%Zn.
34	Mesapata	400	50 x 100 Feet.	404-N	.06	4.15	14.8	18.0	6.2
35	"	400	155 x 310 "	4472-E	.06	4.30	12.6	10.7	23.7
36	"	400	165 x 330 "	404-N	.06	1.85	14.9	6.3	12.0
37	Diamante	400	250 x 500 "	4507-N	.04	1.70	27.6	3.9	10.7
38	"	400	75 x 150 "	R-1-E	.42	5.00	16.9	5.5	30.6
39	"	400	123 x 123 "	R-2-N	.10	3.30	20.6	3.4	24.9
40	"	400	115 x 115 "	R-90-SW	.06	5.50	20.1	8.9	19.7
41	Sn. Alberto	400	35 x 70 "	R-122-E	.06	2.10	23.7	2.2	14.8
42	" "	400	15 x 30 "	R-2-N	.08	3.20	26.0	4.9	12.4
43	" "	400	50 x 100 "	R-140-N	.08	2.10	20.2	3.3	19.8
44	" "	400	20 x 20 "	R-142-E	.60	4.10	25.5	3.3	6.5
45	Diamante	400	95 x 190 "	4492-N	.08	2.7	24.5	7.3	13.4
46	"	400	175 x 350 "	402-W	.06	4.8	17.0	8.0	26.1
47	"	400	210 x 420 "	411-N	.10	6.0	13.9	6.9	32.2
48	"	400	120 x 240 "	410-N	.02	3.3	27.3	5.8	12.8
49	"	400	92 x 184 "	409-S	.04	5.4	17.6	11.2	16.5
50	"	400	90 x 180 "	4466-N	.08	23.4	29.9	8.9	6.3
51	"	400	21 x 42 "	4504-W	.06	2.9	26.9	5.5	14.8
52	"	400	55 x 110 "	404-S	.04	1.6	30.8	3.4	7.0
53	"	400	45 x 90 "	404-S	.34	9.8	20.9	4.3	10.3
54	"	400	55 x 110 "	404-W	.16	4.1	16.8	10.7	23.8
55	Lourdes	300	43 x 43 "	3544-S	.14	3.1	21.0	6.4	13.8
56	Mesapata	300	100 x 200 "	3534-E	.04	1.9	20.0	5.5	13.3
57	"	300	23 x 46 "	3527-S	.10	0.8	18.7	7.6	12.1
58	"	300	39 x 78 "	3519-E	.06	3.0	28.2	6.3	11.4
59	"	300	67 x 134 "	328-W	.02	3.8	25.0	4.9	8.5
60	Diamante	300	70 x 140 "	3432-N	.04	2.5	16.1	7.4	22.5
61	"	300	70 x 70 "	320-N	1.30	2.2	22.0	4.2	8.1
62	"	300	90 x 90 "	302-W	.86	3.1	25.6	6.5	11.9
63	"	300	170 x 170 "	3415-S	.06	4.3	22.3	8.8	15.6
64	"	300	95 x 95 "	302-W	.16	7.8	19.7	13.0	19.2
65	"	300	98 x 196 "	381-E	.84	9.0	19.9	12.6	16.2
66	"	300	110 x 220 "	382-W	1.10	4.8	20.0	8.4	21.7
67	Lourdes	300	W-20'E-35' x 55'	3544-N	.08	4.1	11.7	6.1	6.2

Nº	ORE BODY	LEVEL	SAMPLE DISTANCE	PLACE	%Cu	Ag. <sup>oz</sup>	%Fe.	%Pb.	%Zn.
68	Lourdes.	300	E 35' W 40' x 75 Feet	3544-S	.08	4.5	20.8	6.5	9.5
69	Jr. Alberto.	300	15 x 30 "	3546-E	.64	7.2	25.7	3.6	11.1
70	" "	200	40 x 40 "	296-W	.48	1.3	29.2	1.3	5.5
71	" "	200	45 x 90 "	296-N	.12	2.0	21.9	2.0	14.9
72	Diamante	200	100 x 200 "	210-N	.12	4.0	26.6	4.2	10.1
73	"	200	N 35' S 25' x 60 "	226-E	.04	3.7	20.0	7.4	14.2
74	"	200	E 78' W 25' x 103 "	2249-N	.04	3.9	24.4	5.3	17.1
75	"	200	70 x 70 "	2249-N	.48	2.6	25.1	7.0	10.7
76	"	200	25 x 25 "	2302-S	.24	1.5	26.6	5.5	17.6
	Composite A				.06	5.3	20.3	7.4	19.5
	" B				.18	4.6	20.8	7.0	15.3
	" C				.12	4.5	21.8	7.3	17.2
	" D								
	Sands #57 - 83%				.22	2.5	19.8	7.2	13.2
	Slimes #57 - 17%				.34	4.5	16.5	6.5	11.4

RESEARCH DEPARTMENT  
MONTHLY NEWS LETTER  
OCTOBER 1929

Ore Dressing Section  
Report for September and October

Mr. B. T. Colley,  
Metallurgical Engineer,  
Oroya.

Dear Sir:

Due to illness, no report of operations was made last month. This report, therefore, covers the work done from the first of September to date.

Pilot Concentrator

Aside from a loss of twelve days in September, during which time the carpenters were occupied on more urgent work, the construction of the pilot concentrator has progressed steadily.

The crushing and grinding units are practically completed. Bins, coarse ore feeders, jaw crusher, cone crusher, ball mill, and classifier installations are finished.

The flotation sections are well advanced. The 14-cell M.S. machine is now being equipped with its agitator mechanism and the last of the Forrester machines is being assembled. There are yet to install the 3-cell Kraut and the 10-ft. Mackintosh machine. The latter has not been received from the States to date but its arrival is expected during the next month. The Kraut machine was shipped completely assembled so that it will only be necessary to place it on its respective platform. These two machines will act as double cleaners following the M.S. machine.

The thickening and filtering units are finished except for piping, and the unit for the conditioning of the lead and zinc pulps is nearing completion. The only major machine installation to be made is that of the two low pressure blowers together with their connecting bustle pipes. The minor installations yet to be made consist of pumps, pilot tables, sampling equipment, the Mackintosh and Kraut flotation machines mentioned above, ore feeder from fine bins, reagent feeders, short belt conveyor, and line shafts. Due to the lack of sufficient head room, and the desire to make the sampling of the mill heads as fool proof and accurate as possible, a sampler of the bucket type will be substituted for the Vezin sampler which was to be installed originally.



The electricians have installed the transformers and are now putting in conduit for motorleads. All motors over 10 H.P. will be hooked-in direct to the transformer bank with individual leads. The smaller motors will be connected to one of two distributing boxes. Provision has been made for the possible future addition of several motors of varying horse power. It is expected that all of the wiring will be completed before the first of the year.

There is considerable piping to be done due to the large number of pumps required. This work will be started immediately, and, since the Bodaga has just received a new stock of fittings, this job should progress rapidly.

There are about 140 cubic yards of concrete yet to be poured. Ninety percent of this amount, however, is for the floor and will not be poured until all of the conduit and other sub-floor connections are made.

Although the minor details of construction and installation will consume considerable time, it is still expected that the mill will be in operation by the first of the year.

No plans of the general layout of the entire mill exist, inasmuch as each unit has been designed and built separately and progressively in order to avoid delay in construction. There is now making a plan and elevation of the entire mill showing the respective sizes and locations of all machines. These drawings will be annotated to explain pertinent features and to indicate capacities, pulp flow, and optional variations.

Ore Testing

As the pilot mill is nearing completion, much of the time of the staff is required for attention to details of installation. As a result, it has not been possible to devote as much study to ore investigations as desired.

Cerro lead-zinc ores. Attention is called to the general news letter for August in which the average total extraction of zinc from the eleven samples of Cerro ores was reported as 74.3 per cent. In making this calculation it would appear that in some tests the tabulations of assays were confused with the tabulations of recoveries. The average results from the eleven samples, both good and bad, have been recalculated, and the following results:

Average total extraction of zinc ..... 93.4%  
(Lead concentrate plus zinc cont.)

Rougher Zinc Concentrate  
Average Assays

Ozs Ag	% Pb	% Zn	% Fe	% Insol.
3.3	2.2	44.6	13.8	2.9

Rougher Zinc Concentrate  
Average Distribution

% Ag	% Pb	% Zn	% Fe	% Insol.
34.5	10.5	82.9	44.5	15.4

The zinc recoveries are actually higher than those of silver or lead, whether total or economic recoveries are considered.

Quiulacochoa tailings. An investigation was made during September as to the cause of the high mill tailings that were prevalent at the Quiulacochoa Concentrator. Preliminary tests on tailings indicated quite conclusively that the ore was not being ground sufficiently. There has been devised a most efficient and economic regrinding scheme, and, as a consequence, the metallurgical results have materially improved.

The following is an analysis of the lead tails as of September 21st and 22nd.

	Assay			
Ozs. Ag	% Pb	% Zn	% Fe	% Insol.
8.2	2.9	10.9	7.8	58.8

A wet screen analysis made on this sample gave the following results:

	Mesh	Weight	Percent	% Cumulative
Retained On	48	27	2.0	2.0
" "	65	60	4.3	6.3
" "	100	122	8.8	15.1
" "	150	131	9.5	24.6
" "	200	114	8.3	32.9
Through	200	926	67.1	--

Mesh	Assay			Content		
	Oz. Ag	% Pb	% Zn	Oz. Ag	% Pb	% Zn
48	3.7	1.0	2.2	.07	.02	.04
65	7.1	1.6	4.7	.31	.07	.20
100	13.1	3.3	7.7	1.15	.29	.68
150	20.5	6.1	10.3	1.95	.58	.98
200	17.3	5.9	13.6	1.44	.49	1.13
-200	5.0	2.3	11.8	3.36	1.54	7.91
Head	8.2	2.9	10.9	8.28	2.99	10.94

Mesh	% Distribution			% Cum. Distribution		
	% Ag	% Pb	% Zn	% Ag	% Pb	% Zn
48	.8	.7	.4	.8	.7	.4
65	3.8	2.3	1.8	4.6	3.0	2.2
100	13.9	9.7	6.2	18.5	12.7	8.4
150	23.5	19.4	9.0	42.0	32.1	17.4
200	17.4	16.4	10.3	59.4	48.5	27.7
-200	40.6	51.5	72.5	--	--	--
	100.0	100.0	100.0			

Although 67% of the tailing is minus 200 mesh, about 60% of the total silver and 48% of the total lead is in the plus 200 mesh material.

Aside from the segregation of the valuable minerals in the coarser and consequently less flatable, the microscope reveals lead-zinc (and probably silver mineral) aggregates. In other words, finer grinding is necessary in order to better prepare the minerals for flotation and also to unlock the individual mineral grains.

In regrinding and floating the lead tails the silver was reduced from 8.2 to .7 ounces and the lead from 2.9 to .3 per cent. In this test no effort was made to selectively separate the zinc or to make a high-grade product; the sole aim being to determine the effect of finer grinding. The efficacy of finer grinding is very apparent.

In a later test on the same tailing sample, an effort was made to recover the maximum amount of values in the least possible bulk. The results follow:

Product	% Weight	Assays				
		Oz. Ag	% Pb	% Zn	% Fe	% Insol.
Heads	100.0	8.2	2.9	10.9	7.8	58.8
Ag-Pb Con.	7.7	68.8	26.8	18.8	11.0	14.9
Zinc Con.	20.1	10.8	1.7	45.6	8.1	11.5
Tailing	72.2	1.3	.7	.4	7.5	76.5

Product	% Distribution				
	% Ag	% Pb	% Zn	% Fe	% Insol.
Heads	100.0	100.0	100.0	100.0	100.0
Ag-Pb Con.	62.8	70.8	13.3	10.8	2.0
Zinc Const.	25.7	12.0	83.9	20.7	3.9
Tailing	11.5	17.2	2.8	68.5	94.1

Considering the per cent weight of the silver-lead concentrate, the recoveries in the form of this product are quite good. The

high contamination of zinc, insol. and iron is primarily due to the relatively low content of lead which makes a clean separation very difficult to attain. The silver-lead concentrate could undoubtedly be greatly beneficiated by cleaning.

A test was made at a later date on the final mill tails. Results in the mill had improved somewhat by this time and the tails were appreciably lower. The following tabulation gives the results:

<u>Product</u>	<u>% Weight</u>	<u>Oz. Ag</u>	<u>% Pb</u>	<u>Assays</u>		
				<u>% Zn</u>	<u>% Fe</u>	<u>% Insol.</u>
Heads	100.0	3.2	1.0	1.0	4.6	81.7
Concentrate	3.1	70.6	11.3	13.9	13.2	36.8
Tailing	96.9	1.2	.2	tr	4.2	83.0

<u>Product</u>	<u>% Ag</u>	<u>% Pb</u>	<u>% Distribution</u>		
			<u>% Zn</u>	<u>% Fe</u>	<u>% Insol.</u>
Heading	100.0	100.0	100.0	100.0	100.0
Concentrate	65.4	64.8	99.9	9.1	1.4
Tailing	34.6	35.2	.1	90.9	98.6

The high concentration of silver is of particular interest.

In the above tests the reagent combination was the same as in use at the Quiulacocha Concentrator with the exception of potassium amyl xanthate which was used as the collector. The primary difference in treatment lay in finer grinding. In each case the sample was ground to pass 200 mesh. A secondary difference in conditions, but one which had a marked influence on the grade of lead concentrate produced, was that both tailing samples were contaminated with copper sulphate. The copper sulphate addition is made at the mill prior to the sampling of the lead tails. Naturally there was a strong tendency for the zinc to float with the lead. This accounts for the high zinc content in the scavenger lead concentrate.

As stated previously, finer grinding has now been instituted at Quiulacocha and the results show a marked improvement. Later on, further tests can be made to determine if the ore can be further beneficiated by even finer grinding than is being practiced at present. The economic advisability in this case, however, of extremely fine grinding is rather doubtful.

Production of Cercapuquio Zinc concentrates. The treatment of the lead-zinc-cadmium ore from Cercapuquio for the purpose of producing 100 pounds of zinc concentrate is still in progress. This work should be finished some time this month.

The particular sample that we are treating offers no particular metallurgical difficulties but there are certain physical charac-

teristics inherent in the ore that cause mechanical difficulties. Both the lead and the zinc are very highly disseminated throughout the ore and the crystallization is extremely fine. Most of the zinc shows no crystalline structure whatever but appears to be amorphous. Fine grinding is essential in order to unlock the galena and sphalerite particles, but, in so doing, a great amount of slime is produced. The slimes, composed chiefly of calcium carbonate and amorphous zinc, produce a colloidal pulp that we have been unable to successfully flocculate. As a result, the zinc froth is very thin, voluminous, and difficult to break down. The frothing period is consequently long. Furthermore, the colloidal character of the pulp makes filtration difficult. Roughly, the filtering period is thirty times longer per unit weight when filtering Cercapuquio zinc concentrates than when filtering an average zinc concentrate.

Inasmuch as this sample was purposely taken from a high-cadmium section of the mine, it is probably not representative of the ore body as a whole. Former samples from Cercapuquio showed a high colloidal dispersion but not to such a degree as exists in the sample now being treated. In any case, however, if a mill is ever built to treat this ore the problems of handling a voluminous, sticky froth and a slow filtering zinc concentrate must be kept in mind.

Various measures are being taken to overcome these difficulties, but, on the whole, they have not met with much success. Sodium silicate and lime, the customary flocculating reagents, had no effect in decreasing the colloidal property. Lime, on the contrary, increased the dispersion. Sulphuric acid was tried in various amounts. No appreciable benefit was noticed until 500 pounds per ton of acid were used. With this amount of acid all of the calcium carbonate was converted to calcium sulphate and the pulp acidified. The pulp, due to the production of calcium sulphate, was very thick and the froth more heavily laden. Such a high consumption of acid, however, is not practical mechanically, if economically. The use of acid did not materially reduce the filtering period. The next step was to use a denser pulp. This produced a fairly well laden froth during the zinc float but the stickiness and difficulty of filtration still persisted. Changes in reagent combinations had little or no influence on these troublesome physical features.

This sample of Cercapuquio ore is undoubtedly of higher zinc and cadmium content than the average ore.

	Assay								
	Oz Ag	% Cu	% Pb	% Zn	% Cd	% Fe	% Ins	%CaCO <sub>3</sub>	
Cercapuquio									
Cadmium Special	tr	tr	16.2	37.3	1.13	1.2	1.8	13.9	

A detailed discussion of the metallurgical treatment and results will not be given at this time as there are no definite recommendations formulated. With this sample, since the zinc content is so high, the per cent weight of zinc concentrate is likewise high and a high degree of concentration therefore impossible. The rougher lead con-

concentrate contained a high percentage of zinc but, as in the tests on previous samples (see January report), yielded a 70% lead product in one cleaning operation with very little loss in lead. The lead middling is of sufficient grade (48 to 50% zinc) to be combined with the zinc concentrate. The rougher zinc concentrate, due to the colloidal character of the calcium carbonate, is contaminated to a considerable extent with this mineral. The rougher zinc concentrate assays between 40 and 50 per cent. Cleaning brings the grade up to 52 per cent, the other constituent being chiefly calcium carbonate. The rougher zinc concentrate contains about 90 per cent of the total zinc and 90 per cent of the total cadmium. Disregarding the disposition of the zinc and cadmium in the zinc middling, the cleaned concentrate contains about 85 per cent of the total zinc and 85 per cent of the total cadmium in a product assaying 52% Zn and 1.7% Cd.

Future Test Work: There is a need for a further study of the Cercapuguis ores with the following objectives.

1. To speed up zinc flotation by creating a more armored froth.
2. To decrease the rate of flocculation of colloids.
3. To decrease the  $\text{CaCO}_3$  content of the zinc concentrate since this mineral interferes mechanically and chemically in the leaching process.

These points will be taken up during the current month and an endeavor made to rectify the difficulties.

Further tests will be made on the eleven samples of Cerro lead-zinc ores to determine the effect of time on the rate of oxidation and to determine the metallurgical influence of oxidation.

There are several samples of lead-zinc ore from the Morococha mines which have not as yet been studied.

Respectfully yours,

Theo. R. Wright