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METALLURGICAL TESTS ON SAMPLES OF SILVER ORE

submitted by

PESO SILVER MINES LTD. (N.P.L.)

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Progress Report No. 1

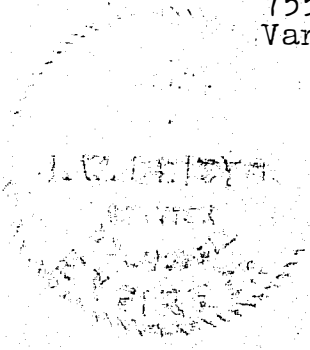
Preliminary Tests on Oxide Ore from No. 1 Vein

Investigation No.: B3

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INTRODUCTION

This report described a series of cyanidation tests carried out on a sample of oxide ore from the No. 1 vein of Peso Silver Mines Ltd. The purpose of the tests was to obtain preliminary information on the silver and gold recoveries obtained when treating this type of ore and to ascertain the nature of the metallurgical problem involved.

The sample used for the tests was part of a 400 pound bulk sample which had been taken at a depth of 25 feet in the shaft and shipped to Vancouver in a sealed drum.

Authorization for the tests to be carried out was given by Mr. S. E. Cropper, Managing Director of Peso Silver Mines Ltd., on January 24th, 1962.

SUMMARY

The sample contained 56.8 ounces of silver and 0.08 ounces of gold per ton, 3.24 % lead, 2.81% antimony, 0.06% copper, 9.40% arsenic, 0.32% sulphur and a trace of zinc.

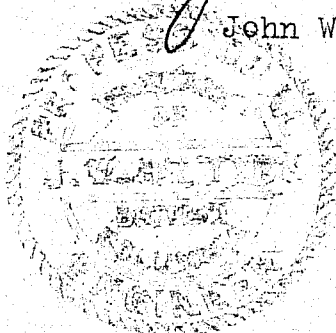
The test results are summarized in the following table:

| Test No. | Treatment time Hours | Silver recovery | | Gold recovery | | Cyanide and Lime consumed - Lb. per ton | |
|----------|----------------------|-----------------|-------------|---------------|-------------|---|------|
| | | % | oz. per ton | % | oz. per ton | NaCN | CaO |
| 1 | 80 | 73.5 | 42.5 | 91.1 | 0.061 | 16.9 | 10.3 |
| 2 | 80 | 68.6 | 40.0 | 85.7 | 0.060 | 15.0 | 18.5 |
| 3 | 80 | 71.5 | 41.2 | 90.3 | 0.056 | 16.2 | 18.2 |
| 4 | 56 | 66.4 | 37.7 | 82.7 | 0.063 | 5.6 | 26.3 |

The silver recovery averaged 70% and the gold recovery averaged 87%. The cyanide consumptions were high in the first three tests but in test 4, by the use of more lime together with aeration of the pulp before adding cyanide, the cyanide consumption was reduced considerably.

Screen analyses of the cyanidation residue from test 4, together with assays of the screen products, indicate that the silver which is not soluble in cyanide may be present in a lead-antimony mineral, probably bindheimite. If this is the case, it may be possible to increase the silver recovery by making a silver-lead-antimony concentrate by flotation or gravity concentration and shipping the concentrate to a smelter.

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SAMPLE PREPARATION

The first stages of sampling were carried out at the Hornby Street, Vancouver premises of Peso Silver Mines Ltd.

The 400 pound sample was first spread on a clean concrete floor and mixed thoroughly by means of a shovel. It was then coned and quartered. One half was returned to the drum and the other half was mixed, coned and quartered. One half was split down by coning and quartering twice to give four equal samples which were retained by Peso Silver Mines. The other half, weighing approximately 100 pounds, was bagged and transferred to the writer's laboratory. It was then partially dried at a maximum temperature of 150°F, mixed thoroughly, coned and quartered. One half was placed in plastic bags and stored and the other half was screened on 3 mesh and 10 mesh Tyler sieves. The screen analysis was as follows:

| | |
|----------------------|--------------|
| Plus 3 mesh | 57.5% |
| Minus 3 plus 10 mesh | 16.5% |
| Minus 10 mesh | <u>26.0%</u> |
| | 100.0% |

The plus 3 mesh fraction was then crushed in a jaw crusher to minus 3 mesh. (A number of metallic pieces were left at the end of crushing; these were at first thought to be native silver but they were later found to have a negligible silver content. Chemical tests showed that they actually consisted of aluminum, apparently derived from detonators used in blasting). The minus 3 mesh product was screened on 10 mesh and the oversize was combined with the original minus 3 plus 10 mesh fraction and crushed in stages to minus 10 mesh. The various minus 10 mesh fractions were combined, mixed thoroughly and riffled to give 500 and 1000 gram test and assay samples. One sample was dried and lost 2.6% moisture. It was then screened on a 100 mesh sieve and the oversize was ground in a mortar until only 3% remained. The whole of the plus 100 mesh fraction and part of the minus 100 mesh fraction were assayed for silver and gold; the minus 100 mesh fraction was also assayed for lead, antimony, copper, zinc, arsenic and sulphur. This procedure was followed in order to overcome sampling difficulties due to the possible presence of coarse silver in the ore.

ASSAY OF HEAD SAMPLE

| | | |
|----------|----|----------------------|
| Silver | Ag | 56.8 ounces per ton* |
| Gold | Au | 0.08 ounces per ton+ |
| Lead | Pb | 3.24 % |
| Antimony | Sb | 2.81 % |
| Copper | Cu | 0.06 % |
| Zinc | Zn | Trace |
| Arsenic | As | 9.40 % |
| Sulphur | S | 0.32 % |

* Plus 100 mesh fraction assayed 6.65 ounces of silver per ton and minus 100 mesh fraction assayed 58.35 ounces per ton.

+ Plus 100 mesh fraction assayed only a trace of gold and minus 100 mesh fraction assayed 0.08 ounces per ton.

Semi-quantitative spectrographic analysis (performed by Technical Service Laboratories, Toronto):

| | | | | | |
|----|-----------------|----|------------|--------------------------------|------------|
| Sb | MH | Ir | ND | Ta | ND |
| As | LM | Ia | ND | Te | X |
| Ba | FT | Pb | M | Tl | X |
| Be | ND | Li | ND | Th | ND |
| Bi | TL (.1%) | Mn | FT (.001%) | Sn | FT (.005%) |
| B | T | Hg | ND | Ti | TL (.1%) |
| Cd | ND | Mo | ND | W | ND |
| Ce | ND | Nd | ND | U | ND |
| Cs | X | Ni | ND | V | FT |
| Cr | FT (.001%) | Pd | ND | Y | ND |
| Co | ND | P | ND | Zn | ND |
| Cb | ND | Pt | ND | Zr | FT |
| Cu | T | Re | X | Al ₂ O ₃ | MH |
| Ga | FT | Rh | ND | CaO | T |
| Ge | ND | Rb | X | Fe | M |
| Au | FT (.1 oz./ton) | Ru | ND | MgO | L |
| Hf | X | Ag | T | SiO ₂ | H |
| In | ND | Sr | ND | Na ₂ O | T |
| | | | | K ₂ O | T |

Note: Figures are approximate

Code:-

| | | | |
|----|--------------------------|----|--|
| H | - High (10-100% approx.) | T | - Trace (.01-.1% approx.) |
| MH | - Medium High (5-50% ") | FT | - Faint trace (less than .01% approx.) |
| M | - Medium (1-10% ") | | |
| LM | - Low medium (.5-5% ") | PT | - Possible trace |
| L | - Low (.1-1% ") | ND | - Not detected |
| TL | - Trace low (.05-.5% ") | X | - Not looked for |

Note: With the exception of the above spectrographic analysis, all assays quoted in this report were performed by Coast Eldridge Engineers and Chemists Limited, Vancouver.

DETAILS OF TESTS

Because of the highly oxidized nature of the ore, it was considered that cyanidation would probably be the best method of recovering most of the silver and gold and this method was therefore investigated first. Additional recovery of silver, together with part of the lead and antimony, could probably be achieved by flotation or gravity concentration either before or after cyanidation and it is proposed to investigate these methods in the next stage of the investigation.

Test 1

A 1000 gram sample of minus 10 mesh ore was ground at 60% solids in a Denver laboratory ball mill for a period of 15 minutes with 1 gram of hydrated lime, equivalent to 1.5 pounds CaO per ton of ore. The pulp was washed into a pail and allowed to settle for 2 hours. A considerable amount of slime still remained in suspension at the end of the settling period. An attempt was made to filter off the excess water but filtration was very slow. A further 2 grams of calcium hydroxide were added; after stirring, the pulp was well flocculated. After settling for 5 minutes the slimes were filtered off and dried. They were then transferred to a Winchester bottle, together with the sands, and water was added to give a 2:1 water/solid ratio. 2 grams of calcium hydroxide, equivalent to 3 pounds CaO per ton of ore, together with 10 grams of sodium cyanide (20 pounds per ton of ore) were added and the bottle was agitated on rolls with free access of air for a period of 16 hours.

The pulp was then filtered and the cake was washed with water until a total of 3000 ml of filtrate and washes were obtained. The pulp was heavily flocculated and filtration was fairly fast.

The filter cake was dried and re-cyanided for a period of 64 hours, the same quantities of cyanide, lime and water being added as in the first cyanidation. Washing of the filter cake was continued until a total volume of 3500 ml was reached. After drying, a sample of the residue was taken for assay and 500 grams were cyanided for a period of 64 hours with 5 grams of sodium cyanide, 1 gram of calcium hydroxide and 1000 ml of water. The filter cake was washed until the total volume of filtrate and washes reached 2000 ml. The grinding water, the three cyanide filtrates and the residue from the second cyanidation were assayed for silver and gold; the cyanide and lime contents of the various filtrates were also determined.

Metallurgical results (overall):

| <u>Product</u> | <u>Amount</u> | <u>Gold</u> | <u>Silver</u> | <u>Distribution %</u> | |
|----------------|---------------|--------------|---------------|-----------------------|---------------|
| | | <u>Assay</u> | <u>Assay</u> | <u>Gold</u> | <u>Silver</u> |
| Grinding sol. | 6000 ml | Trace | Trace | - | - |
| 1 st. filtrate | 3000 ml | 0.66 mg/l | 413.40 mg/l | 88.0 | 64.3 |
| 2nd. filtrate | 3500 ml | 0.02 mg/l | 50.68 mg/l | 3.1 | 9.2 |
| 3rd. filtrate* | 3920 ml | 0.01 mg/l | 4.68 mg/l | 1.7 | 0.9 |
| 3rd. residue * | 976 gm | 0.005oz/ton | 14.75 oz/ton | 7.2 | 25.6 |
| Head (dry wt.) | 974 gm | 0.067oz/ton | 57.78 oz/ton | 100.0 | 100.0 |

*Calculated figures, based on re-cyanidation of whole of second cyanidation residue.

Recovery of gold and silver at various stages:

| <u>Stage</u> | <u>Cyanidation period (total)</u> | <u>Gold Recovery (total)</u> | | <u>Silver Recovery (total)</u> | |
|--------------|-----------------------------------|------------------------------|---------------|--------------------------------|---------------|
| | | <u>%</u> | <u>Oz/ton</u> | <u>%</u> | <u>Oz/ton</u> |
| 1 | 16 hours | 88.0 | 0.059 | 64.3 | 37.15 |
| 1+2 | 80 hours | 91.1 | 0.061 | 73.5 | 42.47 |
| 1+2+3 | 144 hours | 92.8 | 0.062 | 74.4 | 42.99 |

73.5% of the silver and 91.1% of the gold, equivalent to 42.47 and 0.061 ounces respectively per ton of ore, were recovered in a total cyanidation period of 80 hours. Increasing the total cyanidation period to 144 hours gave only slight increases in the silver and gold recoveries.

The Cyanide and lime consumptions were as follows:

| <u>Stage</u> | <u>NaCN consumed</u> | <u>CaO Consumed</u> |
|--------------|---------------------------|--|
| | <u>Lb. per ton of ore</u> | <u>Lb. per ton of ore</u> |
| 1 | 8.58 | 2.98 |
| 2 | 8.34 | 2.65 |
| 3 | 6.76 | 2.44 |
| Grinding | -- | 1.54 |
| Thickening | -- | 3.08 |
| Total stages | | |
| 1 and 2 | 16.92 | 10.25 (including grinding) (and thickening) |

The cyanide consumption was exceptionally high in each cyanidation stage; examination of the various solutions indicated that the heavy consumption was mainly due to the formation of ferrocyanide; the iron necessary for its formation may have been derived from readily-decomposed iron bearing minerals in the ore and/or abraded metal from the grinding mill and balls.

Analysis of the various solutions showed that all of the lime added before grinding and added as a flocculating agent had been used up; in addition, very little lime remained at the end of each cyanidation period. The heavy lime demand was probably due to the high proportion of clay present in the ore. Previous work has shown that the formation of ferrocyanides can be suppressed by increasing the lime content of the solution and this was therefore investigated in the next test.

TEST 2

This test was similar to the first test except that the quantity of hydrated lime added before grinding was increased to 3 grams, equivalent to 4.5 pounds CaO per ton of ore, no lime was required to flocculate the pulp after grinding and the lime added before each stage of cyanidation was increased to 5 grams of Ca(OH)₂, equivalent to 7.5 pounds of CaO per ton. Cyanidation was carried out in two stages, each of 40 hours.

Metallurgical results:

| <u>Product</u> | <u>Amount</u> | <u>Gold Assay</u> | <u>Silver Assay</u> | <u>Distribution %</u> | |
|----------------|---------------|-------------------|---------------------|-----------------------|---------------|
| | | | | <u>Gold</u> | <u>Silver</u> |
| 1st filtrate | 3000 ml | 0.68 mg/l | 451.68 mg/l | 84.8 | 67.7 |
| 2nd filtrate | 3500 ml | 0.006mg/l | 4.96 mg/l | 0.9 | 0.9 |
| 2nd residue | 1007 gm | 0.010oz/ton | 18.18 oz/ton | 14.3 | 31.4 |
| Head (calc.) | 1000 gm | 0.070oz/ton | 58.33 oz/ton | 100.0 | 100.0 |

Overall gold recovery - 85.7% or 0.060 ounces per ton
 Overall silver recovery - 68.6% or 40.01 ounces per ton.

Cyanide and lime consumptions:

| <u>Stage</u> | <u>NaCN consumed</u> | <u>CaO consumed</u> |
|--------------|----------------------|---------------------|
| | Lb. per ton of ore | Lb. per ton of ore |
| 1 | 10.22 | 7.34 |
| 2 | 4.76 | 6.66 |
| Grinding | -- | 4.50 |
| <u>Total</u> | <u>14.98</u> | <u>18.50</u> |

There was again a heavy consumption of cyanide in the first stage of cyanidation, in which almost all of the lime was used up. In the second stage, there was a fairly high concentration of lime in the solution at the end of the cyanidation period and the cyanide consumption was appreciably lower.

The silver recovery, 68.6%, was 4.9% lower than in the first test after 80 hours treatment; no definite explanation for this is known. The percentage gold recovery also appeared to be lower, but it should be noted that the very low assay of the residue creates an assaying problem. As soon as the investigation has advanced to a stage where more accurate figures are required, special measures will be taken in carrying out assays for gold in the residues.

TEST 3

This test was carried out at the same time as test 2 and was identical except that the grinding period was increased from 15 to 30 minutes. The filtration rates of the pulps were slightly lower than in test 2 but were still satisfactory.

Metallurgical results

| <u>Product</u> | <u>Amount</u> | <u>Gold</u> | <u>Silver</u> | <u>Distribution %</u> | |
|----------------|---------------|--------------|---------------|-----------------------|---------------|
| | | <u>Assay</u> | <u>Assay</u> | <u>Gold</u> | <u>Silver</u> |
| 1st filtrate | 3500 ml | 0.54 mg/l | 400.28 mg/l | 88.7 | 70.9 |
| 2nd filtrate | 3500 ml | 0.01 mg/l | 3.52 mg/l | 1.6 | 0.6 |
| 2nd residue | 1006 gm | 0.006oz/ton | 16.32 oz/ton | 9.7 | 28.5 |
| Head (dry wt.) | 1000 gm | 0.062oz/ton | 57.64 oz/ton | 100.0 | 100.0 |

Overall gold recovery - 90.3% or 0.056 ounces per ton
 Overall silver recovery - 71.5% or 41.21 ounces per ton

Cyanide and lime consumptions

| <u>Stage</u> | <u>NaCN consumed</u> <u>Lb. per ton of ore</u> | <u>CaO consumed</u> <u>Lb. per ton of ore</u> |
|--------------|---|--|
| 1 | 10.64 | 7.28 |
| 2 | 5.60 | 6.38 |
| Grinding | -- | 4.50 |
| Total | 16.24 | 18.16 |

As might be expected after finer grinding, the cyanide consumption was higher than in the previous test.

The silver recovery, 71.5% was 1.9% higher than in test 2 but 2.0% lower than was obtained after 80 hours' cyanidation in the first test; the gold recovery was substantially the same as in the first test.

TEST 4

In this test, the lime added before grinding was increased to 15.0 pounds of CaO equivalent per ton of ore (10 grams hydrated lime). After grinding for a period of 15 minutes as in tests 1 and 2, the pulp was divided into two equal parts and agitated in open bottles at 19% solids for 20 hours. The purpose of this was to oxidise any soluble iron present and thereby prevent it from consuming cyanide later. At the end of the aeration period, the pulp was filtered and the cake was washed until a total filtrate volume of 6000 ml was obtained. The cake was then repulped with water and cyanided under the same conditions as in tests 2 and 3, except that the second agitation period was reduced to 16 hours.

In order to determine the distribution of any silver remaining in the residue, a screen analysis was carried out as follows:

The washed filter cake from the second cyanidation was repulped with water to 40% solids, agitated on rolls for 10 minutes, filtered, washed, repulped to 40% solids and agitated for 16 hours with the addition of 10 grams of sodium silicate and 10 grams of

Test 4 Cyanidation results:

| <u>Product</u> | <u>Amount</u> | <u>Gold</u> | | <u>Silver</u> | |
|--|---------------|--------------|--------------|---------------|-----------------------|
| | | <u>Assay</u> | <u>Assay</u> | <u>Assay</u> | <u>Distribution.%</u> |
| | | | | <u>Gold</u> | <u>Silver</u> |
| 1st filtrate | 3500 ml | 0.58 mg/l | 370.28 mg/l | 77.4 | 66.0 |
| 2nd filtrate | 3500 ml | 0.04 mg/l | 2.02 mg/l | 5.3 | 0.4 |
| 1st repulp sol. | 2500 ml | 0.006mg/l | 0.14 mg/l | 0.8 | -- |
| 2nd repulp sol. | 15500 ml | 0.005mg/l | 0.58 mg/l | 3.1 | 0.5 |
| Residue | 1009 gm | 0.01 oz/ton | 18.84 oz/ton | 13.4 | 33.1 |
| Head | 1000 gm | 0.076oz/ton | 56.78 oz/ton | 100.0 | 100.0 |
| Overall gold recovery by cyanidation - 82.7% or 0.063 oz. per ton | | | | | |
| Overall silver recovery by cyanidation - 66.4% or 37.70 oz. per ton. | | | | | |

Cyanide and lime consumptions:

| <u>Stage</u> | <u>NaCN consumed</u> | | <u>CaO consumed</u> | |
|--------------|---------------------------|-------|---------------------------|--|
| | <u>Lb. per ton of ore</u> | | <u>Lb. per ton of ore</u> | |
| 1 | 5.12 | 6.44 | | |
| 2 | 0.50 | 5.04 | | |
| Grinding | -- | 14.82 | | |
| Total | 5.62 | 26.30 | | |

There was a considerable reduction in the quantity of cyanide consumed, 5.62 pounds per ton in the two stages of treatment, as compared with 14.98 pounds per ton in test 2. The cyanidation filtrates contained much less iron than in the earlier tests and it is probable that other cyanide soluble impurities such as copper accounted for most of the cyanide that was consumed.

The recovery of silver, 66.4% showed a further reduction and it appears that high additions of lime have a harmful effect on the solution of silver minerals. Only 0.4% of the silver was recovered in the second cyanidation period, showing that prolonged agitation periods would not appreciably increase the recovery. It is possible, however, that some method might be found whereby the cyanide consumption could be reduced without suppressing the solubility of silver.

The form in which the cyanide-insoluble silver is present in the ore is not definitely known, but the close relationship between the silver, lead and antimony assays of the various screen fractions obtained from the cyanidation residue, as shown in the following table, suggests that the insoluble silver may be present as a lead-antimony-silver mineral:

| <u>Fraction</u> | <u>Ratio of silver assay (oz/ton) to other assays (%)</u> | | | |
|-----------------|---|--------------|--------------|-------------|
| | <u>Ag/Pb</u> | <u>Ag/Sb</u> | <u>Ag/As</u> | <u>Ag/S</u> |
| Plus 200 mesh | 6.75 | 7.46 | 1.11 | 71.7 |
| - 200 + 325 | 6.63 | 7.26 | 1.27 | 91.5 |
| Minus 325 mesh | 5.79 | 6.98 | 2.07 | 118.6 |
| Total residue | 5.81 | 7.03 | 1.92 | 113.5 |

No similar relationship is apparent between the silver assays and the arsenic and sulphur assays. However, in view of the relatively large percentage of arsenic present, most of which probably occurs as the mineral scorodite (iron arsenate), some arsenic may be present with the insoluble silver. The ratio of lead to antimony, 1.2 to 1 (by weight) is similar to that of some samples of the mineral bindheimite (hydrous lead antimonate), which is an oxidation product of Jamesonite (lead antimony sulphide). If in fact the silver is present as bindheimite and the mineral could be concentrated to about 30% lead, it would have a theoretical silver assay of about 175 ounces per ton, with about 25% antimony. In practice, owing to the finely-divided nature of the residue, recovery of such a mineral would be difficult and a fairly high proportion would probably be lost.

CONCLUSIONS

The results of these preliminary tests indicate that, when treating ore of similar characteristics, it should be possible to recover at least 66% of the silver and 82% of the gold by cyanidation. The sodium cyanide and lime consumptions would be about 6 and 26 pounds per ton respectively. These results could probably be improved by a more thorough investigation of the problem. In addition, it may be possible to recover some of the remaining silver, together with lead and antimony, in the form of a concentrate which could be shipped to a smelter.

