CONTACT INFORMATION
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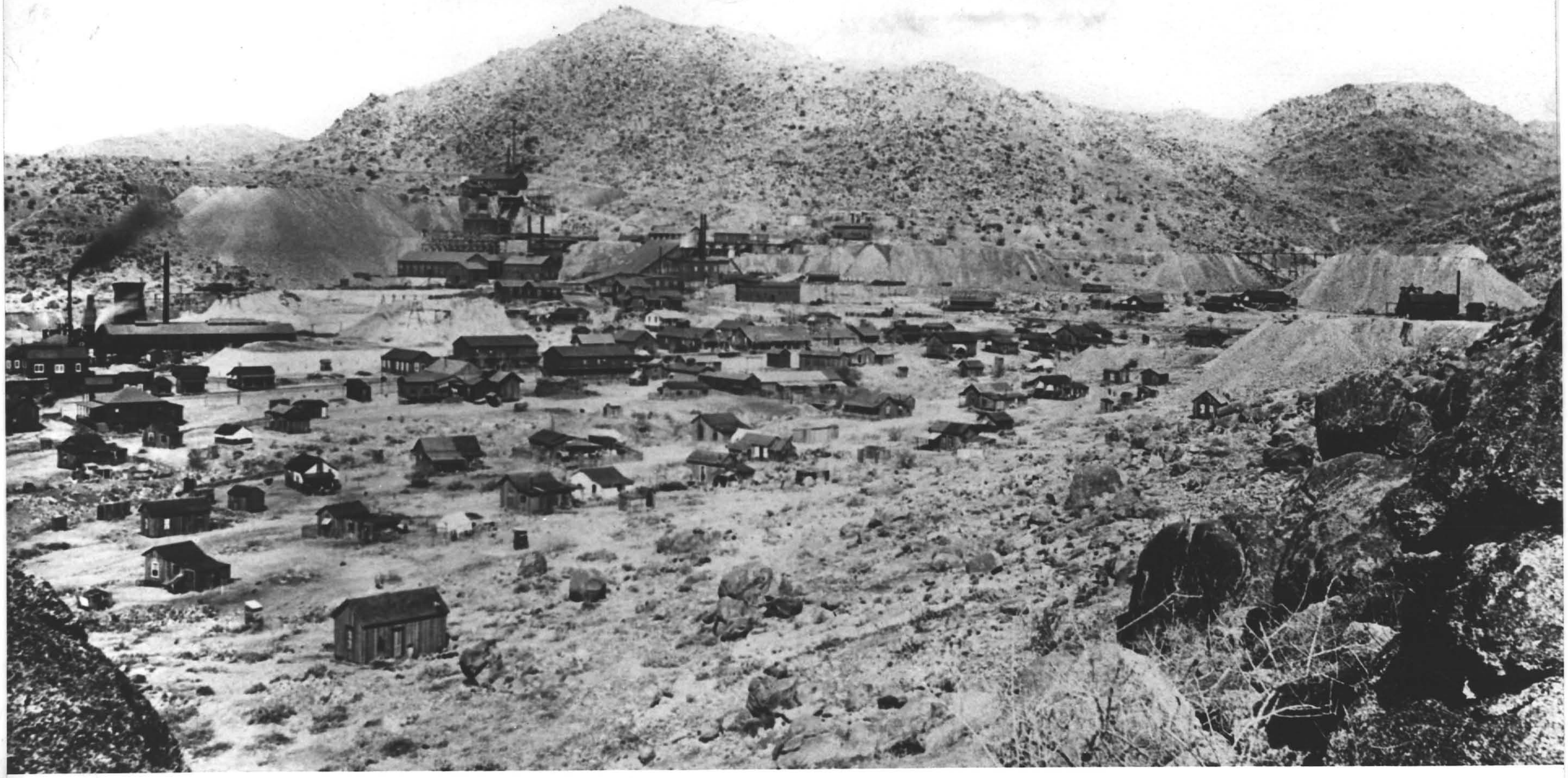

Congress Mine
1910

D.W. JAQUAYS
MINING ENGINEER
132 WEST GRANADA RD.
PHOENIX, ARIZONA

# ORE POSSIBILITIES AT CONGRESS MINE

With Gold @ $150.00/ oz and Silver @ $4.00/ oz

| | | | | | |
|---|---|---|---|---|---|
| No. 1 Dumps | 55,000 | tons | @ 0.10 oz X $150.00 | = | $825,000.00 |
| No. 2 " | 90,000 | " | @ 0.10 " " " | = | 1,350,000.00 |
| No. 3 " | 80,000 | " | @ 0.07 " " " | | 840,000.00 |
| No. 4 " | 4,000 | " | @ 0.10 " " " | | 60,000.00 |
| No. 5 " | 50,000 | " | @ 0.10 " " " | | 750,000.00 |
| No. 6 " | 8,000 | " | @ 0.12 " " " | | 144,000.00 |
| Queen of Hills | 15,000 | " | @ 0.10 " " " | | 225,000.00 |
| Total Rock Dumps | 302,000 | " | | | $4,194,000.00 |
| Old Tailings | 150,000 | " | @ 0.07 " " " | | $1,575,000.00 |
| Sub-Soil | 20,000 | | @ 0.10 " " " | | 300,000.00 |
| Total Tailings | 170,000 | | | | $1,875,000.00 |
| Total all dumps | 472,000 | " | Gross value | | $6,069,000.00 |
| | | | Less 20% tailing loss | | 1,213,800.00 |
| From all dumps --- | | | Possible net recovery | | $4,855,200.00 |

Ore possiblities
Congress Vein ore extensions and low grade
Gob and dike material below 650 level

| | | | | |
|---|---|---|---|---|
| Congress Vein | 600,000 tons | @ 0.25 oz X $150.00 | = | $22,500,000.00 |
| Queen of the Hills | 200,000 tons | @ 0.25 oz X $150.00 | = | 7,500,000.00 |
| Congress extension Sullivan & Suprise Veins | 500,000 tons | @ 0.30 oz X $150.00 | = | $22,500,000.00 |
| Totalpossible unbroken ore not considering vein **extensions** to depth | | | | $52,500,000.00 |
| | Deduct 10% for tailing loss | | | 5,250,000.00 |
| | Total possible recovery from ore | | | $47,250,000.00 |
| | Total possible recovery from dumps | | | 4,855,200.00 |
| Total possible recovery from 1,772,000 tons ore and surface dumps | -------------------- | | | $52,105,200.00 |

Operating at 500 tpd or 180,000 tons year there would be indicated 10 years ore not considering possible additional ore indicated at depth, vein extensions, and undeveloped parallel veins.

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# CONGRESS MINE

## Remilling Data 1938 thru 1942

| | Tons Milled | Ounces Au Recovered | Average Recovered | Remarks |
|---|---|---|---|---|
| 1938 7 Mos. | 49,727 | 3110.76 | 0.0624 | All tailings |
| 1939 12 " | 102,808 | 4,932.05 | 0.0487 | Started milling Dumps Nov. |
| 1940 12 " | 93,990 | 5,380.87 | 0.0572 | 60% Tailing 40% Dumps |
| 1941 12 " | 97,778 | 7,383.57 | 0.0744 | 50% " 50% " |
| 1942 5 " | 41,587 | 3,060.70 | 0.0736 | 50% " 50% " |
| Total 48 Mos. | 387,890 | 23,867.95 | | |

Estimated Silver Recovery 0.10 ounce per ton or 38,789 ounces

Average Gold recovery total tonnage 0.0615 ounces Au. per ton

| | |
|---|---|
| Value gold @ $150.00/ oze | $9.23 |
| " Silver @ $4.50/ oze | 0.45 |
| | $9.68 / ton |

| | |
|---|---|
| Value Gold @ $200.00/ oze | $12.30 |
| " Silver @ $4.50/ oze | 0.45 |
| | $12.75/ ton |

The above data from actual remilling of the tailinga and rock dumps indicates what may be expected in recovery from the estimated 255,000 tons of rock dumps and 150,000 tons of tailings left by previous operators when shut down by W.W. War 2 in 1942.

Historical Data

| | | | | |
|---|---|---|---|---|
| Total gold recovered 1880 to 1911 | | 388,477.00 ozes | Silver | 345,598.0 oze |
| " " " 1938 to 1942 | | 23,867.95 " | " | 38,787.0 |
| | Gold | 412,344.95 " | Silver | 384,385.0 |

| | |
|---|---|
| Total tons mined and milled 1880 to 1911 | 692,332 tons |
| Total tons low grade put on dumps | 412,000 " |
| Total tonnage removed from mine | 1,104,332 tons |
| L ess tonnage remilled 1938 to 1942 | 387,890 " |
| Possible tonnage left in dumps instead of estimated 405,000 tons listed above. | 716,442 " |
| | 405,000 |
| Difference which may be due to underestimatingthe material contained the dumps and erosion of the tailings pile. Does not sound logical that this tonnage could be lost. | 311,442 Tons |

Congress Mine
Cost Analysis

Milling Dumps Only

| | | | |
|---|---|---|---|
| Estimated Recovery | Au .07 oz. X $175.00/ oz= | $12.25 | |
| | Ag..20 " X $4.00/ oz | 0.80 | |
| | | $13.05 | |
| Estimated daily recovery $13.05 X 500 TPD = | | | $6,500.00 |
| Cost per day | | | |
| Labor including Ins. O.A.B. Etc | $1,400.00 | | |
| Supplies | 500.00 | | |
| Power | 500.00 | | |
| Extra for support labor Office Etc | 300.00 | | |
| | | $2,700.00 | |
| Frieght and Smelter Charges on Conc | 500.00 | | |
| Property Taxes | 100.00 | 600.00 | |
| Amortization of $1,500,000.00 plus 10% Int. over 3 years or approximately $2,000,000.00 on 500,000 tons @ $4.00/ ton | | $2,000.00 | |
| Total cost per day milling 500 TPD or $10.60 per ton costs | | | $5,300.00 |
| Estimated profit per day from dumps @ $2.40/ ton | | | $1,200.00 |
| Estimated profit on 500,000 tons dump material milled over three years | | | $1,200,000.00 |

Suggest that after say six months of operation on the dumps and profit trend is established that a large portion of the profits be used to finance the rehabilitation of the underground ratherthan pay a heavy profits tax. In other words let the IRS pay for some of the development, they will be paid enough taxes later on without risking anything.

Add approximately $3.75 per ton for 90% recovery above figures are computed at 70% recovery.

Congress Mine

Program for going underground as quickly as possible and feeding mill with part newly mined ore and dump material.

Example:

| | | |
|---|---|---|
| 200 tpd from mine Estimated Value 0.4oz. or | $70.00/T | $14,000.00 |
| 300 " " Dumps | @ $13.00/t | 3,900.00 |
| Estimated daily recovery | | $17,900.00 |

| | | |
|---|---|---|
| Mining costs for 200 TPD @ 25.00/ ton | $5,000.00 | |
| Milling cost and recovery of 300 tons per day of dump material | $5,300.00 | |
| Amortization of Underground Dev. charges a $2.00/ ton X 200 TPd | 400.00 | |
| Total costs | $10,700.00 | $10,700.00 |
| Estimated profit with 200 TPD new ore and 300 tpd dump material | | $6,200.00 |
| Average profit $12.50/ ton X 180,000/ Year = | | $2,250,000.00 |

After three years most of the dumps would be milled and the mill amortized but would suggest leaving this charge in the costs for later underground costs to amortize the rehabilitation costs and later a cushion against inflation and increased costs.

Example:

| | | |
|---|---|---|
| Mining and Milling 500 TPD New Ore 0.3oz/ ton $53.00= | | $26.500.00 |
| Milling costs including original Amortization charges of $4.00/ ton @ $10,60 | =$5,300.00 | |
| Mining Charges @ $28.00/ ton | $14,000.00 | |
| Amortization of UG charges $2.00/ t | 1.000.00 | |
| Total costs 500 tpd from underground | $20,300.00 | $20,300.00 |
| Estimated daily profit | | $ 6,200.00 |

| Distribution of Costs/ day | | | Per Year Mining and Milling 500 TPD. |
|---|---|---|---|
| Milling | $2,700.00 | $5.40 | $972,000.00 |
| Frt. & Smelter | 500.00 | 1.00 | $180,000.00 |
| Mining @ $28.00/t | $14,000.00 | 28.00 | $5,040,000.00 |
| Taxes | 100.00 | .20 | 36,000.00 |
| Amortization of start up charges | | | |
| Mill $4.00/ tonX 500 | 2,000.00 | 4.00 | 720,000.00 |
| Mine $2.00/ tonX 500 | 1,000.00 | 2.00 | 360,000.00 |
| | $20,300.00 | $40.60 | $7,308,000.00 |

| | | |
|---|---|---|
| Estimated gross Recovery | 180,000/ @ $53,00/ t | $9,540,000.00 |
| Estimated costs | 180,000/ @ $40.60/ t | $7,308,000.00 |
| Estimated profit before taxes | | $2,232,000.00 |

Congress Mine

New Ore Operation with Possible Recovery of $U_3O_8$

| | |
|---|---|
| Gross sales from 500 Tpd Au and Ag. | $9,540,000.00 |
| Possible recovery of 1 lb./ ton | |
| $U_3O_8$ @ $8.00/ Lb X 180,000Tons per year= | $1,440,000.00 |
| Possible gross sales with $U_3O_8$ ------ | $10,980,000.00 |
| | |
| Estimated costs | 7,308,000.00 |
| Possible profits with Au, Ag. & $U_3O_8$ ---- | $3,672,000.00 |

# PROPERTY LOCATION AND GENERAL CONDITIONS

The holdings of the Congress Mining Corporation in Yavapai County, Arizona, consists of the patented and unpatented lobe mining claims listed below with all dumps, tailings and improvements thereon.

| Name | Area In Acres | Recorded In Book of Deeds | Page |
|---|---|---|---|
| Congress | 20.02 | 30 | 476 |
| Queen of the Hills | 17.47 | 30 | 480 |
| Niagara | 20.66 | 30 | 484 |
| Mosouri | 5.36 | 30 | 488 |
| Why Not | 20.66 | 30 | 493 |
| Fraction | 7.40 | 33 | 497 |
| Niagara Mill Site | 4.95 | 33 | 617 |
| Excelsior | 20.66 | 33 | 620 |
| Incline | 20.64 | 41 | 94 |
| Rich Quartz | 20.65 | 41 | 97 |
| Golden Eagle | 18.93 | 41 | 100 |
| Snowstorm | 20.66 | 41 | 104 |
| Ohio | 20.66 | 41 | 107 |
| Old State | 20.24 | 41 | 110 |
| Golden Thread | 20.66 | 54 | 104 |

The surface rights and the dumps thereon to a depth of 40 Feet on the above claims owned by D. W. Jaquays and leased to the Congress Cons. Gold Mng. Co.

## UNPATENTED MINING CLAIMS

All of these claims relocated by D. W. Jaquays and transfered to the Congress Cons. Gold Mng. Co.

| Name | Recorded in Book of Mines | Page |
|---|---|---|
| Bellick | 24 | 291 |
| Remnant | 25 | 314 |
| Boundary | 35 | 161 |
| Sunnyside | 45 | 499 |
| Highland | 45 | 496 |
| Keystone | 50 | 364 |
| East Extension of Golden Thread | 51 | 156 |
| Martinez | 66 | 591 |
| Cphir | 86 | 341 |

## Patented Mining Claims

| Name | Area in Acres | Recorded in Book of Deeds | Page |
|---|---|---|---|
| Jaquays Nos. 5 & 6 | 36.199 | 323 | 112 |
| Jaquays " 7 & 14 | 37.136 | 323 | 100 |

The above Jaquays Claims have been transfered to the Congress Cons. Gold Mining Co.

Congress Claims held by Location Jan. 1, 1975

| NAME | BOOK | PAGE |
|---|---|---|
| Jaquays No. 1 Amended | 848 | 855 |
| Jaquays No. 2 " | 848 | 856 |
| Jaquays No. 3 " | 848 | 857 |
| Jaquays No. 4 | 112 | 316 |
| Jaquays No. 9 Amended | 848 | 850 |
| Jaquays No. 10 | 112 | 321 |
| Jaquays No. 11 | 112 | 322 |
| Jaquays No. 12 Amended | 212 | 213 |
| Jaquays No. 13 | 112 | 325 |
| Jaquays No. 16 | 848 | 859 |
| Jaquays No. 17 | 112 | 329 |
| Jaquays No. 20 | 848 | 851 |
| Jaquays No. 21 | 848 | 852 |
| Jaquays No. 22 | 848 | 853 |
| Jaquays No. 23 | 848 | 854 |
| * B & M | 327 | 5 |
| Congress Extension No. 1 | 657 | 875 |
| " " No. 2 | 657 | 876 |
| " " No. 3 | 657 | 877 |
| " " No. 4 | 657 | 878 |
| " " No. 5 | 657 | 879 |
| " " No. 6 | 657 | 880 |
| " " No. 7 | 657 | 881 |
| " " No. 8 | 845 | 332 |
| " " No. 9 | 845 | 333 |
| * Jaquays No. 35 | 904 | 217 |
| * Jaquays No. 28 | 904 | 218 |
| * Congress Extension No. 13-A | 904 | 214 |
| * " " No. 14-A | 904 | 215 |
| * " " No. 15-A | 904 | 216 |
| * " " No. 20 | 904 | 220 |
| * " " No. 20 Amended | 905 | 215 |
| * " " No. 21 | 904 | 221 |
| * " " No. 22 | 904 | 222 |
| * Jaquays No. 5-A Fraction | 904 | 219 |
| * Congress Extension No. 21 Amended | 944 | 826 |
| * Congress Extension No. 22 " | 944 | 827 |
| * Congress Extension No. 14-A " | 944 | 826 |
| * Congress Extension No. 15-A " | 944 | 825 |
| * Congress Extension No. 16 | | |

The above listed claims have been transfered by D. W. Jaquays to the Congress Cons. Gold Mining Co.

* Indicates claims on which patent has been applied for.

D.W. JAQUAYS
MINING ENGINEER
132 WEST GRANADA RD.
PHOENIX, ARIZONA

CONGRESS CONSOLIDATED GOLD MINING CO.

| | | |
|---|---|---|
| Capitalized at | | 3,000,000 shares |
| Shares issued for property | 800,000 | |
| " sold for cash | 25,500 | |
| Total stock issued 1-175 | | 825,500 |
| Unissued stock in treasury | | 2,174,500 shares |

Working and development capital to be advanced by purchase of treasury stock at $1.00 per share for first 200,000 shares to finance stage one. Stages two and three to be financed by the purchase of additional 500,000 shares of stock at $1.25 per share, the last 500,000 shares at $1.50 or enough to provide funds for the construction of at least a 500 TPD flotation- Cyanide Plant for the retreatment of the mine dumps. The cyanide section could be added later after higher grade mine ore is to be treated.

Suggest as stage one the using of $200,000.00 of the intial investment for a diamond drilling program, the development of water, and metallurgical tests on the rock dumps. Also #3 shaft should be reopened in order further sample the gobs from the 650 level to the 1000 foot level.

It may be feasible after phase one is well along to begin construction of a 500 TPD Flotation plant for the treatment of the rock dumps which are estimated to contain 300,000 tons of low grade material. The cyanide section could be added later when ore begins coming from the mine.

CONGRESS MINE REPORT

BY

G. M. COLVOCORESSES

August 1943

Record from Mining and Metallurgical Society of America, 1937.

Brought up to date, 1942.

COLVOCORESSES, G. M. 1102 Luhrs Tower, Phoenix, Arizona

Consulting, Mining and Metallurgical Engineer

1900, Graduate, Yale University; 1900-01, Day laborer in smelting works and Asst. Chemist and Assayer. Assisted on mine examination, mine sampling and prospecting and exploration trips in Canada and U. S., for Orford Copper Co. and Ontario Smelting Works; 1901, Sent to New Caledonia as Ass't. Supt. of Mines for Nickel Corp., Ltd.; 1902-05, Ass't. Supt. and later Supt. of Mines for Nickel Corp., Ltd. and Societe Miniere Caledonienne; 1905-06, Office work in Paris and London prior to return to U. S. Worked at smelter of Canadian Copper Co. Examination work in Cobalt district and other parts of Ontario; 1906-09, Engineer on staff of International Nickel Co. Office work in New York and examination work in various parts of the U. S., Canada, Australia and New Caledonia, also Cuba and Porto Rico. Consulting Engineer for Massey Copper Mine Co., Ontario. In charge of exploration and development work for Anglo American Iron Co.,; 1908-12. Superintendent of Millerett Silver Mine, Ontario. In charge of exploration and examination work in that district, 1912-1914. Consulting Engineer, New York; 1914-21, General Manager Consolidated Arizona Smelting Co., Humboldt, Arizona, 1921-22, Federal Court Receiver for Cons. Arizona Smelting Co.; 1922-30, General Manager Southwest Metals Co. Humboldt, Arizona; 1916-20, Cons. Engineer Ohio Copper Co. of Utah; 1920-25, General Mgr. Swansea Lease, Inc.; 1919-32, President and Manager Western Metallurgical Co. of Los Angeles, Developing and operating a metallurgical process; 1926-30 Chief Metallurgical Engineer for Carson Investment Co. of San Francisco; 1917-30 Cons. Engineer Nicu Steel Corporation of Toronto, 1928-36, General Manager Meteor Crater Exploration and Mining Co.; 1923-1925, Governor of the Arizona Chapter of the American Mining Congress; 1930-42 Cons. Min. and Metallurgical Engineer, Phoenix, Arizona with employment of various clients and supervision of operations of various mines and plants producing mostly gold, silver and copper.

Similar information may be obtained from "Who's Who in Engineering" for 1931.

NOTE: Mr. Jaquays, I am attaching hereto this page showing some of Mr. Colvocoresses' tremendous background.

Stella Freasier
3/31/60

## HISTORY

According to W. F. Staunton the original Congress locations were made by Dennis May who sold the claims in 1887 to "Diamond Joe" Reynolds and Frank Murphy. The new owners operated the property with a 20 stamp mill and Frue Vanners for concentration until 1891 up to which date they had received a net return of about $592,000.00 from shipments of ore and concentrates. They always made a poor recovery of values since the oxidized ores found near the surface would not amalgamate and the gold in the sulphides was principally associated with marcasite which slimed easily so that tailing losses were high.

After an almost complete shut-down of some three years, work was resumed in 1894 by the Congress Gold Company. Prior to that date a standard gauge railroad (now a branch line of the Santa Fe) had been built to connect Congress Junction with Prescott and Phoenix and this was connected with the mine by a spur 3 miles in length, which has now been removed. The mill had been equipped with 40 stamps and additional vanners. At the mine the #2 shaft then had a depth of 1000' but no stoping had been done below the 650' level. Subsequently the cyanide process was introduced to greatly improve the milling practice. In 1901 another 40 stamps were added and during the next ten years a large part of the original mill tailings were retreated along with newly mined ore. The net returns from the production from 1894 to 1910 was $7,057,422.75.

The total tonnage of ore shipped or milled from March 1899 to the end of 1911 is recorded as 692,332 tons of which 370,022 tons was mined from the Congress vein with an average recovery of about 0.70 oz. of gold per ton. The Niagara vein supplied 293,215 tons with an average recovery of about 0.415 oz. gold per ton. 20,125 tons was mined from the Queen of the Hills vein, the average recovery is not stated but apparently it was a little less than 0.4 oz. A total of 388,477 oz. of gold and 345,598 oz. of silver were recovered and sold.

It would appear that the total mine production up to the end of 1911 was 692,332 tons of ore including all material shipped or treated in the mill, from which over $7,650,000 was realized in net payments for the ore, concentrates, and bullion making the average recovered value $11.81 per ton with gold at $20.67 per ounce and silver at 60 cents. The over all average assay of the ore may conservatively be estimated to have been 0.64 ounce gold and 1.00 ounce of silver which at the present prices would have had a value of over $23.00 per ton. This last figure includes the values of gold and silver left in over 600,000 tons tailings from milling operations. In addition to the above totals substantial values were left in the mine fills and ore dumps which will be described in another part of this report.

Between 1910 and 1937 the operations at Congress were principally confined to the retreatment of small portions of the mill tailings and ore dumps and no attempt was made to reopen the mine except by various lessees who mined some of the small pillars left in the upper workings. During these 27 years it is probable that upwards of 50,000 tons of dump rock and mill tailings had been treated by various parties but no type of operations at the Congress Mine appeared to hold a promise of yielding a profit until the price of gold was advanced from $20.00 to $35.00 per ounce.

In 1935 the property with the then existing improvements, valued at about $5,000 was sold for $26,000 by the Congress Trust (which succeeded the Congress Gold Mining Co.) to Gerald Sherman and Associates who organized the Congress Mining Corporation.

The mine dumps were considered of doubtful value until they were subsequently measured and fairly well sampled on two occasions. Once by the management and once under the direction of Henry G. Carlisle of San Francisco, both samplings indicated 400,000 tons (after allowance for sorting some waste) with an average value of $3.00 per ton in gold and silver. The condition of the mine, including the stope fills was practically the same as at present, except that many of the workings which were then open for inspection are now caved.

The Congress Mining Corporation in 1937 proceeded to erect a 300 ton-a-day counter-current cyanide mill along with a power plant and accessory equipment. This company operated its mill from June 1st, 1938 to June 14, 1942, during which time it treated a total of 385,505 tons of material of which 276,372 tons came from the tailings pile, 106,629 tons from the dumps, and 2,402 tons represented shipments of ore by leasors working in the Congress Mine or from custom shippers.

The recovered value of 51,576 tons of tailings and 37,915 tons of dump rock treated in 1940 averaged $2.05 in gold and silver. In 1941, 97,927 tons of tailings and dumps were treated (segregation not given) with a recovered value of $2.50 per ton. The returns indicate the mill heads averaged respectively $3.00 and $3.60 for an average of better than $3.30 a ton for these two years. These figures check with the monthly mill records for the above periods and it is indicated that the dumps ran higher than the old tailings.

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| Why Not | 20.66 | 30 | 493 |
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| Niagara Mill Site | 4.95 | 33 | 617 |
| Excelsior | 20.66 | 33 | 620 |
| Incline | 20.64 | 41 | 94 |
| Rich Quartz | 20.65 | 41 | 97 |
| Golden Eagle | 19.93 | 41 | 100 |
| Snowstorm | 20.66 | 41 | 104 |
| Ohio | 20.66 | 41 | 107 |
| Old State | 20.24 | 41 | 110 |
| Golden Thread | 20.66 | 54 | 104 |

The surface rights and the dumps thereon to a depth of 40 Feet on the above claims owned by D. W. Jaquays and leased to the Congress Cons. Gold Mng. Co.

## UNPATENTED MINING CLAIMS

| Name | | Recorded in Book of Mines | Page |
|---|---|---|---|
| Bellick | All of these claims | 24 | 291 |
| Remnant | relocated by D. W. Jaquays | 25 | 314 |
| Boundary | and transfered to the | 35 | 161 |
| Sunnyside | Congress Cons. Gold Mng. Co. | 45 | 499 |
| Highland | | 45 | 496 |
| Keystone | | 50 | 364 |
| East Extension of Golden Thread | | 51 | 156 |
| Martinez | | 66 | 591 |
| Ophir | | 86 | 341 |

All claims are located in the Martinez Mining District, Yavapai County, Arizona.

The main workings of the mine are located in a low range of hills known as the Date Creek Mountains, three miles from the Santa Fe Railway station at Congress Junction, Yavapai County, Arizona. The elevation at the tailings mill is 3560' above sea level and the collars of the various shafts are less than 100' higher.

The surface of the claims is rocky and rugged with ridges rising to heights of some 400' above the level of Martinez Creek which drains this area. There is no timber and only scant semi-desert vegetation. The climate, while hot in the summer, is well suited to operations throughout the year with frequent frost and only light snows during the winter months.

The local water supply is deficient for any large scale operations and will be discussed separately in this report.

In normal times ample common labor can be secured from Phoenix (73 miles distant from the mine) and from other cities in the Salt River Valley. Miners, millmen and other classes of skilled workmen can be recruited from various copper camps of the State. The existing living accomodations in and about Congress Junction would serve to house a crew of 50 to 60 men.

Supplies for mining and milling can be delivered to the mine by either railroad or over paved highways by motor truck. Phoenix located 73 miles over paved highway to the south is the mining and industrial supply center of Arizona.

## GEOLOGY AND ORE OCCURRENCE

The country rock forming the Date Creek Range, which lies to the west of the Weaver and Bradshaw Mountains, is mainly granite. Probably of pre-Cambrian age and in some areas with pegmatitic structure. Through this formation in the vicinity of the Congress Mine occur a number of greenstone (diorite) dikes which generally strike in the east-westerly direction and dip from 20-30 degrees to the north. Another series of more recent quartz-porphyry dikes strike northeasterly with nearly vertical dips; these last are believed to be post mineral. The diorite dikes generally carry some iron sulphides with low gold values.

The Congress vein lies along a contact between the granite and one of these dikes which has a width of from five to fifteen feet. The Niagara vein and other smaller veins are formed in fissures in the granite some distance away from the dike contacts. Most of the fissure veins strike in an east-westerly direction and usually dip 20 to 40 degrees to the north.

There is much evidence of minor faulting and one major fault cuts off both the Congress and Niagara veins at their east end and beyond this neither one of them has been positively located.

The pay ore in the Congress vein is associated with quartz, iron sulphide and arsenical iron sulphide, also small amounts of copper and zinc sulphides. In the Niagara vein and smaller veins there is some galena and a higher silver content.

Even though most of the Congress system of veins occur in the granite, there is good reason to believe that all of the ore deposition was due to a deep seated intrusive magma from which ascending solutions worked their way upward through fissures which are remarkably persistent and can be traced for long distances. The gold values are not entirely confined to the main veins but impregnate the wall rock, particularly in the case of the Niagara vein. Values also follow tiny stringers of quartz with disseminations of iron sulphides so that much low grade ore had been left in place in the vicinity of the old workings and a large tonnage of such material was used for backfilling in the stopes or hoisted to the surface dumps along with the waste.

Mr. Staunton, who was manager of the Congress Mine during its greates production period, has made the following comments from which I quote.

> "Some have considered that the dike was in reality the Congress vein since the ore occurred in all possible relations to the dike between the foot and hanging walls of granite, but usually the ore was found near the footwall and accompanied by a clay selvage."

An analysis of the greenstone dike which is usually termed diorite, gave the principal constituents as follows:

| | | |
|---|---|---|
| $SiO_2$ | = | .52.20% |
| $Al_2O_3$ | = | 13.40 |
| Fe O | = | 9.75 |
| Mn O | = | 1.90 |
| Ca O | = | 9.60 |
| Mg O | = | 1.16 |
| Total | | 88.01 |

Minor faulting is in evidence throughout the mine workings and there has been considerable relative movement of the walls of the Congress vein, resulting in local crumbling of the greenstone. The mine workings terminate to the east against a heavy fault, beyond which the vein has not been definitely located. This fault cuts off both the Congress and Niagara veins.

Although the Congress vein is continuous and well defined for a mile or more to the west of the mine workings and shows both the characteristic quartz and sulphides, the pay ore was practically confined to a shoot in the vein pitching to the northwest and coinciding closely with the intersection of one of the fissure veins in the granite. The granite vein is faulted by the Congress vein so that the intersection is obscure in the mine workings. The portion of the granite vein in the hanging wall of the Congress carried bodies of pay ore.

The Congress pay shoot varied greatly in length on different levels, being longest on the 650' level. Several pinches were met in following the vein down. The most serious being at the 1,700' level, where there was no stoping ground. On the theory that if pay ore existed below that point it would probably be found on the general line of trend of the ore shoot above, a deep prospecting winze was sunk from the 1,700' level - in the vein but with a northwesterly pitch corresponding to the established trend of the pay ore in the upper workings. This winze was sunk 1,000' and bore out fully the theory upon which it was projected. The pay ore coming in again as good as ever after a few hundred feet of lean ground.

The 3,900' level was the deepest point at which any considerable amount of development was done. For several levels above this there had been a gradual pinching of the pay shoot, which became small and irregular, although retaining its mineralogical characteristics and the small amount of sulphides which remained still showing the characteristically high gold content above 7.0 oz. per ton. The conditions were similar to those existing at other horizons in the mine where persistent deeper work had been rewarded by expansion of the ore shoot to normal size.

The history of the Congress Mine, its remarkable persistence probably due to its association with an intrusive dike of profoundly deep origin and the existence of similar parallel veins in both hanging and footwall over a wide belt, suggested a careful study of the whole situation to determine the feasibility of a broadly planned scheme of exploration by means of a vertical shaft so arranged as to cut the Congress vein at greater depth than has been attained. To cut and explore the other similar veins, many of which if not cut by the shaft could be reached by crosscuts.

## VEIN SYSTEM

The footwall vein in the Niagara which for some distance strikes nearly east and west and then going west turns to about north 25 degrees west. The dip of the Niagara vein is 40 degrees to the north.

The outcrop of the Congress vein is in the hanging wall about 400' to the north of the Niagara vein. The Congress vein does not turn northward as soon and its western section is only 250' to the north-east of the Niagara vein.

The so called dike vein underlies the Niagara but its outcrop is not shown on the map unless it is what is known as the Risto vein which outcrops on the Golden Thread and Blackhawk Claims. The Dike vein was cut by the 1975' level of the #5 shaft and according to Mr. Staunton it had a good width and average value of $25.00 per ton. It is probable that the ore from this shoot was mined out and I can find no further mention of this vein except that it was apparently developed from the surface by the Katherine shaft.

In the hanging wall to the north of the Congress vein there are outcrops including the Surprise and the Incline vein. It is probable that the former was termed the Spur vein by Brooks who claims that this vein was cut by #3 shaft at a depth of 2700' where it came into the shaft from the hanging wall.

The cross ledge which branches off to the northeast from the Congress vein near #1 shaft apparently runs through the Queens of the Hills and Bellick claims.

## EXTENT AND CHARACTER OF MINING OPERATIONS

Pay ore was mined from the Congress vein to a depth of 4,000' on an incline of 25 degrees, and from the Niagara vein to a 40 degree incline, depth of 2,000'. The maximum length of the ore shoot in the Congress vein was 1,800' on the 650' level but here the width of the pay ore did not exceed three feet, while in other portions of the mine the width of the ore was sometimes greater with some stopes having widths from five to fifteen feet.

On the 1700' level on the Congress vein the ore pinched out but came in again at a greater depth.

The lowest levels of the mine from 2500' to 4000'had shown a gradual progressive pinching or contraction of the ore shoots in the Congress vein. However, the situation was different in the Niagara vein where it is indicated that more ore should be found if further exploration is carried to a greater depth than the 2000' level which represented the greatest depth to which the Niagara vein was developed.

The shafts are as follows:

| | |
|---|---|
| On the Congress vein: | #1 Shaft------------1100 feet deep |
| | #2 Shaft------------1700 feet deep |
| | #3 Shaft------------4000 feet deep |
| On the Niagara vein: | #4 Shaft------------1000 feet deep |
| | #5 Shaft------------2050 feet deep |
| | #6 Shaft------------1800 feet deep |

On the Queen of the Hills vein a shaft was sunk 200 feet below the tunnel level.

The production of ore hoisted from the above shafts is recorded as follows:

| Congress vein | Tons | |
|---|---|---|
| Shaft #1 | 117,899 | |
| Shaft #2 | 122,779 | |
| Shaft #3 | 63,524 | |
| Total | 304,202 | Tons with an average recovered value of 0.7 oz. gold |

| Niagara vein | Tons | |
|---|---|---|
| Shaft #4 | 20,470 | |
| Shaft #5 | 191,734 | |
| Shaft #6 | 81,016 | |
| Total | 293,220 | Tons with an average recovered value of 0.415 oz. gold |
| Queen of the Hills vein | Tons | |
| | 20,125 | Tons value not stated but apparently slightly less than 0.4 oz. gold. |
| Combined Total | 617,547 | TONS |

## MINING PRACTICE

For information in this regard I am principally indebted to Mr. Staunton who wrote as follows in 1932:

> "The method of operation was like this; starting, say at the 1000' level, in the #2 shaft, the ore was stoped out on both sides of the shaft clear to the shaft, leaving no pillars as they were found to give trouble from uneven subsidence of the hanging wall which unavoidably took place when such large areas were taken out. When this stope, on a sloping line, away from the shaft reached 75' upon the vein - another level was started (the 925 in this case). The stope making the level except for a little cutting into the hanging wall. This new level would advance only as fast as the stope from below made it. When the stope above the new 925 level reached the 850 point, another level was started there - and so on. Each stope practically making the next level above it so that ultimately each level would be, say, 75 feet shorter than the one below it. As the vein varied greatly in thickness it was usually necessary to shoot some of the hanging wall and this constituted most of the stope filling together with hanging wall rock broken in the stope themselves. The high grade ore was usually next to the footwall but nearly always there were high grade stringers in all the ground broken. The mineral was very brittle and high grade. Clean mineral going about 8 oz. gold and while attempts were made to keep split lagging brattices between the working face and the filling, a great deal of fine mineral was undoubtedly blasted into the filling and lost. The footwall was frequently rough and though brooms were provided, their use was frequently neglected.

From this you will see that it is highly probable that much fine mineral and some lumps were necessarily shot into the filling besides what resulted from careless cleaning of the footwall. This may easily have been sufficient to give such average value as to make reworking profitable under modern conditions. For instance, the use of drag scrapers and local separation of the fine and coarse and perhaps some hand sorting - the reject going directly back into the stope and saving hoisting on all but the rough concentrates.

As to the quantity of filling - there should be at least as much as the ore taken out and possibly more, say, 700,000 tons.

The subsidence of the hanging wall has undoubtedly compressed the filling so that some powder will be necessary to loosen it. This should be far less than in the original mining. A certain amount of timber in the way of stulls to support weak hanging walls will be necessary. How much - only trial can tell."

## PRESENT CONDITION OF MINE WORKING

The Congress shaft #1 is blocked at the portal and according to all accounts practically all of these workings, the oldest in the mine, are now caved and inaccessible. At intervals during the past few years leasers have tried to open up small sections in which it was reported that good ore had been left. They have been successful in finding and mining small blocks of ground assaying from $10 to $20 per ton, but only a systematic and expensive reopening of this portion of the mine could give much data as to the present conditions and prospective ore reserves. The old records show that this shaft had a depth on the incline of 1100' and that 227,899 tons of ore were mined.

Along the outcrop of the Congress vein, going east from #1 shaft toward the Queen of the Hills and west toward #2 shaft some good ore has been left near the surface between "gopher holes" and trenches put down by leasers in recent years. It is presumed only a shallow surface sill of ore was left above the old stopes.

On the 650' level a long drift was started to connect the workings from #1 shaft with those in the Queen of the Hills section. The old miners who worked in this area claim that this drift followed some excellent ore (0.5 oz.) for a long distance and also cross cut some promising veins that were never mined. It is probable that their recollection of values is exaggerated for Mr. Staunton was too good a miner to pass up ore such as this. It is highly probable that one of these lost veins could make ore today.

The Congress number two shaft was partially reconditioned some time during the 20's and again by the Congress Mining Corporation who used it to pump out the stored water in the mine. The shaft which is on an incline of about 25 degrees is now open down to the 1150' level at which point the water now stands. At one time during 1939 or 1940 number two shaft was dewatered to the 1925' level.

On the 1075' level caves blocked both the east and west drifts a short distance from the shaft. On the 1000' level low grade ore was observed in the stope fills for several hundred feet east and west of the shaft. On the 925' level at a point 300' west of the shaft a sample cut of 3 feet assayed 0.26 oz. gold ($9.10 per ton). This sample was cut in a lengthy section of unmined vein material. The 1000' level connects with the #3 shaft to the west but several caves block travel through it at the present time.

From the 925' level up, the drifts are nearly all caved except for the 800' level which can be entered for a short distance to the east and here the stopes are large and well filled with gob which looks like fairly good ore. Above the 800' level the drifts are caved but in several places dry walls were observed that would do credit to any mason.

Congress #2 shaft was sunk to a depth of 1700' and connected to number three shaft on the 1000, 1150 and 1700 foot levels and again by a winze from the 1700' level to the 2525' level. The tonnage hoisted through number two shaft was 122,779.

Shaft #3 was the last working shaft on the Congress vein and is 8 feet high and 12 feet wide on an incline of 25 degrees to a depth of 4000'. It follows down a long-well defined hanging wall on which the gouge seems to merge into the Congress vein at depth. This shaft is blocked by a cave at the 1100' level, but above that it is in good condition and could be placed in operating condition without too great expense. It is well timbered with Oregon Pine and most of the timber is in good condition.

The 1000' level was not driven to the west but going east it is passable for over 500 feet. Here the vein follows the dike which is sometimes included in the vein and sometime lies on the footwall or hanging wall. The vein itself shows quartz and sulphides and from the small amount of stoping that was done - this section must have been off the ore shoot or low grade material.

The 650' level extends only a short distance west, but to the east there is 500' of open drift. At 500' east the vein is narrow and mixed with wall rock and a sample from four feet of vein material assayed only 0.03 so it is obvious the old operators recognized this as waste.

Should the mine be reopened for recovery of the low grade ore and stope fills it would be logical to use number three shaft as the main haulage shaft for the Congress and use #2 shaft as a second entrace. Although I believe no one has been below the 2000' level in this mine for over 30 years (1940), it is probable that most of the workings with their walls of hard granite would still be found in fairly good condition. The tonnage hoisted through #3 shaft was 63,524 of which most came from below the 1700' level.

The shafts along the Niagara vein are all in granite and the inclines are about 40 degrees which is steeper than the 20 degree dip of the Congress vein.

The #4 Niagara shaft is caved solid at the collar, but the workings are connected to those from #3 shaft on the 1400' level on the Congress. The number four Niagara shaft was sunk to 1000' and produced 20,470 tons of ore.

There are three number five shafts; "New #5", "Old #5", and "Oldest #5". The "New #5" is opened out around the collar by what appears to have been a small glory hole. The reason for which is not clear. One can get down about 200' on the rather steep incline and here it is blocked by a cave. None of the drifts on either side of this shaft can be entered although the long west drift on the 150' level which was used for haulage to the mill might be opened without too much work as I noticed that a strong current of air passes through it.

The vein is well defined in this shaft and mostly quartz with a little sulphide. To the eye it looks to contain good ore with a width of 3' to 5' and is sometimes pretty well frozen on the granite. It is reported that there is still a lot of stoping ground left near the shaft aside from the pillars which will probably run 0.3 to 0.4 oz. No great expense would be involved in reconditioning this shaft - or cleaning out the caved ground mentioned, although there is likely to be other barriers at greater depths.

Old #5 shaft was entered from an adit since some work was recently done here by leasors who mined a little $15.00 ore. In this shaft it was passable to the 700' level but was caved towards the main #5 shaft. Only small sections of the Niagara vein could be examined but the true width appeared to be 3' to 6', often fingering out in quartz stringers into the granite walls.

The oldest #5 shaft is caved solid at the collar.

The new or main #5 shaft was sunk to a depth of 2050' on a 40 degree incline. It is the only shaft needing reconditioning in the event the Niagara vein is mined. The tonnage hoisted through the main number five shaft was 181,734 and the remaining tonnage of stope fills should be proportionately large.

Number six shaft is caved at the collar but on one of the lower levels, thought to be the 400' level, there is a drift into the MacDonald (Golden Key) mine workings which was opened a few years ago when that mine was pumped out. The ore and stope fills in this west end of the Niagara vein could best be recovered through a crosscut haulage drift driven from the footwall of the Congress vein at some point near the #3 shaft.

The number six shaft was sunk to 1800 feet on a 40 degree incline and the tonnage hoisted through this shaft was 81,016. According to Mr. Staunton the ore was cut off by a fault at the west ends of the stope but the displacement is not too great for sections of ore have been found to the west on the adjoining property.

The main tunnel in the Queen of the Hills workings is caved at the portal and entrance to the workings can be made through an upper tunnel probably on a faulted eastward extension of the Congress vein. The shaft and stopes from the lower level are now inaccessible and the small stopes on the upper levels which were open produced some $20.00 ore.

In this section of the mine the general opinion is that there is a large tonnage of new ore that could be mined to advantage in addition to reclaiming to stope fills. The main shaft of Queen of the Hills was sunk to 200' below the lower tunnel level and produced 20,125 tons.

There are a number of other shafts and adits on the property, all of which are more or less caved.

## OLD MILL TAILINGS

According to the records of the Congress Mining Company there were sent to the dump during its period of operation (from 1895 to 1911), a total of 617,542 tons of mill tailings. Of these, 66448 tons were run out directly from the tables and vanners which followed the stamp and had an average assay of cyanide 0.25 oz. gold per ton. The balance 551,094 tons, were milled in the plants and the average assay was 0.063 oz. gold per ton.

It would appear that in 1911 the tailings dump contained 607,342 tons containing 44,786.122 oz. gold. The silver content of the tailings from recent assays was about 0.4 oz. per ton so on this basis the dump would contain 247,016.8 oz. of silver.

The average assay of gold or 0.063 oz. and silver of 0.4 oz. is equivalent to a gross value of $2.56 a ton at present metal prices.

During 1937 to 1942 approximately 450,000 tons of the old tailings were remilled in a cyanide plant and about 150,000 tons of the original tailings remain to be retreated.

## THE MINE DUMPS

Dumps of low grade ore and waste rock were made by the old company near the collars of each of the seven principal working shafts. During recent years some of the best of this material has been sorted or screened and treated by various parties.

In 1938 the management of the Congress Corporation became convinced that some dump rock should be mixed with the mill tailings to improve recovery and increase the tonnage treated in their mill. Since this plan involved considerable expense for crushing and other equipment - a comprehensive sampling of the dumps was made for the first time by the company. In carrying out this procedure numerous pits were dug in the sides of the dumps from which ore ton samples were taken and these were carefully crushed and quartered down for assay. In a few cases representative lots of from 20 to 50 tons were sent directly to the mill as check samples. The detailed results of this work will be noted in connection with the individual dumps, but the general result was an estimation that over 400,000 tons of dump were available with an average assay of $3.84 in gold per ton.

To further check this estimate, the company employed Henry G. Carlisle, Consulting Engineer of San Francisco. He repeated the previous procedure by digging smaller pits in other portions of the dumps and taking smaller samples which averaged only 100 pounds each.

Carlisle's report confirmed the company's estimate of tonnage but reduced the average grade to slightly less than $3.00 per ton. This is about the average value I obtained in a number of smaller check samples taken sometime later.

The physical character of the dumps and the great variation in the size of fragments composing them makes it very difficult to hand or pit sample them with any degree of accuracy. Therefore, I consider that the best and most reliable source of their value is obtained from the mill records of the Congress Corporation which treated 106,629 tons of dump rock from 1939 to 1942. While assays of the dump rock were not properly segregated from those of the tailings indications are that the average value of the dump rock treated was well in excess of $3.00 after sorting out from 10 to 15% waste rock.

In estimating the remaining tonnage, which is very difficult due to the irregular contour of the surface, it is estimated that 250,000 tons of dump material remains for retreatment.

Details of the dumps are as follows:

DUMP #1 (Congress Vein)
Originally this was a very large dump extending west and northeast from #1 shaft and divided into at least four sections. Three of which have been largely reclaimed so that only irregular fragments now remain from which some 6,000 tons of rock might be taken. These sections were largely worked by the Wymans and Jay Burns, and they are reputed to have assayed $6.00 or better per ton.

The main section located in a gulch was worked by the Congress Corporation and the upper portion was scrapped off with a bulldozer into a trap which is still in fair shape. My calculations indicate that about 12,000 tons should still be reclaimed and this figure is confirmed by Ramsden. Carlisle's sampling averaged $3.00 per ton for 21 samples. Liddell took 20 samples which averaged $3.72 gold and 0.02 oz. silver.

The Congress Corporation appears to have milled approximately 50,000 tons of rock from this dump which is reported to have averaged over $3.00 per ton.

From all the above and considering that the remnants of the smaller sections of the dumps are doubtless higher grade than the main section, it would seem safe to estimate that in #1 dump there are still 18000 tons that will average after sorting close to $3.50 per ton - plus 10¢ silver.

DUMP #2 (Congress Vein)
This was a very large hillside dump with a maximumheight of over 80 feet. A portion of which were screened out and treated by the Wymans and Jay Burns. The total tonnage taken from the original dump has probably been 40,000 but the Congress Corporation milled no rock from here except for a test lot of 50 tons which is reported to have run slightly higher than the average of their samples. Calculations of the remaining tonnage place this at 90,000 tons of sorted ore. The road to the dump is good and reclaiming should be easy.

The average grade as first determined from 28 samples taken by Liddell was $3.76. When Carlisle checked this with 22 samples, his average was only $2.84 which checked closely with my own grade samples. Adding the value of the silver the gross value should be very close to $3.00. The haul to the mill is about 600 yards.

## DUMP #3 (Congress Vein)

Dumpe #3 was reclaimed to some extent by Burns and the Congress Corporation merely treated some of his screenings which had been left near by and which are said to have carried $7.00 per ton. The contour of the dump is very irregular and in calculating the tonnage I have been conservative in placing it at 80,000 tons.

Liddell's average of 30 samples was $3.45 and Carlisle apparently did not sample this dump so that we do not have as much data as in the other cases. My own grab sampling of this dump was done twice and the average results were only $2.28 and $2.00 but I carefully noted the pits which had been made by Liddell and his results should have been much more accurate than mine. Moreover it is hardly likely that Jay Burns would have treated so much rock from this dump when all of the dumps were still intact unless he had found it to be at least equally as rich as the others.

Many of the fragments in my samples were barren granite and diorite, some of which would normally be sorted out on a picking belt and I think that I am conservative in estimating the average value at $2.50 per ton. Reclaiming this dump will be comparatively inexpensive but the rock will have to be trucked around the point of a hill to the mill site - a total distance of about 800 yards.

## DUMPE #4 (Niagara Vein)

The #4 dump is small and locally is supposed to be rich, but Liddell's 19 samples averaged only $3.50 and my samples assayed somewhat lower. None of this rock was milled by the Congress corporation.

The length of the dump is 186' with axis N. 15° E. but at the top it is in places - only 6' wide and elsewhere the surface rocks project up thru it so that the tonnage is very difficult to figure. An estimate of 5000 is conservative. Some portions of this dump would be hard to reclaim, unless they could be sluiced down the very steep hill. The haul to the mill is about 800 yards.

## DUMP #5 (Niagara Vein)

The dump originally contained over 100,000 tons and since I am reliably informed that the surface beneath it is a gulch, I accept the estimate of the remaining tonnage made by Rockwood and Ramsden as 50,000 ton of sorted ore. Like Dump #2 the present contour is most irregular.

The average of 10 samples taken by Liddell was $3.97 while the 19 samples taken by Carlisle averaged $2.80, and my grab samples averaged $2.60. 40,000 tons of rock was drawn from here as was from #1 Dump at the time that they closed down.

I have found records which showed that much of the material milled from here assayed $4.20 per ton and Rockwood and Ramsden are both very positive in estimating that the average grade was over $4.00 and that the remaining rock should be equally good. However, these statements do not seem to check with the general mill records. However, in all probability it is of somewhat higher grade than #2 and #3, I think it safer to figure the average at $3.50.

The upper part of the dump was mined with a power shovel which loaded the rock directly into trucks while the lower portion on the south side was scraped by a bulldozer to a trap which is now in poor condition. The haul to the mill is about 500 yards.

DUMP #6 (Niagara Vein)

The larger portion of this material was piled on the Rose Quartz and Los Senate Claims which do not belong to the Congress Company. The greater part of the rock that was on the Why Not Claim of the Congress Company was reclaimed with a drag line and treated by the Congress Company to the extent of about 4,000 tons. It does not appear that more than 2,000 tons remain to be taken from the Congress ground, although probably 8,000 to 10,000 tons are still left on the adjoining property. The average grade of recent shipments made by Findley to the Hayden Smelter was $7.80 per ton. I was told that the material taken out by the Congress Company was not so good, but averaged over $5.00 per ton which is the value that I place upon the remainder which I did not sample on this occasion although I had done so some years ago. The haul to the mill is about 1000 yards.

QUEEN OF THE HILLS DUMP

Five samples taken by Liddell averaged $2.10, but Carlisle's lot of 11 samples averaged $5.07 which does not check unless they represented two different dumps as there were originally 3 or 4 of them. My samples from the largest of the remaining dumps ran $4.20, and I have estimated the grate at $4.00.

My estimate of tonnage in the Queens Dumps is 5,000 which will be rather expensive to reclaim. The Congress Mill appears to have treated about 12,000 tons concerning the grade of which I could obtain no details except that according to Ramsden, it was richer than the average. The haul to the mill is over 1200 yards.

Concerning all of the dumps it may be said that appearances indicate that the material is largely a mixture of quartz and pegmatite, both of which doubtless carry good value. Granite and diorite which look to be practically barren except as they are enriched by little veinlets or seams of quartz and sulphide that are scattered through many of the fragments.

The value of the dumps is therefore largely dependent on the relative percentage of waste rock to low grade ore and this varies in each of the dumps and in different portions of the same dump so that sampling is very difficult and it seems to me quite remarkable that the pit samples taken by Carlisle and Liddell should have been so closely checked and generally improved upon by the mill runs which constituted by far the most accurate sampling.

A summary of my estimate of the most essential data concerning the dumps is shown in the following summary:

## SUMMARY OF DATA REGARDING CONGRESS DUMPS

| DUMP | Estimated Gross Tonnage 1937 | Average Grade of Pit Samples | | Record of Rock Milled by Congress Corp.** | |
|---|---|---|---|---|---|
| | | Liddell | Carlisle | Approx. Tonnage | Value Per Ton |
| #1 | 75,000 | $ 3.72 | $ 3.11 | 50,000 | $ 3.00+ |
| #2 | 105,000 | $ 3.76 | $ 2.84 | None | |
| #3 | 100,000 | $ 3.45 | | None | |
| #4 | 5,000 | $ 3.45 | | None | |
| #5 | 100,000 | $ 3.97 | $ 2.80 | 40,500 | $ 3.50+ |
| #6 | 7,000 | $ -0- | | 4,000 | $ 5.00+ |
| Queen of the Hills | 20,000 | $ 2.10 | $ 5.07 | 12,000 | $ 4.00 |
| | 412,000 | $ 3.84 | | 106,500 | $ 3.00+ |

** Allowance of 20 per cent for material to be sorted out.

| DUMP | Estimate 1943 Remaining Tonnage | Approximate Grade of Sorted Rock |
|---|---|---|
| #1 | 18,000 | $ 3.50 |
| #2 | 90,000 | $ 3.00 |
| #3 | 80,000 | $ 2.50 |
| #4 | 5,000 | $ 3.00 |
| #5 | 50,000 | $ 3.50 |
| #6 | 2,000 | $ 5.00 |
| Queen of the Hills | 5,000 | $ 4.00 |
| | 250,000 | $ 3.00 |

## MINE FILLS (GOB)

From Mr. Staunton's description of the method of mining given on previous pages of this report, the reasons for the existence of so large a tonnage of gob and for a logical assumption that it has a high value will be evident.

The pay streak in both the Congress and Niagara veins usually were three feet wide but in order to permit economical mining -- the stopes were intentionally broken to approximately double that width and at times, due to the brittle character of the hanging wall, the width was even considerably greater.

Insofar as can be learned from the records and from personal examination of the accessible workings of the mine, all of the stopes were back filled and this back filling is still in place. It may be that below the water level much of this gob will be compressed and re-cemented as Mr. Staunton suggests. This is not the case in any of the stopes which I was able to inspect for in most stopes the gob still would flow freely when moved.

The maps of the underground workings give a general idea of the extent of the filled stopes in which my very rough calculations confirm Mr. Staunton's estimate of about 700,000 tons. It is not possible at present to determine whether all of this material could be economically reclaimed.

Since 1934 various engineers, including Mr. Colburn and Mr. Ramsden have taken many samples more or less at random and these, they have told me, averaged well over $5.00 per ton.

In 1939, Carlisle took 7 samples on an unidentified level of number 2 shaft (Congress vein) the poorest of which ran $1.40 while the highest ran $8.56; the average being $4.05. From the 1000' level in number two shaft, six gob samples were taken and varied from $21.10 to $10.03 and averaged $6.49 per ton.

Seventeen samples taken from the gob in stopes on the Niagara vein assayed $1.51 to $9.45 and averaged $4.96 per ton.

One sample taken from gob in the Queen of the Hills workings assayed $7.35 per ton.

I think it fair to say that present indications point to an average value of $5.00 per ton, particularly when it is considered that the highest grade material is in the fines which were left on the floor where it was not likely to be included in any of the samples.

From samples taken to date all of the gobs can be considered probable ore and could be recovered cheaply by the use of slushers and scrapers. Some underground sorting would seem to be advantageous while the final cleaning of the floors of the stopes on which the high grade fines have concentrated might possibly be accomplished by sluicing as is done in hydraulic pits.

With the mine reopened and equipped for operation it seems that $1.00 per ton would be a very liberal estimate of the average cost of reclaiming this gob material and placing it in the mill. Its character is very similar to that of the sump rock and it would appear that treatment by flotation and cyanide should recover 85 percent of the gold and silver values. A very substantial amount of profit is indicated to be represented in the old mine fills.

## PILLARS, SILLS AND LOW GRADE ORES

Except in the immediate vicinity of the shafts and main levels, it is unlikely that any substantial pillars or sills of high grade ($20.00) ore remains in any part of the old workings and while small portions of these may be recovered when drawing out the gob, it does not appear that enough tonnage would be recovered to warrant the estimation of this tonnage.

The situation in respect to low grade ore is very different for in previous operations - every effort was made to keep up the average grade of production and with the gold price at $20.00 per oz., no vein material containing less than 0.35 oz. gold per ton was intentionally mined.

According to the statement of previous operators, particularly Mr. Staunton, there was a very large but undetermined quantity of low grade ore partially opened up in many places of the mine, especially at the ends of the higher grade ore shoots. It was never developed and no measurements or sampling were done. To quote Mr. Staunton in substance:

> "Just as soon as the grade of the vein fell to $7.00 (0.35 oz.) we dropped the stope and went elsewhere or, if we had to go through it we left the below $7.00 material in place."

That several remaining sections of the vein did contain such low grade is undoubtedly a fact, and this has repeatedly been emphasized by Mr. Staunton - although he had always refused to make any estimate of tonnage or average grade.

During my recent examination of the accessible workings of the mine, I noted many sections of the vein adjoining the filled stopes where ore was left, had similar appearance to that which was left in the pillars. The only difference was that the ore left contained less sulphides and was therefore undoubtedly lower grade. Elsewhere the character of the vein was different, being deficient in quartz and still less sulphides and this material from samples taken would indicate that it would not be ore classed as ore even at the present gold price.

In the upper levels, leasers, working during the past few years have mined and are still mining ore assaying from $10.00 to $20.00 per ton. In many of these workings it appeared to me that a considerable tonnage could still be mined if it were possible to obtain a profit on $7.00 ore.

Messrs. Colburn and John Price have told me that they have taken many samples of vein material which assayed better than $10.00 per ton. Percy Ramsden claims to have taken some 300 samples of ore remaining in the Congress vein between #2 and #3 shafts from the 1000' level to the 1925' level and these samples averaged better than $8.00 per ton.

The above material to which I refer is sufficiently developed to be classed as positive ore or at least highly probable. Unfortunately it has never been systematically sampled or measured. Such a program is not now possible as the mine is flooded to the 1100' level. However, some resampling of the low grade ore for a moderate expense and in my opinion, should be well worth while.

As to any estimate of tonnage and value, this can be at best, merely a guess. However, from what I have been able to see and learn from others, who were familiar with the old workings and from the study of the maps, I think that it is a very conservative estimate and a strong probability that there remains 200,0 tons of $8.00 ore to be mined within the limits of the old workings.

Fortunately any work done to determine the character and value of the gobs and low grade ore would also throw a great deal of light on the possibilities of finding entirely new ore bodies.

## NEW ORE POSSIBILITIES

Obviously the management of the Congress Mine did not intentionally overlook any likely ore prospects in the 20 years they actively operated - nor did they close down until they were convinced that continuance of operations would no longer be profitable. Almost from the very start the mine was well financed and managed by able men who adopted the most improved methods of mining and milling. However, it does not appear that at any time did they give a great deal of attention to economic geology, nor did they ever employ a geologist of recognized standing to survey the mine and possibly suggest new areas for exploration.

Mr. Staunton and his successor, Mr. Meade Goodloe, were unquestionably the most skillful and efficient mine managers of their time but work was guided by experience and intuition, rather than scientific theory. In following down the ore shoots in the two main veins, their problem was relatively simple but they left, unsolved, the faults at both ends of those veins between them they may -- as Mr. Staunton readily admits, have overlooked a number of blocks of ore dislocated by many minor faults.

As to the numerous off shoots of cross-veins and which in aggregate produced a large tonnage including some of the highest grade ore, it seems that these were usually developed by following only the most promising of the many stringers of quartz which branched from the main vein into the hanging wall. Some of these cross-veins proved to be barren but Mr. Staunton has told me that he now thought that they should have followed a large number of these cross-veins.

Messrs. Brooks and Colburn were also much impressed with the possibilities of further exploration in the hanging wall and such is my own opinion although it is obviously very difficult to draw any conclusions at present when so much of the underground workings cannot be examined and the outcrops of the veins are often covered with buildings or surface dumps.

Mr. Staunton has particularly recommended that exploration should be conducted at a greater depth below #1 shaft on the Congress vein and near the 1700' and 2500' levels from #2 shaft. He well remembers a large body of low grade ore was left in this area. This is the area mentioned as being sampled by Mr. Ramsden.

Mr. Staunton does not believe any large new ore body in or near the Congress vein will be found below the 3000' level as this area was very thoroughly explored with disappointing results. He does think however, that additional ore may well be found on the upper levels of the Congress vein. His theory was that the Congress vein was mineralized from a quartz cross-vein from which solutions followed along the fissure formed by the intrusive dike where chemical conditions were favorable for the deposition of metals.

In regard to the possibilities of further discoveries of ore in depth on the Niagara vein, Mr. Staunton is more optimistic and regarding the general problem of future exploration has written as follows:

"Underlying the Niagara vein, which strikes east-west and dips about 40 degrees there is a greenstone dike with a slightly different strike and with a dip of 25 degrees. This dike is almost identical in character with the Congress dike which carried the ore in the mine. The Niagara vein intersected this dike at about the 1975' level in the extreme easterly part of the mine close to the big fault. The dike was heavily mineralized at the intersection and the ore in the dike was the same character and grade as in the Congress vein. This high grade ore extended east to the big fault - where it was cut off. To the west, the work was first confined to the dike, but as the distance from the intersection increased, the grade became lower. Crosscuts were then run into the hanging wall of the Niagara vein and thereafter all work to the west was done on the Niagara vein. The line of the dike and the Niagara vein intersection runs downward to the northwest.

The #5 Niagara shaft is an incline on the vein and its course happens to coincide closely with the course of the dike and the Niagara vein intersection. It seems highly probable that a new line of high grade stopes can be opened by sinking the #5 - Niagara shaft below its present depth of 2050 feet. The 1900' level is connected to the Niagara #4 shaft, located 700 feet to the west. This would provide good ventilation. Sinking of this character is comparatively cheap as it amounts to little more than running a drift on an incline. The little water encountered may be readily bailed.

On the surface, about 200 feet west of the Congress #1 shaft, a vein entirely in the granite and locally known as the "cross-vein" intersects the Congress dike vein and the position of the Congress ore shoot roughly corresponds with the line of intersection of these two veins. The part of the "cross-vein" in the hanging wall of the Congress vein had considerable stoping ground, the footwall section had less ore. The "cross-vein" ore pinched out or was lost as it approached the big fault. In other sections of the mine in this area heavy bodies of high grade ore were cut off cleanly by the fault.

It was thought at one time that we had found the measurements of the throw of this fault on the 650' level. The east drift on the "cross-vein" encountered the fault and after going through about 40 feet of fault breccia, a vein, looking like the Congress vein was found. However, the vein at this point did not make ore.

The existence of the big fault and the fact that in several cases good ore was cleanly cut off by it - naturally suggests that ore of comparable size should be found to the east of the fault. Work done with that in view was not successful as it was not carried far enough.

My recollection is that on the 3125' and the 3200' levels and perhaps the 2750' level in the #3 shaft, something was found east of the fault that looked like the Congress vein. If this is true it would indicate a much less throw on the fault than on the surface."

In 1917 the Congress Mine was examined by Edward W. Brooks, a geologist, from Los Angeles. He concluded that the veins were what were termed the "pegmatite type" and the gold was almost entirely associated with arsenical pyrites. He considered that the mine was worked out (at the old price of gold) down to the limits of the workings, with one very important exception. Mr. Brooks stated that the length of both the Congress and Niagara veins was 3500 feet within the property lines which probably represented their extreme limit of the pay ore but he refers to an upper - or spur vein which had a length of 1500 feet within the property. He also states that all ore shoots pitch to the north in all veins.

Brooks then went on to recommend that future development should be undertaken from the #3 Congress shaft and concentrated on the exploration of the Congress vein below the 1250' level and work should be done on the spur vein from the surface down to the 2700' level where it intersected the Congress vein. As no work was done on the spur vein - he reasoned that the spur vein was still virgin ground and that a similar condition should maintain below the 2700' level in the Congress vein. His recommendations were based on the belief that the vein encountered in #3 shaft at the 2700' level must have been the Spur vein and not the Congress vein because it came into the shaft from the hanging wall. Brooks calculated the unworked sections of these two veins should contain some 300,000 tons (allowing an average width of 3 feet) and that this ore should have an average value of 0.6 oz. gold per ton. This is based on a diamond drill hole which cut the vein.

In event the mine is reopened and pumped out to the 2700' level this matter should be carefully reconsidered and in the meantime some further inspection should be made of the surface outcrops of the spur vein.

Mr. Staunton further adds that across the Bellick and Queen of the Hills claim and extending over the MacDonald claims - to the west, there is a wide mineralized dike which is reported to carry gold values of 0.1 oz. and better along the surface. If these values can be substantiated a very large tonnage of ore susceptible to cheap mining could be developed.

Insofar as records show no systematic diamond drilling was ever done on the property and from recent analysis of the property - the lack of diamond drilling could very well be the reason many new ore bodies are left to be found.

## CONCLUSION AND RECOMMENDATIONS

Considering the risks which are involved in any mining enterprise the expected profit from the treatment of the tailings and dumps, classed as positive ore would be insufficient to make the venture attractive. However, in this case it serves to eliminate nearly all the risk excepting that which depends on the price of gold. Retreatment of the tailings and dumps should insure the return of initial investments even if all further investigation and exploration should be entirely disappointing.

In the category of "probable ore" I have placed the mine fills and a certain amount of low grade ore left by former operators. The tonnage and grade of both these classes of materials has been fully discussed in the body of this report and more accurate information can only be obtained by further investigation.

Should these investigations confirm the existing indications the additional quantity of ore which could be classed as positive or highly probable should amount to from 800,000 to 1,000,000 tons with an average grade of about $6.00 per ton. A large percentage of this material would consist of broken ore in the form of mine fills and mining costs would be relatively low. Immediately after the above confirmation of the ore values and tonnages in the gobs and low grade areas, plans should be drawn up for the construction of a mill of at least 500 tons daily capacity.

The possibilities noted in the above paragraph make this venture attractive and while the future possibilities of the mine are still too nebulous to justify any figures, I think that there are reasons for believing that upwards of 500,000 tons of additional ore may be developed. Should the grade of the additional ore be only $11.00 per ton or one half as good as the ore that was mined in the past, a net profit of over $3.50 per ton should be easily earned from its exploitation.

If the property is to be acquired at all, it should be done in the very near future even though this will involve some risk in respect to the economic position of gold after the close of the war.

In attempting to fairly evaluate both the favorable and unfavorable facts of the Congress Mine I have reached a firm conclusion that this presents an exceptionally favorable mining venture. On that basis I strongly recommend that the property be acquired and that plans be made to resume mining and milling as soon as conditions will permit.

Very truly yours,

G. M. Colvocoresses

(copy)

August 30, 1948

Messrs. Colburn, Byron Moyer, Richard Heilmann and Associates.

SUPPLIMENTARY REPORT ON CONGRESS MINE

Gentlemen:

Although my examinations of the Congress Mine in 1935, 1942, 1943 and 1944 were made first for the purpose of determining the advisability of treating the tailings and dumps and next with the idea of conducting a large scale operation for the treatment of tailings, dumps, mine fills and low grade ore in place;-the results as embodied in my long report of August 1943, to which I later made some additions, included much information which would be valuable in guiding a small operation treating only the higher grade ore. In this connection the following supplimental notes are submitted and in order to call particular attention to portions of my reportwhich have a bearing on your present problems, I have side lined certain paragraphs in pencil and also made a number of notes on the margin of the copy with which I shall furnish you.

1. The remaining tailings will probably not pay to work, but it will be advisablw to investigate the sub-soil where some samples which we took in 1943 showed material carrying $18.00 per ton; but this may since have been removed.

2. As to the dumps while these will not average much over $3.00 per ton, there are some portions which might be worth sorting over and perhaps screening since it is of record that on one occasion 147 tons were sorted from a dump and assayed $18.55 per ton. Some of my samples from the dumps ran better than $10.00 per ton but these were averaged with lower grade material.

3. From all sampling it appears that the mine fills down to the 1500 level will average at least $5.00. Here again there are sections which are much richer than the average, but we purposely cast out high grade samples. Some idea of the probable value of each portion of the fill can generally be obtained by visual inspection after washing, as the presence of quartz and sulphides nearly always indicate gold values except in the Queen of the Hills where there is a lot of nearly barren quartz. Before actually preparing to mine any of this material, I suggest that some grab samples should be taken from the best looking sections as I ontained several samples which ran better than $8.00 per ton particularly from the Congress vein near No. 2 shaft on the 925 ft. level.

4. The mine was unwatered in No. 2 shaft to the 1950' level in 1941 and Ramsden told me that he found some very good ore on the 1925' level in No. 3 shaft and extending up to the 1700'.

HIGH GRADE ORE

The existance of high grade ore remaining in the old Congress workings has been made the subject of persistant rumors many of which I believe to be unfounded or greatly exaggerated. However, as a matter of record, I think it proper to repeat some of these for what they may be worth.

Near to the surface, especially in the vicinity of thb No. 1 shaft there still remain small sections of pillars and sills of high grade ore some of which has been goughed out by leasers during recent years and shipped to the Hayden Smelter after the Congress Corporation shut down their mill. This ore was difficult and expensive to mine and probably had an average value in the order of $20.00 per ton but was hand sorted until the shipping product becomes much richer. Profits to the miners seem to have been small since the work was intermittent and had been practically discontinued during the latter part of 1944. The tonnage of such ore now remaining in this section of the mine must be small and cannot be considered in any estimate although it is quite likely that after regular mining and milling operations are resumed, new leasers may furnish a few hundred tons or more per annum.

Frank Stone of Prescott who did some leasing on the mine claims to have climbed up 90' in an inclined raise from the 600' to the 700' level east of No. 2 shaft and there to have found and sampled a cross vein with a width of 18" to 2' that carried up to 4.00 Oz. of gold. This story was related by Herscowitch who apparently does not know just where the raise is located and in any event it is probably now inaccessible except after some preparitory work.

Regarding the Queen of the Hills workings, Stanton could give little information since most of this work was done after he was no longer manager of the mine, but it is of record that leasers operated here with some profit during the 1930s and that the mine workings are much more extensive than shown on the map and at one point a winze had been sunk to a depth of 1750'. Samples taken in some of the pillars ran better than $10.00 per ton and several reliable men stated that a substantial tonnage of similar grade of ore remained in the sections of the vein which they had examined prior to 1940.

It is my opinion that a comparitively small amount of cleaning up would permit the examination of much of these workings which are now inaccessible and I suggest that special attention should be paid to the QUEEN of the Hills which seems to have been much less thoroughly prospected than other portions of the property and which probably contains the faulted extension of the Congress Vein.

LOWER GRADE ORE

As to the lower grade of ore remaining in various portions of the workings, I can add nothing to the previous statements and those in my report except to mention that Snow confirmed the findings of Colburn, Price and Ramsden to the effect that many samples cut in the vein between the old stopes would run from $7.00 to over $10.00 per ton and some of my samples carried over $9.00 although all high grade material was purposely avoided.

Stanton and others who were familiar with the old mine mentioned the fact that there had been left in the upper levels of the Niagara vein ore which assayed just a shade below the old limit of 0.35 oz. per ton and which could be reached for sampling from the #5 shaft if a little cleaning up was done in the shafts and drifts. While no accurate estimate of this ore was made, the tonnage was represented as being quite substantial and some portions of it would carry better than 0.40 oz. per ton.

METALLURGY

In further reference to the treatment of the higher grade ore to be produced from small scale operations (about 50 tons per day) it seems that this might best be started through the use of flotation with shipment of concentrates to a smelter. These concentrates and even the flotation tailings could later be cyanided (with or without roasting if the extra recovery would make this worthwhile.

I have a record of gravity concentrates shipped by the Congress Company to the Humboldt Smelter in 1906 which carried 7 to 8 oz. in gold and 13 oz. in silver, and I believe that a high recovery of values could be obtained on most of the ore by modern flotation alone whereas the installation of cyanide equipment would involve much extra expense and the operation of a small cyanide plant is comparitively costly.

CONCLUSION

To sum up the situation I would call your attention to the estimate of ore reserves given on page 104 of my long report and especially the possible ore amounting to 200,000 tons with an average gross value of $11.00 per ton. Neither the quantity nor grade of this material can be made the subject of an engineering estimate based upon mathematical data but it is based largely on conversations and correspondence with competent engineers who were familiar with the old workings of the mine and it has been checked to some extent by my own findings and those of other engineers who assisted me or were associated in our investigations.

Of course all cost estimates in the long report must now be substantially revised upward and without attempting to go into detail I have figured that the cost of developing, sorting and mining ore will be in the order of $6.25 per ton in place of $5.00 and milling, etc. will cost about $1.75 making a total operating expense of $8.00 and leaving a profit of $2.00 per ton on this class of material if an average recovery of $10.00 can be obtained.

The total expected profit from the operation, after deducting the repayment of capital which must be invested, may not seem to make the venture particularly attractive, but one must bear in mind the chance (and I think it is a very good one) of finding and mining considerable ore of a much higher grade some of which may well run to a value of $20.00 as produced in the old operations and considering that all mining is at best a speculative venture, I feel that there is a strong probability that the initial investment will be repaid and a reasonable expectation that a very substancial profit may be earned either because of the development of higher grade ore or because of the anticipated increase in the price of gold. Moreover it should be noted that all of the samples listed in the report were purposely taken without sorting while such sorting, at a comparitively small expense, would have raised the grade of those taken from ore in place and also from portions of the gob from 15% to 30%.

In carrying on a small operation you will doubtless find it advisable to apply both selective mining and sorting and may thus be able to bring the average value of mill heads to perhaps $13.00 per ton which would be most desirable.

FIRST PROCEDURE

The condition of the workings which I visited and sampled some five years ago has doubtless changed somewhat for the worse and before mining and milling is actually started, it will be essential to have made accessible a sufficient number of faces of pay ore or gob to permit the desired daily production.

In order to be reasonably sure of producing 50 tons of pay ore or gob I feel that first of all, and before making any large purchases of equipment, it would be your best policy to employ a competent young engineer with a small crew of miners who could work under the direction of Mr. Colburn, clearing out the drifts or stopes where pay is known or believed to exist and re-sampling these ore shoots or sections of the gob with proper sorting followed by preperation for for actual extraction of the pay ore.

By following this program you should be able to avoid the mistake of going to more than a trivial expense in preparing to develop and mine ore shoots which are not sufficiently rich to pay the working costs.

CAPITAL EXPENDITURES

I have refigured the capital expendbture which will probably be involved in this undertaking and assuming that you can purchase the present power plant for $12,500.00 and obtain good second hand machinery for your other principal items of equipment, I think that the $80,000.00 which you propose to provide should be sufficient provided you do not attempt to cyanide either the concentrates or tailings from the flotation plant. Should such cyanide treatment prove to be necessary or advisable, I believe that you should arrange to have available an additional $10,000.00 or preferably $20,000.00 which last figure would raise your total capital investment to $100,000.00 and serve to provide a certain amount of working capital which is often of great importance.

My conclusions are again made on the assumption that there will not be any further advances in the cost of labor or other commodities resulting in serious inflation with a decrease in the value of our currency which would make present estimates entirely worthless as long as the value of gold is fixed at $35.00 per ounce.

Yours very truly,

GMC:IM (signed) G. M. Colvocoresses.

Samples of the Congress Mine
by Byron Moyer

| | |
|---|---|
| **3-15-48** | |
| 1 - McKinley Tunnel east of No. 2 shaft | $ 9.50 |
| 2 - " " above tunnel | 11.80 |
| 3 - " " 20' below tunnel | 15.20 |
| 4 - " " Hanging wall | 12.00 |
| 5 - " " west of Stope | 14.10 |
| 6 - Level grab east of No. 2 shaft | 10.50 |
| 7 - " " " " " " | 11.30 |
| 8 - 300' level grab on floor of No. 2 shaft | 18.10 |
| 9 - " " " " " " " scr. thru 1" | 18.20 |
| 10 - " " Sulphide | 11.20 |
| 11 - 650 level grab east of No.2 shaft | 12.00 |
| **3-19-48** | |
| 12 - 650 level grab west of shaft No.2 shaft | 11.10 |
| 13 - 750 " " at shaft " " | 9.40 |
| 14 - 750 " gob east of shaft " " | 15.20 |
| 15 - 250 " quartz " " | 11.80 |
| 16 - 350 " " " " | 13.00 |
| 17 - 750 " " " " | 21.20 |
| 18 - 1000 " " No. 3 shaft | 12.50 |
| **3-21-48** | |
| 1 - 250 Level No. 2 shaft Heavy Sulphide | 68.80 |
| 2 - 650 " " " banded sulphide east | 51.62 |
| 3 - " " " " sulphides in hanging wall | 35.45 |
| 4 - 250 " " 5 " " top in cave | 31.15 |
| 5 - " " " " " " bottom of cave | 28.38 |
| 6 - 1000 " " 3 " " east | 63.56 |
| 7 - 300 " " 2 " banded sulphides east | 54.71 |
| **4-6-48** | |
| 1 - surface above No.2 shaft new vein | 20.60 |
| 22 - " dump at No. 3 shaft loose quartz on dump | 31.50 |
| **3-15-48** | |
| 1 - Tunnel level old No. 5 shaft at fault | 26.75 |
| 2 - " " " " | 9.50 |
| 3 - " " " " | 8.25 |
| 4 - " " " " loose quartz | 11.00 |
| **4-20-48** | |
| 1 - 750 Level No. 2 shaft grab east | 26.50 |
| 2 - 800 " " " " west | 13.20 |
| 3 - 500 " " " quartz hanging wall | 15.50 |
| 4 - 1000 " No.3 " grab | 11.20 |
| 5 - " " " " hanging | 12.00 |
| **5-1-48** | |
| 1 - Queen of the Hills Upper Tunnel lower | 8.00 |
| 2 - " " " float | 12.35 |
| 3 - " " " middle | 9.20 |
| 4 - " " " upper | 8.95 |
| **5-6-48** | |
| 1 - 1000' level No. 3 shaft | 26.18 |
| 2 - " " " " sulphide on gob | 19.72 |
| **5-2-[illegible]** | |
| 1 - grab of fills | 21.00 |
| 2 - " " " | 11.90 |
| 3 - " " " | 6.30 |
| 4 - " " " | 5.60 |
| 5 - " " " | 4.55 |
| 6 - " " " | 20.30 |
| 7 - " " " | 7.70 |
| 8 - " " " | 11.90 |
| 9 - " " " | 44.90 |

IN REPLY REFER TO: #1

UNITED STATES
DEPARTMENT OF THE INTERIOR

Geological Survey

Date: June 24, 1955

# Laboratory Report

Name: Mr. W. A. Murray
3410 17th Street, N.W.
Address: Washington, D. C.

Laboratory Identification No. RW-9644

Total Weight: 9 lbs.

Location of Sample: Arizona

Type of Material: samples numbered 3 and 4 - schist and vein material with liebigite

Results of examination: (Items apply only if block /__/ is marked with ✓; do not apply if marked na (not applicable)).

1. /na/ Radioactivity is low; the material is currently of no value as a source of uranium.

2. /na/ Minerals indicate they are from pegmatite. The radioactivity is low, moderate, high. Many deposits of this type contain small concentrations of radioactive minerals byt very few contain them in quantities sufficient to justify mining for radioactive constituents alone.

3. /na/ Equivalent uranium by radioactive determination is xxxxxxxxxxxx %.

4. /✓/ Uranium by chemical analysis (true value) is (3) - 2.4 (4) - 0.51 %. The attached circulars explain the specifications as to grade and quantity for mine and concentration products.

5. /na/ Excess of equivalent over chemical uranium (true value) indicates the higher radioactivity value is caused by thorium or other radioactive elements. No Government schedule exists now for general purchase of thorium-bearing minerals and ores, although thorium may some day be in demand for the Atomic Energy program. See attached explanation of "equivalent uranium."

6. /✓/ The information requested on the attached form would be appreciated, especially location, size, and type of deposit from which the sample was taken. A stamped and addressed envelope is included for your convenience in sending it to Mr. A. P. Butler, Jr., U.S. Geological Survey, Bldg.25, Denver Federal Center, Denver, Colorado. Any information furnished will be kept in strict confidence.

7. /✓/ All material submitted will be held for a period of 30 days from the date of this report after which it will be discarded unless you request that it be returned.

Please refer to the laboratory identification number if you should have occasion to write about this sample or report.

Approved:

Atomic Energy Commission Program
Geochemistry and Petrology Branch

UNITED STATES ATOMIC ENERGY COMMISSION
DIVISION OF RAW MATERIALS
SALT LAKE EXPLORATION BRANCH

Address reply to:
U. S. Atomic Energy Commission
P. O. Box 4336, Commerce Station
Phoenix, Arizona

DATE June 7, 1955

E. A. Colburn
Box 152
Wickenburg, Arizona

Dear Sir;

The samples which you recently submitted to this office have been radiometrically assayed with the following results:

| Sample No. | | $\%eU_3O_8$* |
|---|---|---|
| *A-3625 | 1000 East drift, Shaft and 40ft. Select | 0.43 |
| *A-3624 | " " " " " 55ft. East | 0.88 |

* Sent to the USBM for chemical assay.

Very truly yours,

Millard L. Reyner, Chief
Phoenix Sub-Office
Salt Lake Exploration Branch
Division of Raw Materials

*e - Radiometric, per cent equivalent $U_3O_8$

UNITED STATES ATOMIC ENERGY COMMISSION
DIVISION OF RAW MATERIALS
SALT LAKE EXPLORATION BRANCH

Address reply to:
U. S. Atomic Energy Commission
P. O. Box 4336, Commerce Station
Phoenix, Arizona

DATE 27 June 1955

Mr. E. A. Colburn
Box 152
Wickenburg, Arizona

Dear Sir:

The results of your samples which we submitted to the U.S. Bureau of Mines, Tucson, Arizona, have been reported as follows:

| NUMBER | %e$U_3O_8$* | %c$U_3O_8$** |
|---|---|---|
| A-3625 | 0.43 | 0.92 |
| A-3624 | 0.81*** | 1.60 |

***Contains thorium

Very truly yours,

Millard L. Reyner, Chief
Phoenix Sub-Office
Salt Lake Exploration Branch
Division of Raw Materials

*e -Radiometric, per cent equivalent $U_3O_8$

**c-Chemical, per cent $U_3O_8$

UNITED STATES ATOMIC ENERGY COMMISSION
DIVISION OF RAW MATERIALS
SALT LAKE EXPLORATION BRANCH

Address reply to:
U. S. Atomic Energy Commission
P. O. Box 4336, Commerce Station
Phoenix, Arizona

DATE 16 Mar 1955

Mr. E. A. Colburn, Jr.
Box 152
Wickenburg, Arizona

Dear Sir:

The results of your samples which we submitted to the U.S. Bureau of Mines, Tucson, Arizona, have been reported as follows:

| NUMBER | RADIOMETRIC | CHEMICAL |
|---|---|---|
| A-2973 | 0.10 | 0.14 |
| A-2974 | 0.10 | 0.14 |
| A-2976 | 0.04 | 0.10 |

Very truly yours,

Millard L. Reyner, Chief
Phoenix Sub-Office
Salt Lake Exploration Branch
Division of Raw Materials

# ASSAYS AND SHIPMENTS FROM CONGRESS MINE SINCE 1947.

************

with returns from carload shipments from the Queen of the Hills claim during 1936 as below:-

| Date | Shipper | Lot No. | Gold Ozs. | Silver ozs | Cu% |
|---|---|---|---|---|---|
| 8/13/36 | Queen & Sons | 954 | 0.81 | 1.7 | 0.20 |
| 9/3/36 | " " | 1006 | 0.69 | 1.7 | 0.13 |
| 8/10/36 | Lee & Sons | 464-947 | 0.74 | 1.4 | 0.10 |
| 8/26/36 | " " | 422-979 | 0.75 | 1.6 | 0.10 |
| 12/14/36 | Queen & Whitwell | M719-1517 | 0.61 | 3.0 | 0.27 |

<u>Hand Samples Gold at $35.00</u>

| | | |
|---|---|---|
| 3/15/48 | McKinley tunnel E of No 2 shaft | $ 9.50 |
| | " " above tunnel, grab | 11.80 |
| | " " 20 ft. below tunnel | 15.20 |
| | " " " " " h.w. | 12.00 |
| | " " west in stope | 14.10 |
| | 250 Level No. 2 shaft grab east | 10.50 |
| | " " " " " " | 11.30 |
| | 300 " " " grab on floor | 18.10 |
| | " " " " screening thru 1" | 18.20 |
| | " " " " sulphide | 11.20 |
| | 650 " " " grab east of shaft | 12.00 |
| 3/19/48 | " " " " " west of shaft | 11.10 |
| | 750 " " " " east of station | 9.40 |
| | " " " " " east of shaft | 15.20 |
| | 250 " " " quartz | 11.80 |
| | 350 " " " " | 13.00 |
| | 750 " " " " | 21.20 |
| | 1000 " No.3 shaft " instope | 12.50 |
| 3/21/48 | 250 " No.2 " heavy sulphise | 68.80 |
| | 650 " " " banded sulphide east | 51.62 |
| | 650 " " " sulphides in hanging wall | 35.45 |
| | 250 " No.5 " " in top above cave | 31.15 |
| | " " " " " " bottom " " | 28.38 |
| | 1000 " No.3 " " east in stope | 63.56 |
| | 300 " No.2 " banded sulphides east | 54.71 |
| 4/6/48 | Surface above Congress workings new vein | 20.60 |
| | " dump No.3 shaft loose quartz on dump | 31.50 |
| | Tunnel level old No 5 shaft at fault | 26.75 |
| | " " " " " " " | 9.50 |
| | " " " " " | 8.25 |
| | " " " " " | 11.00 |
| 4/20/48 | 750 Level No.2 shaft grab in drift east | 26.50 |
| | 800 " " " " west | 13.20 |
| | 500 " " " quartz hanging wall | 15.50 |
| | 1000 " No.3 " grab | 11.20 |
| | " " " " hanging | 12.00 |
| 5/1/48 | Queen of the hills upper tunnel lower | 8.00 |
| | " " " " float | 12.35 |
| | " " " " middle | 9.20 |
| | " " " " upper | 8.95 |

| | | |
|---|---|---|
| 5/6/48 | 1000 Level No.3 shaft | $26.18 |
| | " " " " sulphide in gob | 19.72 |
| 5/21/48 | No description except grab of fills | 21.00 |
| | " " " " " " | 11.90 |
| | " " " " " " | 6.30 |
| | " " " " " " | 5.60 |
| | " " " " " " | 4.55 |
| | " " " " " " | 20.30 |
| 6/8/48 | " " " " " " | 7.70 |
| | " " " " " " | 11.90 |
| | " " " " " " | 4.90 |
| | | |
| General average of above samples | | 18.17 |
| Average of all fills | | 11.60 |
| Average of fills through $\frac{1}{4}$" mesh | | 15.38 |
| | ****************************** Au at $32.20. | |
| 9/28/49 | Queen, Lower tunnel | 16.74 |
| | " " " | 14.61 |
| | " " " | 10.75 |
| | " " " | 26.81 |
| | General average of above | 17.23 |
| | | |
| 10/6/49 | Queen | 3.83 |
| | " | 3.74 |
| | " | 7.48 |
| | " | 6.44 |
| | " | 6.48 |
| | " | 8.98 |
| | " | 3.99 |
| | " | 21.66 |
| | " | 10.42 |
| | Composit of above 8 samples | 8.05 |
| | | |
| 8/20/49 | Queen Pile top of hill | 19.60 |
| | " " side hill | 14.70 |
| | " " dug from vein | 7.70 |
| | " Quartz from vein | 9.45 |
| | " " top tunnel, left | 4.55 |
| | " Veins above road | .70 |
| | | |
| 11/10/49 | Queen 10 ft wide | 5.13 |
| | Queen gob | 5.79 |
| | " Lower level 10 ft. wide | 5.79 |
| | " " " 4½ " | 4.18 |
| | " " " tunnel | 6.11 |
| | " Middle tunnel 9 ft. wide | 5.95 |
| | " wide vein lower tunnel 8' | 2.80 |
| | " sulphide footwall vein 2' wide | 16.84 |
| | " at turn of winz above lower tunnel 4' | 15.35 |
| | " beyond cave footwall middle tun. 2' | 14.90 |
| | " hanging wall vein Middle Tun. 2' | 17.88 |
| 1/14/50 | 650 Level No 2 shaft disc sulph. east of cave | 16.80 |
| | " " " " same pl. sulph. 1½' wide | 252.73 |
| | " " " " grab fill cross vein | 15.40 |
| | " " " " rock as above | 8.40 |
| | Queen smelter check | 7.00 |
| | " " " | 25.11 |
| 1/23/50 | " Middle tunnel samples | 5.85 |
| | | 13.46 |
| | | 19.32 |

| | | |
|---|---|---|
| 5/11/50 | Queen Middle tunnel underhand at winz 30" | $ 9.12 |
| | " " " " " " " | 8.08 |
| | " " " " " " 3' | 8.20 |

Shipments from Cross Vein at surface

| | | |
|---|---|---|
| Lot No2 | Gold at $32.3185 | 23.89 |
| Lot 1 | " " | 35.76 |
| Lot 3 | | 16.85 |
| Feb 17, 1950 | Lot 270 | 16.54 |
| Feb 17, " | " 271 | 31.12 |
| March 25 1950 | 1 Queen | 8.95 |
| May 15, " | 2 " | 10.36 |
| " " " | 3 " | 99.82 |
| June 2, " | 4 " | 11.18 |
| July 22, " | 5 " | 8.93 |
| October 13," | Niagara | 3.88 |

**************************

| | | |
|---|---|---|
| 2/23/50 | Queen 3 ft. wide | 17.90 |
| | " 4 " " | 4.20 |
| | " 3 " " | 13.29 |
| | New strike on hill in shaft 50" deep, 8" | 100.64 |
| | " " " " surface trench | 10.15 |
| 2/7/50 | No record | 19.70 |
| 2/16/51 | Bellick Dyke tun. near end 2 ft. | 10.36 |
| 3/2/51 | Vein 2' wide surface toe of Wyman waste dump | 19.96 |
| | Queen Lower tun. H.W. Bellick dyke 2' | 10.79 |
| 3/30/51 | No 1 shaft 120' deep stringers in dyke | 102.05 |
| | " " " " dyke | 1.60 |
| | Queen upper tunnel | 3.16 |
| | Waste from No 1 dump | 1.06 |
| 5/22/51 | No 1 shaft, 157' level 65' west gob | 14.40 |
| | No 2 shaft, 650 level 200' E. 3' 75' up stope | 4.06 |
| | Queen upper tunnel 40" wide below level | 22.80 |
| | No 2 shaft 650 level 200' E. gob | 2.12 |
| | No 1 shaft, 157' level 70' W. gob | 10.43 |
| 5/31/51 | No record | 4.56 |
| 6/1/51 | " " | 4.65 |
| 7/17/51 | No 1 shaft 157' level 20-120' W. Dyke & quartz 200# sample | 14.04 |
| | Same after crushing -3/8" | 27.32 |
| | " " " plus 3/8" | 20.84 |
| 9/17/51 | Mill tailing sample from old dump | 2.90 |

************************

| | Per unit | Tungsten at $63.00 |
|---|---|---|
| Queen middle Tun quartz | 0.025 | 1.57 |
| Congress dyke on surface | 0.08 | 5.20 |
| Mill tailing | 0.045 | 2.92 |
| Dyke on tunnel dump | 0.38 | 24.70 |
| " " " " | 0.33 | 21.45 |
| Dyke No 2 dump | 0.43 | 27.95 |
| Mill tailing | 0.13 | 8.45 |
| Congress dyle surface | 0.20 | 12.60 |
| Dyke from No 3 dump | 0.29 | 18.27 |
| Yellow rock No 3 dump | 0.75 | 4.72 |
| Congress dyke surface | 0.145 | 9.13 |
| Congress dyle surface | 0.075 | 4.72 |
| " " " | 0.125 | 7.87 |

Black sand from road above mill
Au. 7.11 oz. Ag. 21.5 Value at $35 $246.84

E. A. Colburn, Jr.
P. O. Box 152
Wickenburg, Arizona

September 1, 1955

Mr. W. A. Murray
Washington,
D. C.

Dear Mr. Murray:

In order that you may have the basic facts about the Congress Mine, Congress, Yavapai County, Arizona, without too much detailed data ordinarily included in a mine report I am, below, more or less abbreviating the salient points regarding the property.

There are 14 patented claims and 9 unpatented claims in the property, all located in a block about two and onehalf (2½) miles by good level road from Congress Junction which is a station on the Santa Fe Railroad. It is also on paved Highway 89 and paved State Highway 70. The claims are on the flat desert, and on the Date Creek Mountains in the Martinez Mining Dbstrict. This is one of the most advantageous situations for a mining property in the west.

Past authenticated production has been close to $8,000,000.00, mostly in gold with some silver and the production was had prior to 1910 when the price of gold was at $20.00 per ounce. From old letters and papers it is apparent that the production in tons ran around 700,000, at least that was what was run through the mills. Recovery ran 94.33% from 1894 to 1910. Gross value of the ore was $13.01 with gold at $20.67 and silver at $0.60. Present value would be over $20.00 per ton.

Development consists of several shafts and many levels run at the interval of around 75 ft. as measured on the vein.

These levels consist of many miles, perhaps 20 or over.

CONGRESS VEIN WORKINGS

| | | | |
|---|---|---|---|
| Shaft No. 1 | 1,000 | ft. | deep |
| " " 2 | 1,700 | " | " |
| " " 3 | 4,000 | " | " |
| Niagara Shaft No. 4 | 1,900 | " | " |
| " " 5 | 2,050 | " | " |
| " " 6 | 1,800 | " | " |
| Queen of the Hills Winze | 600 | " | " |

The Cross vein worked from both No. I Shaft above the 650 ft. level and No. 2 shaft below that level with a winze extending from the 650 ft. level to the 1,000 ft. level. Queen of the Hills vein ( probably a displaced segment of the Cross Vein) had three tunnels; top, middle, and lower, the latter one going clear through the hill.

There is ample dump room for both mines waste and mill tailing, possibly about 100 acres of flat land some of which is now occupied by tailing, but not filled by any means. There are several mill sites on the property.

Some water is available from Martinez Creek about 6,600 ft. from the main workings and now connected by pump and pipe line. Electric line already in at the pump which could be extended to the mine, or power could be brought in from the south where the distance is somewhat greater but the terrain better. There is plenty of water in the old mine workings to keep a good sized mill in operation if the water is taken care of by filtering etc.

Proposed mill capacity to start with would be about 100 tons per 24 hours, to be located at or near No. I shaft., which would be the main production opening for the Congress vein. The flow sheet would depend upon future work in the development of the uranium ore body but probably would include gravity and flotation concentration for uranium and the cyanidation of tailings for gold and silver. Sorting and washing arraingements in the crushing plant would provide sources of waste for the mine dumps and shipping grade ore to be sent to outside mills or smelters for final recovery.

The cost of ~~such a~~ plant would be in the neighborhood of $150,000.00. This would include cost of power line (if any) and other extras incident to the mill. The power company has told me that they would bring in a high tension line without cost if we had a commercial load. The ideal power hookup, in my judgement, would be to buy power for the steady loads, such as milling etc. and make with Diesel engines the fluctuation loads as developed by hoists, compressors etc. which run only a portion of the 24 hours, but this would necessitate the purchase of engines and generators and the employment of extra labor. However, I believe that it would pay in the long run as it would keep the peak power demand quite low and thus avoid excessive power costs. Sunchronous motors on any large horse power drives would also be an aid here as well as keeping up the power factor.

On a basis of producing about 100 tons per day the No. 3 Shaft would have to be equipped with a double drum hoist and a four or five drill compressor. This would run about $30,000.00. For No. 5 Shaft and Queen of the Hills tunnel, seperate equipment would have to be furnished as the openings are at considerable distance apart. These two together would run about the same as the figure quoted above.

Underground supplies and equipment would include skips, cars, track and pipe and some timber, although there is several thousand dollars worth of good mine timber available in No. 3 Shaft. Also would need rock drill, hose, slusher hoists etc. Total of around $25,000.00 depending on the scale of operations.

The tonnage of the uranium ore is not especially well known just at the present time, but from indications, samplings and experience with conditions in the mine, I would judge that there should be available about 30,000 tons in and adjacent to No. 3 Shaft I would indicate an area 150 ft. long, 800 ft. high and 3 ft. wide extending from the 200 ft. level to the 1,000 ft. level and pitching at an approximate angle of 30 deg. west. This area has been developed by the shaft and the old gold stopes. Then there is a showing all the way down No. 2 Shaft from the tunnel level to the 650 ft. level which is in the hanging wall of the old gold stope which is now filled with ore. It is my judgement that the combined mineral content over the entire width of the dyke in the area where the footwall seam was mined for gold will make pay ore if properly mined and milled. However, such an operation would have to be on a good tonnage basis. This would run the tonnage figure to around 3,000,000 tons. There are certain areas in the dyke where the gold stringers, which are more nearly vertical than the dyke, are closer together and which form pay ore for the whole dyke and the uranium ore as shown in No. 3 Shaft seems to follow very closely the fissuring and shattering of the dyke, sometimes being nearly from dyke wall to wall. This condition may well obtain in No. 2 Shaft in the hanging wall of the old stopes. In the Queen of the Hills lower tunnel we have a gold ore body which is developed between that level and the middle tunnel above which should contain 30,000 tons of ore of mill grade. This gold quartz is associated with some pretty good spots of uranium ore which occurs both in the foot wall and the hanging in some gouge slips and fault fissures. Everywhere the uranium ore seems to be associated with post gold faulting and is largely localized in said fault gouge and in fault shattered rock regardless of whether it is dyke rock or vein material associated with granite. On straight gold ore contained in the old filled stopes, disregarding any uranium content, I would estimate at least 400,000 tons of an average grade of over $11.00 per ton as shown by my samplings. This would be net tonnage to be milled after screening and sorting.

The surface showing of uranium is quite small due to the solubility of that mineral and the almost endless combinations it makes in nature with other metals and bases. But it does not extend from the so called break just west of the No. 1 Shaft on the Congress vein westerly to and beyond the No. 2 Shaft. On the tunnel level or 200 ft. level, where the tunnel passes behind the above shaft, the background count is quite high. This is not due to radon gas for thereis a very heavy movement of air at this point and any concentrations of gas would be swept out to surface as the air is moving rapidly in that direction. Here the low grade mass extends from foot to hanging wall of the dyke, a distance of about 20 ft. No work has been done here or elsewhere to limit the ore body. The ore was followed down No. 2 Shaft to the 650 level by the AEC engineer and several samples taken. No. 2 Shaft being sunk in the dyke or on the footwall of it and in direct contact with the old gold stopes which have been filled with low grade ore shows a general avérage of about 0.15% U308 all the way down. The uranium is confined to hanging wall of the dyke and occupies the fault plane which extends downward from the granite for about a couple of feet. Where ever the dyke is crushed the uranium ore follows into these fine seams contained in the dyke and mineralizes them. On the tunnel level west of No. 2 Shaft the bottom of the drift shows up well on the Geiger counter and this would seem possibly to be the upward extension of the uranium ore in shaft No. 3 farther west, as explained above.

The Queen of the Hills vein, also as explained above, is entirely in the granite country rock and has no dyke with it. However, we find good grade uranium ore in direct contact with the quartz gold ore, not that the quartz is radio active, but the gouge on either side of the quartz is. There has been considerable movement horizontally along this vein to account for this gouge.

Farther north where this vein crossed the Bellick dyke, which is more or less parallel to the Congress dyke, and very much like it otherwise, the situation is again quite active in the line of uranium. The background count is so high at this point, which is the situation of the Queen of the Hills 600 ft. winze, that one is practically unable to get a reading on the counter without getting out of the tunnel and waiting for a couple of hours before the counter comes back to normal. Then only can a reading be had on material collected in the tunnel. No chemical assay tests for uranium have been made there.

It seems to me that the above showings in uranium is well worth the time, effort and money to be expended thereon. The Uranium mineral contained in the Congress ores has been classified as LEIBIGITE, a high grade mineral of unique occurence. This classification came from the Naval Gun Factory Laboratory in Washington and should be authentic.

Now as to the tonnage of gold ores exposed in the workings. Reports show that the mine has produced over 700,000 tons of ore and that tonnage has been run through the mill. There is at least an equal tonnage left in the old stopes which have been sampled several times at about $11.60 per ton in minus one inch size which would mean sorting some of the waste out and thus reducing the tonnage to about 400,000 tons. This is an enormous backlog for a small milling plant and constitutes only a portion of the ore available for milling. The cross vein near to the No. 1 Shaft should produce ore without sorting of around $14.00 per ton. Samples were taken in a tunnel close to the surface at $12.00 and on the same vein at the 650 ft. level at $16.00 per ton. Many samples have been taken in various place showing $14.00 to $18.00 per ton even in large mill runs and, of course, some have been taken of a lower grade offrom $4.90 up to $10.00 all in broken ore. There is quite a tonnage available in the old stopes of the Niagara vein and one quite large body of unbroken ore therein that shows assays of $28.00 and $31.00 at 350 ft. depth. This was discovered when caving from the hanging wall disclosed it to view about 10 ft. in width and at least 100 ft. long.

Perhaps the most interesting thing from a high grade standpoint is the ore body at the bottom of No. 5 Shaft on the Niagara vein. This vein lies to the south of the Congress and is in the granite formation on all upper levels and dips more steeply than the other vein or about 30 to 40 degrees. The ore was wider but not quite so good as that contained in the Congress vein and it contains some free gold and some heavy pyritic ore as well. As it reaches the lower level it comes in contact with a basic dyke quite similar to that accompaning the Congress vein and flattened off on it making a fine long, high grade body of ore on the contact. The dyke comes in from the footwall and carries the vein along with it making a virgin mine from that point downward of good high grade ore of an equal to that contained in the Congress vein in early days. It could even be shipped to the smelter at a fine profit, but would be much more profitable if milled locally along with the lower grade ores.

There are no maps covering the territory stoped above this area except one drawn many years later from reports of development work and shipping and milling data of the ore. This map shows within dashed lines an area with the following caption "Large stope position and boundary indefinate, records show 140,000 tons taken from No. 5 Shaft since 1903, probably from this general area." This shaft was closed in 1910.

The ore in this shoot should be from 500 ft. to 700 ft. in length and should extend downward indefinately, as the Congress mine has never been bottomed and is at a very much greater depth. Also some of the ore in the easterly section of the Niagara vein has been cut off by what is known as the "East Fault". This, to date, has not been solved, but from recent disclosures it would seem that it should be readily figured out, as I have been able to determine the horizontal thrust and have figured the vertical component at about 150 ft. This would throw the lost segment that much nearer the surface.

Another vein of great promise is the New Strike vein north of and up the hill from the Congress vein. It is exposed at several points on surface and has produced one shipment that I know about of $51.00 per ton. The vein is rather narrow, about 18", but is high grade and contains considerable specimen rock showing free gold.

There are several parallel veins in the Congress workings which show high grade ore that has not been stoped, notably on the 650 level, now inaccessable, where a vein in the hanging rock wall having a width of about a foot shows $75 rock over a stope length of around 100 ft. It wouldn't take too much work to get at this ore whenever the mine is opened. I personnally saw this ore more than 20 years ago, in fact I have seen most of the ore I have written about except that in the bottom of the No. 5 Shaft and that data is from a letter from the former manager who believed this deposit to be the best in the mine.

Cost of mining should be held down to $4.00 per ton and if only recovering ore from the fills quite a little less than that figure. On a basis of breaking the whole width of the dyke I believe that it can be done for $3.00 per ton. Milling will run about $4.50 per ton not taking it into account the marketing expense of the uranium concentrate, which is not known to me at this writing.

Cost of mining and cleanup together with the initial machinery expense fund would run about $100,000.00. Cost of milling building as given above $300,000.00. And there should be working capital of about $50,000.00, at least, to tide over until the mill would come into production. It is apparent that these figures could be modified downward if it were necessary to limit working places and equipment for lack of capital, but with the scheme as outlined above the results would come quicker and more surely.

Very truly,

E. A. Colburn, Jr.

## Mining and Milling Experience of

## E. A. Colburn, Jr.

While yet in HighSchool in Colorado Springs, Colorado, the city of my birth, I visited the Cripple Creek Mining District and went underground in one of the properties owned by my father. He had been one of the first to appreciate the great opportunities available in that camp and had secured an interest in the original discovery there, the Gold King in Poverty Gulch, quite near the town of Cripple Creek. During the time this property was being developed he secured interests in or the entire ownership of several other mining claims and groups within the District, among which was the Ajax, Theresa, Dolly Varden, Strong and others. During that first visit I made up my mind that I wanted to follow mining for a life work, and proceeded so to do in vacations and between other schooling.

Starting in at the bottom with tramming, mucking, machine mining, I steadily progressed through all of the skills in and about the mines, finally reaching a foremanship on the Ajax over a detached portion of the operation. Next I was assigned to the superintendancy of the Theresa Mine in Goldfield, Colorado, in which position I had control of more than 60 men. When this mine was sold we were engaged in building new ore bins and washing plant, sorting belts, new and larger gallows frame, hoisting engine and compressor along with the necessary boiler capacity for the enlarged plant.

Later I went back to college at Colorado Springs, Colo., for a time and again back to the Ajax for additional work in the assay department and underground as assistant superintendent. At that time were about 300 employees on the mine. Next I took over the operations on the Golden Wedge Mine near Anaconda, Colo., where I sank a 600 ft. vertical shaft, installed a surface plant, gallows frame and other equipment and mined out the vein to that depth where the ore shoot dipped off our property. Then followed some exploration work on the Dolly Varden in Squaw Gulch about a mile from the last named mine.

The superintendancy of the Ajax was next. By this time the mine was worked only by leasers, the surface plant being operated by the company. Profits were low or non existant and most of the leasers dissatisfied with conditions as handled by the former superintendent. In order to better the mine service and decrease costs I made changes that allowed a very nice profit yearly. Such changes were paid for within a years time by extra profits with quite a little left over. These improvements included the raising of the gallows frame, installation of skips in the place of cages, putting in an electric ore car for transportation of ore to the bins and waste to the dumps, etc.

E. A. Colburn, Jr.
Box 152
Wickenburg, Ariz.



Some time later a 300 ton cyanide plant was designed and built under my direct supervision and placed in operation. This plant was unique in the Cripple Creek District in that we treated the sulpho-telluride ores without roasting or other oxidizers and were able to get high recoveries on ore of all grades. Other mills were able to treat the lower grade ores without roasting, but at that time none of them were able to handle the higher grade material. We made many runs of segregated ore of a value of about $30. per ton and finally reduced the tailing to around a dollar per ton.

The process consisted of cyanidation with the aid of electrolysis which enabled us to get the gold held in the tellurides into solution quickly and completely. Also a further extension of the process was worked upon which included the precipitation of gold from the mill pulp without seperation of the solution from the ore pulp. We were very successful in this and in getting a barren solution for returning to the head of the plant for further use. However we did not put this into actual mill practice.

The Ajax mine produced over $5,000,000,,and while I was in charge the annual production was around a quarter million for each of the seven years.

My brother and I produced and patented a flotation machine at about this time, that is early in the development of flotation, and later on had flotation laboratories in both Denver and San Francisco, where we made tests on many varieties of ores. I also patented a new type machine in 1927 which was sold later on to the Denver Equipment Company, and parts of which are now in production.

I came to Arizona in 1934 as Superintendent of the Illinois Mining Company, which had a lease on the Congress Mine at that time. A raise was run on the 300 ft. level from the Congress vein which did not extend far enough to cut the vein we were looking for, and this work was about the only serious attempt to work underground since the mine was closed in 1910. A few leasers worked at and near the surface only.

Did considerable work in testing on the mill while it was in operation on the Congress dumps and determined the best flow sheet for the ores.

Later on was manager for the Du Boise interests and ran the Mammoth mine and the Belle Mine in the Eureka Mining District, Yavapai County, Arizona.

Secured the Congress Mine in 1947 and have been engaged with that property to date.

# REPORT ON CONGRESS MINE
# AND
# TAILINGS DUMP
# YAVAPAI COUNTY, ARIZONA

The Congress Mine and tailings dump were examined by W. A. Leddell and Gerald Sherman for the Congress Mine Corporation, of 100 Broadway, New York, beginning on June 14, and continuing through the months of July and August, 1935.

From measurements, sampling, and milling tests on the tailings, the following results are calculated:

## CONGRESS TAILINGS

QUANTITY

| | | |
|---|---|---|
| White Tailings | 120,000 | Tons |
| Brown Tailings | 280,000 | Tons |
| | 400,000 | Tons |

WHITE TAILINGS

| | | | | | |
|---|---|---|---|---|---|
| Gold | 0.0735 ozs. per ton | $2.53 @ 85% | $2.15 | | |
| Silver | 0.045 ozs. per ton | .34 @ 75% | 0.25 | | |
| | | | $2.40 | | |
| Operating Cost per Ton | | | .836 | | |
| | | | $1.564 | | |
| OPERATING PROFIT | 120,000 Tons @ $1.564 | | | | $187,500 |

BROWN TAILINGS

| | | | | | |
|---|---|---|---|---|---|
| Gold | 0.525 ozs. per ton | $1.81 @ 70% | $1.26 | | |
| Silver | 0.33 ozs. per ton | .25 @ 65% | 0.16 | | |
| | | | $1.42 | | |
| Operating Cost per Ton | | | .836 | | |
| | | | $0.584 | | |
| OPERATING PROFIT | 280,000 Tons @ $0.584 | | | | 163,500 |
| | | | | | $351,000 |

| | | |
|---|---|---|
| MILL AND PLANT CONSTRUCTION | $92,320 | |
| WATER SUPPLY | 15,000 | |
| ENGINEERING | 5,300 | |
| CONTINGENCIES | 7,380 | |
| WORKING CAPITAL | 20,000 | 140,000 |
| | | $211,000 |

The Congress Mine was owned by the Murphy Estate and managed by T. J. Byrne, Attorney, of Prescott, Arizona, as Trustee.

The property is situated in the Martinez Mining district, Yavapai County, Arizona, about 3½ miles north of Congress Junction on the A.T. & S.F. Railway. It is reached by a good road on a slightly ascending grade from Congress Junction. The property consists of 15 patented and 9 unpatented claims.

Patented Claims

| | |
|---|---|
| Fraction | Niagara Mill-Site |
| Why Not | Ohio |
| Mosouri | Rich Quartz |
| Niagara | Golden Eagle |
| Congress | Incline |
| Queen of the Hills | Old State |
| Excelcior | Snow Storm |
| Golden Thread | |

Unpatented Claims

| | |
|---|---|
| Bellick | Highland |
| Remnant | Keystone |
| Boundary | East Extension of Golden Thread |
| Sunnyside | Martinez |
| Ophir | |

The Congress Mine was located in 1887, or shortly before. Active production covered the years 1889 to 1891 and 1894 to 1910, since when it has been practically or entirely idle. During these periods it produced in gold and silver sold, $7,649,497, which was taken from 692,332 tons of ore. The recovered value per ton was approximately $11.50, having a gross value of $13.00, which would now be worth in gold and silver contained, about $22.00 per ton

PRODUCTION FROM CONGRESS MINE by W. F. STAUNTON

| | |
|---|---|
| Production | 692,332 Tons |
| Gold | 388,477 Ozs. |
| Silver | 345,598 Ozs. |

| | | | |
|---|---|---|---|
| Value | Gold | $20.67 | $8,029,900 |
| | Silver | 0.60 | 204,560 |
| | | | $8,234,460 |
| Value per ton | Gold | $11.60 | |
| | Silver | .19 | |
| | | $11.79 | |
| | Tailings | 1.20 | |
| | Gross Value per Ton | $12.99 | |
| Value | Gold | $35.00 | $13,597,000 |
| | Silver | 0.75 | 259,000 |
| | | | $13,856,000 |
| Value per ton Gold | | $19.63 | |
| | | .37 | |
| | | $20.00 | |
| Tailings (Sampled) | | 1.95 | |
| | | $21.95 | |

These figures were taken from an article by W. F. Staunton, published in the Engineering and Mining Journal of November 13, 1926. Mr. Staunton is a mining engineer, who was Superintendent and Manager of the operations during 1894 to 1910.

The tangible assets of the Congress property consist of the tailings milled during the productive period and now situated in a pile about 1000 ft. long, 300 ft. wide, and of varying thickness up to 37 ft. on the Excelcior and Golden Eagle, with a small extension at a higher level on the Incline Claim.

Attention had been drawn to this tailings dump by a report of The Merrill Company, dated December 31, 1915, in which its quantity and value in gold and silver is estimated with the plant required for its treatment and operating costs, and a statement of probable profits, to be obtained at $20.67 Gold.

This report was investigated, and it was found that the sampling was done by Frank H. Ricker, still of The Merrill Company organization, and Jack Moulton. The final summary and estimate was made by M. H. Kuryla, now Vice-President and General Manager of the United States Smelting and Refining Company of Pachuca, Mexico. The quantity of tailings was estimated at 505,000 tons and the gold at $1.46 per ton ($20.67) and silver at 21¢ per ton. A recovery of about 67% was expected, by regrinding and cyaniding. Under those conditions and at the prevailing price of gold, the project was not attractive, and The Merrill Company allowed it to drop. When the price of gold reached $35.00 per oz., the estimated gross value of the tailings rose to $2.78 per ton, with a much greater increase in the margin of profit.

The Merrill sampling was carefully done with 75 holes drilled and an equal number of cyanide tests, and the data on which the estimates were made were no doubt correct. The recovery by cyaniding was low, but no particular obstacle appeared which might prevent the successful treatment of the tailings. With a greater value, finer grinding and longer agitation would be permissible if it yielded a higher gold recovery.

Several attempts have since been made to treat the tailings, of which only one was successful. Otto Ellerman, now Manager of Perez & Company, Assayers and Metallurgical Engineers of Los Angeles, Calif., then associated with a Mr. Curran and acting for an investor, designed, built, and operated a cyanide plant which treated 10,000 tons of tailings and produced a little less than $12,000 in gold and silver (at $20.67 gold) on which a profit was made. For lack of capital required to pay for the plant already constructed, and provide a better screen and pump, the operation had to be abandoned.

Another attempt was made to treat the tailings under the direction of J. T. Shimmin, who remodelled and enlarged the Sunshine mill at Kellogg, Idaho, in the winter of 1934 and 1935, and is now associated with the Buchans Mining Company, Buchans, New Foundland. The tailings were sluiced down by water to the plant, but it was discovered that appreciable quantities of the gold were water soluble which were lost in transit. These losses made the operation of the plant unprofitable, and again lack of capital prevented further action.

In 1933, the Congress Gold Inc. obtained a lease on the property to treat the tailings, and later to sort out and treat ore from the mine waste dump. Both operations failed, because of a shortage of equipment which prevented sufficiently fine grinding and time for agitation in solution, when treating the desired tonnage, and a lack of capital to correct it. This operation was followed by a lease and option to the Illinois Mining Corporation, which operated on similar lines, and relinquished their option in the latter part of 1934. In February 1935, an option was given to N. C. Clark, an attorney of Phoenix, Arizona, under which an examination was made by Wasserman & Company, of 40 Wall Street. Their results were not satisfactory to them and they returned the option. It was then given to Gerald Sherman for the Congress Mining Corporation.

It is estimated from survey and drilling records obtained in the examination of June, July, and August, that there are altogether more than 400,000 tons of the tailings. For details of the estimates, reference is made to a plan of drill hole locations and cross sections from surveys at 50 ft. intervals from end to end of the dump, and records of the drill holes, sampling, and calculations based on them.

The tailings first made, were roasted before cyaniding, but in later years, it was found that roasting was not economically necessary and the remainder of the ore was cyanided, raw. The dump, therefore, consists of roasted tailings of a light-brownish color, below, and unroasted tailings which were deposited on top of them, of clean white sand.

Drill sampling showed that there is a difference in the value of the two classes and of their metallurgical characteristics. The white sands carry 0.0735 ozs. of gold per ton, which yielded 85% by cyaniding and the roasted tailings * was noted in all drill holes and separate samples were taken from each class of material.

The roasted tailings do not contain as much gold as indicated by the Merrill Company sampling. The difference was probably caused by the migration of water soluble gold downward into the subsoil during the 20 years since the Merrill sampling was done.

In proportion to the investment, the profits were not particularly attractive and the option was released to Clark. A reduction in the purchase price and the possibility of obtaining a loan from the R.F.C. changed the situation and in January the property was purchased for $17,500.

The above estimate of tonnage and gold and silver content, on page 1, is based on the drilling of 49 holes, 3 vertical channels, and 2 shallow pits. The total feet drilled was 1320. In a number of cases the holes were repeats of those already drilled to check possible errors of sampling. The working data for the estimate is attached, as listed below:

Plan of the dump, showing its outline, and the location of drill holes.

Sections across the dump at intervals of 50 feet used in the calculation of tonnage and value.

List of drill holes and sample re urns.

Reconcilement of holes redrilled for checking, or to obtain more material for milling tests.

Average assay value for white and roasted tailings by sections.

Comparison of average assays from drill holes and sectional averages, etc.

Calculation of tonnage and value.

Calculation of weight per cubic foot.

The tailings were moist within a foot or two of the surface and with one or two exceptions, stood without caving when drilled to the original surface.

After putting in a short collar pipe to reach the moist sands, the material in the hole was taken out by "drop" bits, consisting of short length of 2 or 2½ in. pipe with the inside bevelled to a sharp outside edge at the bottom. They were about 2 ft. long and suspended by a bail from a half-inch rope. By raising the bit 2 or 3 ft. and dropping it, by its own weight to the bottom, some of the tailings were wedged up into the tapered bottom of the bit for an inch or so. A sample of moist tailings was thus picked up, lifted out and shaken into a sample sack by striking the pipe. In deep holes, a tripod and light pulley were used.

A few holes were blocked by striking timbers on the way down.

The white sands and roasted tailings were sampled separately, making two samples for each hole. In some cases the white and brown sections were split into an upper and a lower portion, to discover if there is any enrichment near the bottom. In one or two instances this was found to be the case as in the two pits, #32, and #43, which were dug into the subsoil. On the whole, there was no consistent difference between the upper and lower portions either of the unroasted or roasted tailings, but there are still possibilities of encountering pockets of richer material from which the soluble gold could not drain away.

There are probably 15,000 to 20,000 tons of tailings retreated by former operators and discharged on the flat below the old dump, which average about 0.074 ozs. per ton. They contain not only retreated tailings but tailings from quantities of ore sorted out from the mine dump by the Congress Gold operations and raised to an expected value of $4.00 to $6.00 per ton. This probably accounts for the high samples and may indicate the occurrence of a few thousand tons of tailings on which good profits can be made, but the sampling and measurements are not sufficiently complete to include these piles in the estimate.

Composite samples were made up of each class of ore independently to be sent for testing to John G. Graham, Professor of Mining and Metallurgy at the Texas College of Mines, at El Paso, Texas. Other duplicate lots were sent to the American Cyanamid Company of New York, and a few to the R. A. Perez Company, of 120 North Main Street, Los Angeles, where they were tested under the direction of Otto Ellerman, Manager.

The milling tests were made under the supervision of W. A. Leddell, Metallurgical and Mechanical Engineer, Mills Building of El Paso, Texas.

The report of Mr. Leddell on the tests with a flow sheet of a mill and estimates for its construction, are attached with accompanying reports by Graham and the American Cyanamid Company.

Mr. Leddell's results are incorporated in the estimate of operating expense and profits, which appear on page 1.

The original water supply was from a well on Martinez Creek about one mile east of the mine which is now equipped with a pump, and may be used for a mill and domestic supply up to its normal flow. While the Congress was in active production, there was not enough water in Martinez Creek to supply the stamp mill and a community of some hundreds of people. In order to make good the deficiency, wells were sunk on Date Creek about six miles to the north and water pumped over the divide between Date and Martinez Creeks, from which it flowed by gravity to the tanks at the Martinez Creek well and was pumped from there to storage tanks at the mine.

With 15 to 20 men and an economical use of water in milling, it may be possible to operate with the Martinez Creek water. In case this is insufficient, easements for wells and rights of way have been obtained from holders of agricultural and homestead claims on Date Creek, and application has been made for a water right from that creek for a quantity up to 30,000 gals. per day. A pipe line is planned, which would be about seven miles in length following a course indicated on the attached U. S. topographical map.

The Congress Mine:

Most of the information on the Congress Mine was obtained from an article in the Engineering and Mining Journal and letters, by W. F. Staunton, of which copies are attached.

Ore was mined from two veins, the Congress and the Niagara. They both outcrop in the foothills of the Date Creek Mountains, and dip under a minor projecting ridge. Both lie in an area of Granite, probably a part of a similar formation on the west slope of the Bradshaw Mountains.

It is contained in and runs through a greenstone trap dyke about 15 ft. in width, which strikes northwest and southeast and dips at about 25 degrees into the hill to the north. The ore is more often found on the foot wall but may appear in any part of the dyke. The ore shoot follows in a general way, the intersection of the dyke with a fissure vein in the granite.

The Niagara vein cuts through the granite, strikes more nearly east and west and dips about forty degrees to the north, thus departing from the Congress vein in depth and toward the west. This vein does not accompany a dyke, but it is believed that an intersection with a flat greenstone dyke, striking more to the northwest which cuts it on a line, dipping in a northwesterly direction, has had some influence on the ore shoot found adjacent to it.

The coincidence of both ore shoots being apparently associated with the intersections of veins with dykes, and the occurrence of more or less ore in both formations near the intersections may lead to the discovery of other ore shoots.

Both veins were productive, the Niagara ore probably wider, but the Congress somewhat richer.

The two veins were developed and operated through several inclined shafts, which followed tham down. The Congress shafts are No. 1, 1100 ft.; No. 2, 1700 ft.; and No. 3, 4000 ft. deep. The Niagara shafts are No. 5, 2050 ft.; No. 4, 1000 ft.; and No. 6, 1800 ft. deep. There are some shallow older shafts on the Niagara vein east of No. 5 Shaft toward the eastern fault which cuts off the vein in that direction, and there are also shallow workings on the Queen of the Hills and Bellick claims. Records indicate that the Queen of the Hills produced about 20,000 tons of ore.

All shafts are caved and the underground workings connecting with them are practically inaccessible, except the Congress No. 3 2 Shaft, open to the water level at about the 1300 Level, and the Niagara No. 5 for 300 ft. By climbing over waste piles caused by broken lagging, some of the level drifts can be followed for severl hundred feet from the

Congress No. 2 Shaft. The Niagara No. 5 Shaft, blocked below 300 ft. is connected with some of the shallower workings toward the eastern fault.

The ore is found in two main shoots, one in the Congress and one in the Niagara vein. The Congress ore shoot appears to have been uniformly good and to have been mined out completely. There is, therefore, practically no ore in place that can be inspected without cleaning out the drifts, for access to the stope faces.

The Congress vein being flat, the ore was broken separately so far as possible and only enough rock to make room for stoping, and to fill the workings and thus support the roof. In this process, fragments of the vein quartz which is friable, were thrown back into the gob and lost. The hanging wall in some cases, carried mineralized stringers that could be mined. During the period of production, material containing less than $7.00 0.35 oz. did not pay to mine. It is assumed, therefore, that some high grade ore lost in the process of stoping and the reject of some lean ore which was brought by sorting up to a workable average, and some of the weakly mineralized hanging wall stringers, were left in the fill.

This idea is supported by the letter from W. F. Staunton on the subject, dated October 27, 1933.

In order to check this theory 21 samples were taken from the stope fill on various levels from the No. 2 Congress Shaft, as indicated on the underground map of the Congress vein. They were obtained by cutting the lagging, drawing out the fill and rejecting material that would be sorted out by hand, and using the remainder to represent the fine material that might be profitable, extracted for milling.

The average of all samples of fines was $5.65 per ton. 4 samples averaged $11.57 per ton. 4 additional samples were taken from points near them which averaged $4.68 per ton.

In the 850 West, 1250 West, 1300 West drifts, the samples yielded $10.35, $10.38 per ton and $7.59, respectively, per ton, at current metal prices.

It is concluded that there are areas accessible from the Congress No. 2 Shaft with slight expense, that can be worked, by roughly sorting out the coarse waste and shovelling or scraping fine material to the level below, the coarse material being packed behind and retained by occasional timbers to support the roof, and the fines sent to the mill. The operation can be best handled by contract, paying for the tonnage and value of the fine material hoisted.

In order to prove out such areas, more extensive sampling would be required, followed by taking out and hoisting all the fill from certain promising sections of the stopes, having a width on the strike of 15 to 20 ft., from one level to the next. On the surface, it would be screened and weighed to obtain the value and quantity that could probably be recovered, per unit of area.

Because of the increase in the value of gold, the edges of the stopes where mining stopped may carry sections that would now pay to take out. The stope boundaries are indicated in the map of the Congress vein, by red and blue lines.

A section of the vein is accessible between the second and third levels west of the No. 2 Shaft. Sampling there over 170 ft. indicates the occurrence of a shorter section that would pay to mine now. This may be assumed to yield as follows:

| | |
|---|---|
| 40 ft. 23" | $10.14 per ton |
| 50 ft. 22" | 9.40 per ton |
| 80 ft. 21 " | 8.30 per ton |

Some ore there is accessible and can be mined at any time if there were a mill available for its treatment, although there is probably little of it at this spot.

It was reported that the ore shoot above the 1700 Level and mined through No. 2 Shaft, produced the best ore mined. The east and west stope boundaries of this block, have an aggregate length on the vein of at least 3000 ft. It is very likely that ore extensions will be found east or west from those lines some one or two of which may extend for considerable distances and produce important quantities of ore.

While cleaning out the old stopes and exposing the faces pointing out into new ground, enough information may be obtained on which to base more extended explorations.

It would cost little to repair the Congress shaft and lay track to the water level. As the mine made so little water that it was taken out by bailing, the water level could be lowered to the 1700 Level at small expense.

The Niagara vein was rather lower in grade than the Congress. It is Staunton's opinion that it is likely to produce marginal ore from the untouched vein, made profitable by the increase in gold price, than the Congress, but its development would cost considerably more and work on it would be postponed until more information can be collected on its possibilities.

No value is set on the mine, but conditions are believed to be favorable for development.

Within the past few weeks, applications have been made to us for leases on the Queen of the Hills and earlier, to open the Niagara No. 6 Shaft. This might provide a foothold for further work.

No other development work in the mine is desirable until the dump operation has been well established. In the case that ore from the fills or from fresh faces can be produced, it would be necessary to provide a fine crushing plant to feed the grinding mill before the ore could be treated by cyaniding.

Plans:

During construction, pumping tests would be made on the Martinez well and the construction of the pipe line and wells at Date Creek would not be started until it has been proved to be necessary.

The details of mill design will be completed and construction begun as quickly as possible.

Tests of the Martinez well will be made and it will probably be necessary to lay a pipe line and put in a small semi-automatic gasolene-driven pump at Date Creek.

Since the white sands lie above the roasted tailings, it is proposed to treat them first in order to liquidate the cost of the plant as r pidly as possible.

When production is well established, attention will be directed to the mine.

Gerald Sherman.

March 17, 1936

## CONGRESS MINE

## OPERATING SCHEDULE

| | | |
|---|---|---|
| PLANT DESIGN AND ERECTION | | 6 Months |
| TAILINGS TREATMENT: | | |
| | 75,000 Tons White Tailings<br>25,000 Tons Brown Tailings | 12 Months |
| TREATMENT: | | |
| | 25,000 Tons White Tailings<br>25,000 Tons Brown Tailings | 6 Months |
| TREATMENT: | | |
| | 20,000 Tons White Tailings<br>230,000 Tons Brown Tailings | 30 Months |
| | 4½ Years | 54 Months |

# SUMMARY OF CONGRESS DRILL HOLES IN TAILINGS DUMP

## JUNE - AUGUST 27, 1935

| Hole No. | White Feet | White Assay | Red Feet | Red Assay | Total Feet | Total Assay |
|---|---|---|---|---|---|---|
| 1 | 20.0 | 0.07 | 15.0 | 0.046 | 35.0 | 0.059 |
| 2 | 26.75 | 0.07 | 11.25 | 0.046 | 38.0 | 0.063 |
| 3 | 8.0 | 0.07 | 28.0 | 0.046 | 36.0 | 0.051 |
| 4 | 15.0 | 0.07 | 10.0 | 0.046 | 25.0 | 0.06 |
| 5 | 16.0 | Lost | . | . | . | . |
| 6 | . | . | 10.5 | 0.046 | 10.5 | 0.046 |
| 7 | . | . | 12.0 | 0.05 | 12.0 | 0.05 |
| 8 | . | . | 5.0 | 0.05 | 5.0 | 0.05 |
| 9 | . | . | 5.0 | 0.05 | 5.0 | 0.05 |
| 10 | . | . | 13.0 | 0.05 | 13.0 | 0.05 |
| 11 | . | . | 9.0 | 0.05 | 9.0 | 0.05 |
| 12 | . | . | 13.0 | 0.05 | 13.0 | 0.05 |
| 13 | . | . | 8.5 | 0.06 ) | | |
| | | | 12.0 | 0.02 ) | | |
| | | | 8.5 | 0.04 ) | 29.0 | 0.038 |
| 14 | 12.0 | 0.07 | Caved | | 12.0 | 0.07 |
| 15 | 10.0 | 0.08 ) | | | | |
| | 8.0 | 0.11 ) | 6.0 | 0.06 | 24.0 | 0.085 |
| 16 | . | . | 20.0 | 0.06 | 20.0 | 0.06 |
| 17 | 5.5 | 0.10 | 19.5 | 0.07 ) | | |
| | | | 7.0 | 0.07 ) | 32.0 | 0.076 |
| 18 | . | . | 9.0 | 0.06 | 9.0 | 0.06 |

# SUMMARY OF CONGRESS DRILL HOLES

( continued )

| Hole No. | White Feet | White Assay | | Red Feet | Red Assay | Total Feet | Total Assay |
|---|---|---|---|---|---|---|---|
| 19 | . | . | | 33.7 | 0.06 | 33.7 | 0.06 |
| 20 | 31.5 | 0.05 | | . | . | 31.5 | 0.05 |
| 21 | . | . | | 24.5 | 0.04 | 24.5 | 0.04 |
| 22 | 9.4<br>14.6 | 0.07 )<br>0.07 ) | (a) | 9.5 | 0.05 | 33.5 | 0.064 |
| 23 | 16.0 | 0.10 | | 16.75 | 0.08 | 32.75 | 0.089 |
| 24 | 3.5 | 0.07 | | 19.0 | 0.04 | 22.5 | 0.044 |
| 25 | . | . | | 17.0<br>12.5 | 0.05 )<br>0.04 ) | 29.5 | 0.045 |
| 26 | . | . | | 9.6 | 0.04 | 9.6 | 0.04 |
| 27 | . | . | | 8.5 | 0.04 | 8.5 | 0.04 |
| 29 | . | . | | 8.0<br>12.1<br>10.4<br>1.4 | 0.06 )<br>0.06 )<br>0.04 )<br>0.065) | 31.9 | 0.054 |
| 30 | 16.9 | 0.065 | | 15.6 | 0.045 | 32.5 | 0.055 |
| 31 | . | . | | 29.4 | 0.04 | 29.4 | 0.04 |
| 35 | 20.7<br>5.2 | 0.07 )<br>0.12 ) | (b) | 8.2<br>3.5 | )<br>0.045) | 37.6 | . |
| 36 | 28.5 | 0.07 | | 6.4 | 0.065 | 34.9 | 0.069 |
| 37 | 34.6 | 0.065 | | 0.7 | 0.09 | 35.3 | 0.065 |
| 38 | 33.0 | 0.075 | | . | . | 33.0 | 0.075 |
| 39 | Lost | . | | . | . | . | . |
| 39a | 3.6 | 0.09 | | 21.6<br>1.1 | 0.055)<br>0.06 ) | 26.3 | 0.06 |
| 46 | . | . | | 14.5 | 0.075 | 14.5 | 0.075 |
| 51 | 31.7 | 0.08 | | . | . | 31.7 | 0.08 |
| 52 | . | . | | 25.0 | 0.04 | 25.0 | 0.04 |
| 61 | . | . | | 29.2 | 0.06 | 29.2 | 0.06 |

Note (a) - Assay 0.23
(b) - Assayed 0.17 - use 0.12 for estimate

## SUMMARY OF CONGRESS DRILL HOLES

(continued)

| Hole No. | White | | Red | | Total | |
|---|---|---|---|---|---|---|
| | Feet | Assay | Feet | Assay | Feet | Assay |
| 1a | 0 | | 28.2 | 0.05 | 28.2 | 0.05 |
| 3a | 8.2 | 0.08 | 24.3 | 0.055 | 32.5 | 0.061 |
| 13a | 3.5 | 0.065 | 23.5 | 0.05 | 27.0 | 0.052 |
| 16a | 0 | | 33.8 | 0.05 | 33.8 | 0.05 |
| 17a | 8.2 | 0.06 | 20.2 | 0.05 | 28.4 | 0.052 |
| 62 | 16.0 | 0.09 | 21.2 | 0.04 | 37.2 | 0.062 |
| 63 | 12.0 | 0.055 | 24.0 | 0.07 | 36.0 | 0.065 |
| 64 | 19.0 | 0.09 | 14.5 | 0.08 | 33.5 | 0.086 |
| 65 | 0 | | 27.1 | 0.055 | 27.1 | 0.055 |
| 66 | 0 | | 28.7 | 0.07 | 28.7 | 0.07 |

Not used in estimates

## PITS

| Pit No. | | | Depth | Perez | Goeglein |
|---|---|---|---|---|---|
| 32 (a) | Bottom sand | 9.3' to 10.3' | 1.0 | 0.095 | 0.12 |
| | Subsoil | 10.3' to 10.9' | 0.6 | 0.08 | 0.12 |
| | Subsoil | 10.9' to 11.9' | 1.0 | 0.04 | 0.08 |
| 43 | | | 9.8 | 0.07 | 0.08 |
| | | | 1.0 | 0.095 | 0.10 |
| | | | 1.0 | 0.10 | 0.11 |

Note (a) - Sampled for bottom sands and subsoil only

## VERTICAL CHANNELS

| Channel | White | | Red | | Total | |
|---|---|---|---|---|---|---|
| No. | Feet | Assay | Feet | Assay | Feet | Assay |
| 1 | 9.0 | 0.16 | 13.0 | 0.06 | 22.0 | 0.10 |
| 2 | . | . | 14.0 | 0.03 | 14.0 | 0.03 |
| 3 | . | . | 15.0 | 0.05 | 15.0 | 0.05 |
| 4 | . | . | 20.0 | 0.06 | 20.0 | 0.06 |
| | | Average | | 0.051 | 71.0 | 0.064 |

CONGRESS ~~VEIN~~ Tailings

HOLES DRILLED

| | |
|---|---|
| 49 Holes | 1227.0 ft. |
| 3 Vertical Channels | 71.0 |
| 2 Pits | 21.7 |
| 6 Holes in re-treated tailings | 32.0 |
| | 1351.7 ft. |

Nine shallow holes were drilled with an auger to locate the bottom of the tailings without sampling.

| | |
|---|---|
| #40 | |
| #44 | #48 |
| #45 | #50 |
| #46 | #58 |
| #47 | #59 |

## RECONCILEMENT OF REDRILLED HOLES

| Hole | White | | Red | | Total | |
|---|---|---|---|---|---|---|
| No. | Feet | Assay | Feet | Assay | Feet | Assay |
| 1 | 20.0 | 0.07 | 15.0 | 0.046 | 35.0 | 0.059 |
| 1a | . | . | 28.2 | 0.05 | 28.2 | 0.05 |
| Used in Estimate | | | | | | |
| | 10.0 | 0.07 | 25.0 | 0.048 | 35.0 | 0.055 |
| 3 | 8.0 | 0.07 | 28.0 | 0.046 | 36.0 | 0.051 |
| 3a | 8.2 | 0.08 | 24.3 | 0.055 | 32.5 | 0.061 |
| Used in Estimate | | | | | | |
| | 8.0 | 0.075 | 28.0 | 0.050 | 36.0 | 0.56 |
| 13 | . | . | 29.0 | 0.038 | 29.0 | 0.038 |
| 29 | . | . | 31.9 | 0.054 | 31.9 | 0.054 |
| 13a. | 3.5 | 0.065 | 23.5 | 0.05 | 27.0 | 0.052 |
| Used in Estimate | | | | | | |
| | 3.5 | 0.065 | 28.4 | 0.047 | 31.9 | 0.047 |
| 16 | . | . | 20.0 | 0.06 | 20.0 | 0.06 |
| 16a | . | . | 33.8 | 0.05 | 33.8 | 0.05 |
| Used in Estimate | | | | | | |
| | . | . | 33.8 | 0.054 | 33.8 | 0.054 |
| 17 | 5.5 | 0.10 | 26.5 | 0.07 | 32.0 | 0.076 |
| 17a | 8.2 | 0.06 | 20.2 | 0.05 | 28.4 | 0.052 |
| Used in Estimate | | | | | | |
| | 6.8 | 0.076 | 25.2 | 0.061 | 32.0 | 0.064 |

## RECONCILEMENT OF REDRILLED HOLES

( continued )

| Hole No. | White Feet | White Assay | Red Feet | Red Assay | Total Feet | Total Assay |
|---|---|---|---|---|---|---|
| 21 | . | . | 24.5 | 0.02* | 24.5 | 0.02 |
| 52 | . | . | 25.0 | 0.04 | 25.0 | 0.04 |
| Used in Estimate | | | | | | |
| | . | . | 25.0 | 0.04 | 25.0 | 0.04 |

* Assay of duplicate sample ran 0.04

| Hole No. | White Feet | White Assay | Red Feet | Red Assay | Total Feet | Total Assay |
|---|---|---|---|---|---|---|
| 14 | 12.0 | 0.07 | . | . | 12.0 | 0.07 |
| 62 | 16.0 | 0.09 | 21.2 | 0.04 | 37.2 | 0.061 |
| Used in Estimate | | | | | | |
| | 16.0 | 0.081 | 21.2 | 0.04 | 37.2 | 0.063 |

## CONGRESS TAILINGS DUMP

## DRILL HOLES AND SAMPLING ON CROSS SECTIONS FOR ESTIMATES

| Cross Section | Drill Hole | White Tailings | | Brown Tailings | |
|---|---|---|---|---|---|
| 0+50 | 24 | 3.5 ft. | 0.07 ozs. | 19.0 ft. | 0.04 ozs. |
| 1 | 30 | 16.9 | 0.065 | 15.6 | 0.045 |
| | 64 (½) | 9.5 | 0.09 | 7.2 | 0.08 |
| Average | | 26.4 | 0.074 | 22.8 | 0.056 |
| 1+50 | 23 | 16.0 | 0.10 | 16.75 | 0.08 |
| | 64 (½) | 9.5 | 0.09 | 7.2 | 0.08 |
| Average | | 25.5 | 0.096 | 23.95 | 0.08 |
| 2 | 4 | 15.0 | 0.07 | 10.0 | 0.046 |
| | 22 | 24.0 | 0.07 | 9.5 | 0.05 |
| | 13 ) 29 ) 13a) (½) | 1.8 | 0.065 | 14.2 | 0.047 |
| Average | | [illegible]0.8 | 0.07 | 33.7 | 0.047 |
| 2+50 | 13 ) 29 ) 13a) (½) | 1.8 | 0.065 | 14.2 | 0.047 |
| | 35 | 25.9 | 0.08 | 8.2 | 0.045 |
| | 18 (½) | . | . | 4.5 | 0.06 |
| Average | | 27.7 | 0.078 | 27.4 | 0.047 |
| 3 | 21 ) 52 ) | . | . | 25.0 | 0.04 |
| | 63 (½) | 6.0 | 0.055 | 12.0 | 0.07 |
| | 18 (½) | . | . | 4.5 | 0.06 |
| | 43 Pit | | | | |
| Average | | 6.0 | 0.055 | 41.5 | 0.051 |

# CONGRESS TAILINGS DUMP

| Cross Section | Drill Hole | White Tailings | | Brown Tailings | |
|---|---|---|---|---|---|
| 3+50 | 31 | . | . | 29.4 ft. | 0.04 ozs. |
| | 16 )<br>16a) | . | . | 33.8 | 0.054 |
| | 63 (½) | 6.0 ft. | 0.055 ozs. | 12.0 | 0.07 |
| | 43 Pit | | | | |
| Average | | 6.0 | 0.055 | 75.2 | 0.051 |
| 4 | 7 | . | . | 12.0 | 0.05 |
| | 19 | . | . | 33.7 | 0.06 |
| Average | | . | . | 45.7 | 0.0575 |
| 4+50 | 1 )<br>1a) | 10.0 | 0.070 | 25.0 | 0.048 |
| | 15 (½) | 9.0 | 0.093 | 3.0 | 0.06 |
| | 26 (½) | . | . | 4.8 | 0.04 |
| Average | | 19.0 | 0.081 | 32.8 | 0.048 |
| 5. | 20 | 31.5 | 0.05 | . | . |
| | 26 (½) | . | . | 4.8 | 0.04 |
| | 15 (½) | 9.0 | 0.093 | 3.0 | 0.06 |
| Average | | 40.5 | 0.060 | 7.8 | 0.048 |
| 5+50 | 14 )<br>62 ) | 16.0 | 0.081 | 21.2 | 0.040 |
| | 8 (½) | . | . | 2.5 | 0.050 |
| | 9 (½) | . | . | 2.5 | 0.050 |
| Average | | 16.0 | 0.081 | 26.2 | 0.042 |
| 6 | 36 | 28.5 | 0.07 | 6.4 | 0.065 |
| | 9 (½) | . | . | 2.5 | 0.05 |
| | 8 (½) | . | . | 2.5 | 0.05 |
| | 27 (½) | . | . | 4.2 | 0.04 |
| Average | | 28.5 | 0.07 | 15.6 | 0.0534 |

# CONGRESS TAILINGS DUMP

| Cross Section | Drill Hole | White Tailings | | Brown Tailings | |
|---|---|---|---|---|---|
| 6+50 | 2 | 26.8 ft. | 0.07 ozs. | 11.2 ft. | 0.046 ozs. |
| | 17 )<br>17a) | 6.8 | 0.076 | 25.2 | 0.061 |
| | 27 (½) | . | . | 4.2 | 0.04 |
| Average | | 33.6 | 0.071 | 40.6 | 0.545 |
| 7 | 37 | 34.6 | 0.065 | 0.7 | 0.09 |
| | 10 (½) | . | . | 6.5 | 0.05 |
| Average | | 34.6 | 0.065 | 7.2 | 0.054 |
| 7+50 | 51 | 31.7 | 0.08 | . | . |
| | 3 )<br>3a) | 8.0 | 0.075 | 28.0 | 0.050 |
| | 10 (½) | . | . | 6.5 | 0.050 |
| Average | | 39.7 | 0.078 | 34.5 | 0.050 |
| 8 | 38 | 33.0 | 0.075 | . | . |
| | 61 | . | . | 29.2 | 0.06 |
| Average | | 33.0 | 0.075 | 29.2 | 0.06 |
| 8+50 | 39 )<br>39a) | 3.6 | 0.09 | 22.7 | 0.0552 |
| 9 | 12 | . | . | 13.0 | 0.05 |
| | 65 | . | . | 27.1 | 0.055 |
| | 11 (½) | . | . | 4.5 | 0.05 |
| Average | | . | . | 44.6 | 0.053 |
| 9+50 | 1 (½) | . | . | 4.5 | 0.05 |

# CONGRESS TAILINGS

## Estimate of Quantity and Value

| | | | |
|---|---|---|---|
| White Tailings | | | 120,000 Tons |
| Brown Tailings | Main Dump | 255,000 Tons | |
| | Upper Dump | 25,000 Tons | 280,000 Tons |
| | | | 400,000 Tons |

Retreated Tailings

| | | | | |
|---|---|---|---|---|
| East Dump | 10,000 Tons | 0.07 ozs. | | |
| West Dump | 5,000 Tons | 0.06 ozs. | 15,000 Tons | |
| | | | 415,000 Tons | |

White Tailings

| | | |
|---|---|---|
| Average all drill holes | 421 ft. | 0.0735 ozs. Gold |
| Weighted Average by sections | | .0739 |

Brown Tailings

| | | |
|---|---|---|
| Average all drill holes | 806 ft. | 0.0527 ozs. |
| Weighted Average by sections | | .0527 |

Mill Test Heads

| | | | Gold | | | Silver | |
|---|---|---|---|---|---|---|---|
| White Tailings #1 | 69.75 ft. | 0.08 | Graham | 0.0697 ozs | 0.47 | 0.485 ozs |
| | | 0.07 | Diehl | | 0.47 | |
| | | 0.059 | Am.Cyan. | | 0.50 | |
| #2 | 66.9 ft. | 0.083 | Graham | | 0.5 | 0.50 ozs |
| | | .08 | Perez | | 0.5 | |
| | | .0692 | Am.Cyan | | 0.51 | |
| Brown Tailings #1 | 74.75 ft. | 0.045 | Graham | 0.0463 ozs | 0.43 | 0.43 ozs |
| | | 0.04 | Diehl | | 0.43 | |
| | | 0.054 | Am.Cyan. | | 0.43 | |
| #2 | 189.7 ft. | 0.050 | Graham | 0.0493 | 0.40 | 0.41 ozs |
| | | 0.045 | Perez | | 0.40 | |
| | | 0.053 | Am.Cyan. | | 0.44 | |

## RETREATED TAILINGS

By Burns and others

**East Dump** — Rough Estimate - 19,000 tons

| | Depth | Goeglein | Diehl |
|---|---|---|---|
| Auger Sample | | 0.10 ozs gold | |
| Pits | | | |
| A | 7.5' | | 0.14 ozs gold |
| B | 7.5' | | 0.05 ozs gold |
| C | 6.2' | | 0.06 ozs gold |

**West Dump** — Rough Estimate - 7,00[illegible] tons

| | Depth | Goeglein | Diehl |
|---|---|---|---|
| Auger Sample | | 0.08 ozs gold | |
| Pits | | | |
| D | 8.0' | | 0.06 ozs gold |
| E | 2.8' | | 0.07 ozs gold |

## WEIGHT OF TAILINGS

### Weight of Brown Tailings in Place -

Cut hole in sand in place 18" x 18" x 15.59 equals 2.92 cu. ft.

Weight wet equals 326#

Weight dry equals 286#

40# Moisture

Wet sand contains 12.26% moisture.

$\frac{286\#}{292}$ Equals 97.9# per cu. ft. in place.

20.5 cu. ft. per ton

### Weight of White Tailings in Place -

White sand was tested separately and weighed approximately 97# per cu. ft. in place.

Metal containers were filled with white tailings and weighed against the same filled with water.

Brown tailings checked August 9th, made 102.06# per cubic foot.

# CONGRESS MINE

## Total Tailings March 11, 1936

| Cross Section | Square Feet | Square Feet | Tons |
|---|---|---|---|
| | | 797 | 1992 |
| 0 | 1594 | 3022 | 7555 |
| 0+50 | 4451 | 5173 | 12932 |
| 1 | 5895 | 7033 | 17582 |
| 1 50 | 8371 | 9840 | 24600 |
| 2 | 11509 | 10650 | 26625 |
| 2+50 | 9792 | 10349 | 25872 |
| 3 | 10906 | 10900 | 27250 |
| 3+50 | 10894 | 9582 | 23955 |
| 4 | 8271 | 8452 | 21130 |
| 4+50 | 8632 | 8015 | 20037 |
| 5 | 7398 | 6692 | 16730 |
| 5+50 | 5985 | 7018 | 17545 |
| 6 | 8050 | 7934 | 19835 |
| 6+50 | 7819 | 7880 | 19700 |
| 7 | 7940 | 8787 | 21968 |
| 7+50 | 9634 | 10246 | 25615 |
| 8 | 10858 | 8933 | 22332 |
| 8+50 | 7008 | 5360 | 13400 |
| 9 | 3714 | 3000 | 7500 |
| 9+50 | 2286 | 1143 | 2857 |
| 10 | 150807 | 150806 Sq. Ft. | 377012 Tons |

Average Area of Cross Section x 50 ft. ÷ 20 = Area x 2.5 = Tons

# CONGRESS TAILINGS

## Calculation of Tonnage

### White Tailings

| Cross Section | Sq. Ft. | Tons |
|---|---|---|
| 0 | | |
| | 2179 | 5188 |
| 0+50 | | |
| | 3158 | 7518 |
| 1 | | |
| | 4266 | 10157 |
| 1+50 | | |
| | 5267 | 12541 |
| 2 | | |
| | 4049 | 9641 |
| 2+50 | | |
| | 1709 | 4212 |
| 3 | | |
| | 746 | 1776 |
| 3+50 | | |
| | 421 | 1002 |
| 4 | | |
| | 2112 | 5029 |
| 4+50 | | |
| | 4266 | 10157 |
| 5 | | |
| | 3935 | 9369 |
| 5+50 | | |
| | 3136 | 7467 |
| 6 | | |
| | 3075 | 7322 |
| 6+50 | | |
| | 3483 | 8283 |
| 7 | | |
| | 4199 | 9998 |
| 7+50 | | |
| | 4520 | 10762 |
| 8 | | |
| | | 120,422 Tons |

Average areas x 50 ft. ÷ 21 = Area x 2.381 = Tons

## CONGRESS TAILINGS

White Tailings Average Gold Content

| Cross Section | Gold | Area in Sq. In. | Product |
|---|---|---|---|
| 0 | 0.07 ozs. | 2.550 | 17850 |
| 0+50 | 0.07 | 4.411 | 30877 |
| 1 | 0.074 | 5.694 | 42136 |
| 1+50 | 0.096 | 7.956 | 76378 |
| 2 | 0.07 | 8.897 | 62279 |
| 2+50 | 0.078 | 4.056 | 31637 |
| 3 | 0.055 | 1.604 | 8822 |
| 3+50 | 0.055 | .781 | 4296 |
| 4 | 0.06 | .566 | 3396 |
| 4+50 | 0.081 | 6.191 | 50147 |
| 5 | 0.06 | 7.459 | 44754 |
| 5+50 | 0.081 | 5.133 | 41577 |
| 6 | 0.07 | 4.903 | 34321 |
| 6+50 | 0.071 | 4.936 | 35046 |
| 7 | 0.065 | 6.209 | 40358 |
| 7+50 | 0.078 | 7.229 | 56386 |
| 8 | 0.075 | 7.234 | 54255 |
| | | 85.811 Sq. In. | 634,515 |

Average Gold 0.0739 ozs. per ton

# CONGRESS MINE

## BROWN TAILINGS

| Cross Section | Total Tons | White | Brown |
|---|---|---|---|
| 0 | 1992 | | 1992 |
| 0+50 | 7555 | 5188 | 2367 |
| 1 | 12932 | 7518 | 5414 |
| 1+50 | 17582 | 10157 | 7425 |
| 2 | 24600 | 12541 | 12059 |
| 2+50 | 26625 | 9641 | 16984 |
| 3 | 25872 | 4212 | 21660 |
| 3+50 | 27250 | 1776 | 25474 |
| 4 | 23955 | 1002 | 22953 |
| 4+50 | 21130 | 5029 | 16101 |
| 5 | 20037 | 10157 | 9880 |
| 5+50 | 16730 | 9369 | 7361 |
| 6 | 17745 | 7467 | 10278 |
| 6+50 | 19835 | 7322 | 12513 |
| 7 | 19700 | 8283 | 11417 |
| 7+50 | 21968 | 9998 | 11970 |
| 8 | 25615 | 10762 | 14853 |
| 8+50 | 22332 | 0 | 22332 |
| 9 | 13400 | 0 | 13400 |
| 9+50 | 7500 | 0 | 7500 |
| 10 | 2857 | 0 | 2857 |
| | 377, 012 Tons | 120,422 Tons | 256,590 Tons |
| | | Upper Dump | 23,000 |
| | | Roasted Tailings | 279,590 |

# CONGRESS TAILINGS

Brown Tailings                    Average Gold Content

| Cross Section | Gold | Square Feet |
|---|---|---|
| 0 | | 0 |
| 0+50 | 0.0400 ozs. | 1694 |
| 1 | 0.0560 | 2336 |
| 1+50 | 0.0800 | 3199 |
| 2 | 0.0470 | 5948 |
| 2+50 | 0.0470 | 7256 |
| 3 | 0.0510 | 9904 |
| 3+50 | 0.0510 | 10406 |
| 4 | 0.0575 | 7917 |
| 4+50 | 0.0480 | 4763 |
| 5 | 0.0480 | 2736 |
| 5+50 | 0.0420 | 2777 |
| 6 | 0.0534 | 4986 |
| 6+50 | 0.0550 | 4734 |
| 7 | 0.0540 | 4059 |
| 7+50 | 0.0500 | 5116 |
| 8 | 0.0600 | 6337 |
| 8+50 | 0.0552 | 7008 |
| 9 | 0.0530 | 3714 |
| 9+50 | 0.0500 | 2286 |
| 10 | | 97176 |

Average                    0.0527 ozs.

## CONGRESS MINE

## UPPER DUMP ON INCLINE CLAIM

## BROWN TAILINGS

| Hole | Depth | Gold |
|---|---|---|
| # 6 | 10.5 ft. | 0.046 ozs. * |
| #25 | 29.5 | 0.045 |
| #66 | 28.7 | 0.07 |
| | Average | 0.055 ozs. |

The upper dump was estimated in 1918 to contain 34,000 tons of roasted tailings. In this estimate the whole dump contained 9,275,000 cu. ft. although 6 out of the 11 holes did not reach bottom. The upper dump was estimated at 850,000 cu. ft.

For this report, the dump is assumed at 460,000 cu. ft. but at 20 cu. ft. per ton instead of 25.

* Struck timber and was abandoned.

# CONGRESS VEIN

## Gob Samples #2 Shaft

| Location and Level | | No.* | 1st Series | 2nd Series | Average |
|---|---|---|---|---|---|
| 850 West | 141 From Station | 1 | $16.29 | | |
| | 153 " " | 2 | | $4.48 | $10.38 |
| 1100 West | 312 From Station | 3 | $ 5.52 | | |
| | 334 " " | 4 | | $2.76 | $ 4.14 |
| 1225 West | 213 From Station | 5 | $13.11 | | |
| | 66 " " | 6 | | $7.60 | $10.35 |
| 1300 West | 20 From Station | 7 | $11.38 | | |
| | 50 " " | 8 | | $3.80 | $ 7.59 |
| | Averages | | $11.57 | $4.68 | |

Total Average - $8.11 0.232 oz

Above samples were taken from the best of the stoped area.

When the first samples proved to be valuable, the second series was taken from points somewhere near the first samples on the same levels.

Samples were taken by cutting the lagging, throwing back the coarse rock and sampling the finer fill left behind. It is difficult to get a correct proportion of coarse and fine material.

It is evident that material now having a value was left, or sorted out and thrown into the fill. There must be good and poor areas.

The fill can be tested by drawing out a narrow stope from level to level, hoisting both coarse and fine, screening out the coarse on the surface and sampling the finer portion. This will give the value and proportion of the fine material and indicate the cost of extraction.

*Sample Number

# OTHER SAMPLES #2 SHAFT

| Sample No. | Level & Location | Value |
|---|---|---|
| 1 | 300 West | $5.17 |
| 2 | 300 West | $3.45 |
| 3 | 550 East | $2.10 |
| 4 | 650 East | $9.10 |
| 5 | 700 West | $6.55 |
| 6 | 700 West | $7.00 |
| 7 | 700 East | $4.20 |
| 8 | 750 West | $0.69 |
| [illegible] | 750 East | $3.50 |
| 10 | 750 West | $5.60 |
| 11 | 800 West | $1.40 |
| 12 | 1050 West | $4.14 |
| 13 | 1050 West | $1.40 |
| | Average | $4.17 per ton |
| | 8 Samples - Average | $8.11 per ton |
| | Total 21 Samples - Average | $5.65 per ton |

## CONGRESS VEIN

**Shaft #2** — 200 to 300 Level West

### Ore Face in Stope

| Sample No. | Width Sampled | Gold | Value |
|---|---|---|---|
| 25 | 14" | .44 | $15.40 |
| 26 | 6" | .02 | .70 |
| 27 | 16" | .10 | 3.50 |
| 28 | 32" | .02 | .70 |
| 29 | 60" | .10 | 3.50 |
| 30 | 60" | .04 | 1.40 |
| 31 | 27" | .26 | 9.10 |
| 32 | 5" | .08 | 2.80 |
| 33 | 24" | .10 | 3.50 |
| 34 | 29" | .34 | 11.90 |
| 35 | 24" | .20 | 7.00 |
| 36 | 18" | .34 | 11.90 |
| 37 | 22" | .28 | 9.80 |
| 38 | 18" | .16 | 5.60 |
| 39 | 27" | .08 | 2.80 |
| 40 | 31" | .06 | 2.10 |
| 41 | 16" | .06 | 2.10 |

40 ft. 23" $10.14

50 ft. 22" $9.40

80 ft. 21" $8.30

170 ft. sampled at 10 foot intervals

# CONGRESS MINE

## PATENTED CLAIMS

| Claim | Mineral Survey Number | |
|---|---|---|
| Queen of the Hills | 879 | |
| Congress | 878 | |
| Fraction | 883 | |
| Niagara | 880 | |
| Mosouri | 881 | |
| Why Not | 882 | |
| Incline | 1193 | |
| Golden Thread | 1352 | |
| Golden Eagle | 1191 | |
| Excelcior | 921 | |
| Rich Quartz | 1192 | |
| Ohio | 1190 | |
| Old State | 1189 | To cover well |
| Snow Storm | 1188 | |
| Niagara Mill-Site | 880 | Covers well on Martinez Creek |

CONGRESS MINE

UNPATENTED CLAIMS

| CLAIM | DATE OF LOCATION | RECORD BOOK OF LOCATION NOTICE | PAGES |
|---|---|---|---|
| Bellick | September 6, 1887 | 24 | 291 |
| Remnant | March 12, 1888 | 25 | 314 |
| Boundary | February 1, 1893 | 35 | 161 |
| Sunnyside | February 20, 1897 | 45 | 499 |
| Highland | February 20, 1897 | 45 | 496 |
| Keystone | January 4, 1899 | 50 | 364 |
| East Extension of Golden Thread | March 8, 1899 | 51 | 156 |
| Martinez | February 4, 1903 | 66 | 591 |
| Ophir | November 30, 1933 | 36 | 341 |

PRODUCTION FROM CONGRESS MINE by W. F. STAUNTON

| | | | |
|---|---|---|---|
| Production | | 692,332 Tons | |
| Gold | | 388,477 Ozs. | |
| Silver | | 345,598 Ozs. | |
| Value | Gold | $20.67 | $8,029,900 |
| | Silver | 0.60 | 204,560 |
| | | | $8,234,460 |
| Value per ton | Gold | $11.60 | |
| | Silver | .19 | |
| | | $11.79 | |
| | Tailings | 1.20 | |
| | Gross Value per Ton | $12.99 | |
| Value | Gold | $35.00 | $13,597,000 |
| | Silver | 0.75 | 259,000 |
| | | | $13,856,000 |
| Value per ton Gold | | $19.63 | |
| | | .37 | |
| | | $20.00 | |
| Tailings (Sampled) | | 1.95 | |
| | | $21.95 | |
| | | @ 80.00 au | 31,078,160 |
| | | 2.00 ag | 691,196 |
| | | 1973 | $31,769,256 |

# WAGE SCALE

| CLASS OF LABOR | Per Hour | 7 Hour Day |
|---|---|---|
| **UNDERGROUND:** | | |
| Shift Boss | $ .70 | $ 4.90 |
| Timberman | .57 | 3.99 |
| Timberman Helper | .50 | 3.50 |
| Miner | .57 | 3.99 |
| Slusherman | .57 | 3.99 |
| Trammer - mule tram | .50 | 3.50 |
| Trammer - hand | .50 | 3.50 |
| Mucker | .50 | 3.50 |
| Laborer | .50 | 3.50 |
| Hoistman | .57 | 3.99 |
| Tool Nipper | .50 | 3.50 |
| Repairman | .57 | 3.99 |
| Repairman Helper | .50 | 3.50 |
| Miscellaneous | .50 | 3.50 |
| **MILLING:** | | **8 Hour Day** |
| Mill Operator | $ .63 | $ 5.04 |
| Mill Helper | .50 | 4.00 |
| Mill Labor | .43 | 3.44 |
| **SHOPS - SURFACE:** | | **7 Hour Day** |
| Shop Boss | $ .70 | $ 4.90 |
| Blacksmith | .64 | 4.48 |
| Welder | .64 | 4.48 |
| Electrician | .64 | 4.48 |
| Carpenter | .64 and .75 | 4.48 and 5.25 |
| Shop Helper | .57 | 3.99 |
| Shop Labor | .50 | 3.50 |
| Miscellaneous | .43 | 3.01 |

# REPORT UPON CONGRESS TAILINGS
## MERRILL METALLURGICAL COMPANY,
## December 31, 1915

## Sampling

This deposit was first sampled in a preliminary way by a representative of the Merrill Metallurgical Company, early in August, 1915. As a result of this first sampling, the deposit was estimated to contain 400,000 tons, assaying \$1.42 gold, and \$0.18 silver, or a total of \$1.60.

A second sampling was made three months later and the deposit was very carefully drilled with approximately 75 holes, the cores being divided into approximately 150 assay samples. At the same time, the dumps were very carefully surveyed and contoured.

As a result of this work, a very accurate estimate of tonnage and value is possible and this final estimate is as follows:

| | | Present Price |
|---|---|---|
| Total tons tailings available............. | 505,000 | |
| Average assay value....................... | \$1.46 gold | \$ 2.47 |
| Average assay value....................... | \$ .21 silver | \$ .31 |
| Total assay value......................... | \$1.67 | \$ 2.78 |
| Total value tailings...................... | \$843,360 | \$ 1310000 |

## Testing

Approximately eighty-five separate cyanide tests were made, most of them upon an average sample of the total tailings; a number of tests were also made upon individual portions of the tailings - roasted, unroasted and slime. Tests were run with and without regrinding, and with varying cyanide strengths and treatment periods. The results of these tests may be summarized as follows:

Sand must be ground to pass at least 100 mesh.
Time of treatment required: 24 to 36 hours.
Average recovery value per ton \$1.10
Average cyanide consumption per ton 0.6#
Average zinc consumption per ton 0.2#
Average lime consumption per ton 10.0#

## Operating Cost

After a careful study of local conditions at the mine, particularly as regards labor and power supplies, the following estimate of costs and profits is obtained.

Transportation, dump to mill bins ................\$0.10
Power, exclusive of drag lines and water supply.... .10
Labor, exclusive of drag lines:

| | | | |
|---|---|---|---|
| 3 @ \$4 | - \$12.00 | | |
| 3 @ 2.50 | - 7.50 | | |
| 1 @ 4.50 | - 4.50 | | |
| 1 @ 3.00 | - 3.00 | | |
| 2 @ 2.50 | - 5.00 | | |
| 1 @ 8.00 | - 8.00 | - \$40.00................ | .10 |

| | | |
|---|---|---|
| Supplies, Kcy .6# @ 30¢ | $0.18 | |
| Zinc 1/5# PTO 80# @ 25¢ | .05 | |
| Lime 10# PTO @ 1/2¢ | .05 | |
| Miscellaneous | .05 | |
| Water Supply | .03 | |
| Stacking tailings | .05 | |
| Miscellaneous | .04 | $0.75 |

Estimated recovery on general average sample:

| | | |
|---|---|---|
| Total gold and silver | 1.10 | |
| Total operating | .75 | |
| Gross Profit | .35 | |
| Royalty 10% | .35 | |
| Net Profit | .315 | |
| Cost of plant, $75,000.00 | .148 PTO | |
| | .167 | |
| Interest on $75,000, 3½ years @ 6% | .03 PTO | |
| Actual net profit | $0.137 - | $69,185.00 |

Construction Cost

The following estimate covers the actual construction cost of a 400-ton regrinding, agitation, filtration, cyanide plant to retreat to Congress tailings:

| | |
|---|---|
| Drag Line, boiler and hoist | $1,300.00 |
| Drag Line, cable 2500 ft. @ $0.82 | 205.00 |
| Drag Line bucket | 75.00 |
| Loading pocket (old timber) | 250.00 |
| Belt conveyor and trestle (old and new), or bucket elevator | 600.00 |
| Conveyor over bins | 250.00 |
| 400 ton bin (new timber) | 1,500.00 |
| Slime pulping equipment | 500.00 |
| Sluicing feeders and launders | 200.00 |
| Classifiers - 2 | 1,000.00 |
| Tube Mills, 2 (erected) | 6,000.00 |
| Lime shafting and pulleys | 500.00 |
| 1 - 240 HP Diesel Eng (erected comp) 4 cyl, 14x21, 88,500# | 18,000.00 |
| 1 - 35 HP generator | 350.00 |
| 1 - 10 HP, 3-5 HP, 1-3 HP motors | 750.00 |
| Electric wiring and installations | 750.00 |
| Sand pumps, 2 - 3" | 500.00 |
| Dorr thickness 26'x7' (old tanks) 3 @ 1,400 | 4,200.00 |
| Dorr agitators, 30'x12' - 4 @ 2,300 | 9,200.00 |
| Filters, 3 - 12"x16', $12,000 f.o.b.(erected) | 13,500.00 |
| Vacuum pump 870 f.o.b. NY (erected) | 1,250.00 |
| Air compressor 300 cu. ft. @ 20#, $500 fob La. | 750.00 |
| Solution sumps, 5 (old tanks) moving and erect. | 750.00 |
| Pumps: 1-3" triplex 4" GPM) 2-3" centrifugal, Sol. fob 70.) | 250.00 |
| Clarifying equipment (vacuum leaves) | 500.00 |

| | |
|---|---|
| Precipitation equipment, 1 40 frame 36" comp........ | 4,500.00 |
| Tailings conveyor and trestle........................ | 500.00 |
| Tailings drag line................................... | 1,580.00 |
| Buildings............................................ | 3,500.00 |
| Piping, air, solution, water, oil.................... | 1,290.00 |
| Miscellaneous........................................ | 1,000.00 |
| TOTAL............ | $75,500.00 |

Comments

The question of power plant has been very closely studied, the total requirements approximating 240 HP exclusive of drag line equipment.

There are two feasible ways of obtaining this power - to equip the existing No. 2 steam plant with generators, electric transmission and motors in the cyanide plant, or to install a new Diesel gas engine unit in the cyanide plant, with approximately 85% of the power belted directly from the main line shaft.

The first of these alternatives allows a savings in initial installation cost of approximately $11,000.00, but in spite of this the use of a Diesel type power plant shows an economy which may reasonably be expected to yield a net saving from $10,000 to $15,000. in three the one-half years operations. Furthermore, the #2 power plant might be requisitioned by the Congress Company should work be resumed in the mine.

A filter plant is essential in handling these tailings, first because a very close saving of the soluble values must be had, and second because it appears highly desirable to stack the re-treated tailings on company land.

In the Estimate of Plant Cost, account has been taken of such tanks and material as will be available for use in building the new plant, and it is to be remembered that all new machinery and equipment is figured at from 5 to 20% discount allowed the Merrill Metallurgical Company as dealers. In other words, the cost of this plant, if built under usual purchasing conditions, would undoubtedly be from 10% to 15% above these figures.

Obviously the two barriers to the profitable treatment of these tailings are the poor extraction obtained, even with very fine grinding, and the high cost of chemicals consumed. Under normal prices for cyanide and zinc, the gross profits would be increased by $40,000.00. Also, there are indications, at this time that fuel oil may continue to advance in price, at least during the continuance of the war.

MERRILL METALLURGICAL COMPANY

C O P Y

W. F. STAUNTON
MINING ENGINEER
124 WEST FOURTH STREET
LOS ANGELES, CAL.

Los Angeles, Calif., Oct. 27, 1933

Mr. Gerald Sherman,
120 East 85th Street,
New York, N. Y.

Dear Mr. Sherman:

I have not been able to answer your letter of the 14th as soon as I should have done. It was welcome, however, and I am quite willing to give you any information I may have relating to the Congress mine about which you wrote. Also, your letter brings to mind meetings with you many years ago, when you were at Bisbee, which I remember with pleasure.

I have no doubt that somewhat better metallurgical results could be obtained today, and considerable reduction in costs also, as compared with our rather crude methods of 30 years ago. For one thing, our practice involved bedding and drying the tailings from the concentrating mill and then reclaiming them for cyanidation, all the cost of which would be eliminated today. I think our recovery was a little better than your figures seem to show. The concentrate recovery was very low. I have yearly figures from 1894 to 1910 showing the amount of gold to account for in the concentrating mill, by tons milled and battery assays, as $7,118,644. and gold actually paid for as $4,062,239. This would indicate a saving of only 57.20%. The cyanide plant figures for the same time show $3,192,219. contained and $2,769,566. paid for, a recovery of 86.76%. Taken together the total recovery seems to have been 94.33%. It is partly confirmatory that nobody has yet succeeded in reworking our tailings, although many have tried.

You mention smelter recovery. I find a memorandum of a yearly contract I made in 1894 with the Kansas City Works (shipment probably to El Paso) showing the following rates: Silver, 95%, gold, $19.50. Freight and treatment, $15.00 per ton f.o.b. Prescott. Iron, 15¢ up or down from neutral basis. As the excess of iron was about 30% this gave a net rate of $10.50 frt. & Treatment f.o.b. Prescott. It cost us $12.80 a ton to haul the concentrates to Prescott, so that the total charge on concentrates was $23.30 per ton. As the average grade was 7 oz., this meant a charge of $3.33 per ounce of gold. Adding the $1.11 difference between $19.50 and $20.67 makes a total charge on the gold of $4.44 per ounce.

We did better after the railroad was built, but the last contract I have a memorandum of, made for three years in 1901, was: Silver, 95%; gold, $19.50; Freight & Treatment f.o.b. Congress Junction, $16.00 per ton, with 15¢ per unit for iron in excess of silica (say, $4.50) leaving a net charge of $11.50 freight and treatment.

I do not think there is very much to be looked for in increased mill recovery; possibly 50¢ a ton at most. But if the concentrates should prove susceptible to cyanide treatment so as to put the whole product into bullion, there would be a large saving, perhaps $1.00 a ton.

The really big difference today is, of course, the 50% increase in the price of gold. We used to consider $7.00 a ton as about the splitting point as between ore and waste, that is, roughly, 0.35 oz. Today, with $30. gold, this would be only 0.233 oz. It seems probable that, allowing for savings under modern conditions, anything above 0.2 oz would pay. The question is, how much ore above that grade can be reasonably counted on, and I don't see any way of arriving at an answer to that than actual examination and sampling. I believe there is a large tonnage in the Niagara mine, but this is a mere guess.

In the Congress mine itself, as distinguished from the Niagara, I think there are possibilities in the old stope fillings, on account of the way in which mining was done. The vein being narrow and flat, about 25 deg. dip, it was usually necessary to blast some of the hanging wall, which, however, frequently carried high grade stringers. This wall rock constituted the filling which kept close to the stoping faces. The mineral was very brittle and high grade and while attempts were made to keep split lagging between the working face and the filling, a great deal of fine mineral was undoubtedly blasted into the filling and lost. This condition may easily prove to have given sufficient value to the gob to make reworking profitable under modern conditions, as, for instance, the use of drag scrapers and local separation of fine and coarse and perhaps some hand sorting, the reject going directly back into the stopes, saving hoisting on all but the rough concentrates.

In regard to tonnage of such gob available, there should be at least as much as, and probably more than, the amount of ore produced, say 700,000 tons.

Certain parties have undertaken to work the surface waste dumps. I have never regarded them as valuable.

Owing the the increased price of gold, there are certainly some possibilities in reworking the old stope fills on the Congress vein, and in milling ore already opened on the Niagara vein, but I think there is a really good chance in the Niagara in new ground at greater depth.

Underlying the Niagara vein, which has an easterly-westerly strike, and dip of perhaps 40 deg., there is a greenstone dike with slightly different strike, and dip of around 25 deg. This dike is almost identical in character with the Congress dike which carried the ore in that mine. The Niagara vein intersected this dike at about 1975 feet depth in the extreme easterly part of the mine close to the big fault. The dike was heavily mineralized at the intersection and the ore in the dike was of the same character and grade as in the Congress as distinguished from that in the Niagara vein, which belongs to the class entirely in the granite. This high grade extended easterly to the big fault where it was cut off. To the west the work was at first confined to the dike but as the distance from the intersection increased the high grade gradually failed and crosscuts were run into the hanging to the Niagara, and thereafter the work was done on that vein. The line of intersection would run downward to the northwest.

The Niagara shaft is an incline on the vein and its course happens to coincide closely with the course of the intersection of the planes of the two veins. It seems highly probable that a new line of high grade stopes can be opened by sinking the Niagara shaft a few hundred feet below its present depth of 2050 feet. The 1900 level is connected to the No. 4 shaft, 700 feet west, so that there is good ventilation. Sinking of this character is comparatively cheap. The little water met with is readily bailed. It amounts to little more than running a drift on an incline.

You will perhaps be surprised at having drawn down on yourself such a wordy reply. My apology is my interest in the old Congress from my many years connection and my belief that it has a future for somebody.

Sincerely yours,

W. F. Staunton

WFS/H

W. F. STAUNTON
MINING ENGINEER
124 WEST FOURTH STREET
LOS ANGELES, CAL.

Los Angeles, Calif., Jan. 11, 1935.

Mr. Gerald Sherman,
120 East 85th Street,
New York, N. Y.

Dear Mr. Sherman:

We are being constantly told by the New Dealers that history and human experience are no longer safe guides; that similar causes can no longer be depended on for similar effects and as precedants have gone the way of supply and demand, but I guess I am old fashioned and feel better able to guess what will happen if I know what has already happened. So I am going to be historical in replying to your letter of January 7 asking about the water supply at Congress, and to do so, I have been digging into sundry old note books.

We bought the mine from the Diamond Joe Renolds estate early in 1894, and I went there in June as superintendent. We had been told that the well in Martinez Creek could be depended on for an ample supply for the 40 stamp mill and the camp.

We seem to have had a little shortage at first, on account of imperfect arrangements for saving water, which were being improved, after which we got along fairly well through the summer, which was very dry, with no good rain until late in October.

The first definite note I find is dated July 29, 1894 and is as follows:

> "Water for mill supplied by 7-1/2 x 4 x 10 Knowles pump (in well) running at 120 total strokes per min. 6-1/2 hrs per 24. This includes all water used at hoist and town as well as mill, and figures 25,567 gals. at full stroke and no allowance for slip. Probably 20,000 gals a day runs everything. Water raised 500 ft. through about 1 mile of 4 in. pipe."

That was the picture when we started. As we were milling 100 tons a day, it would appear that we were getting along on the almost incredibly small allowance of 200 gals. per ton. We began experimenting with the cyanide process, and, of course, the camp use of water increased, and I find a note dated Nov. 11, 1894 showing that I made a check-up and decided that we needed 30,000 gals. a day of new water for everything. That would be 300 gals. per ton, which is probably about right.

Under normal weather conditions the Martinez well proved sufficient for the 40 stamps. In dry spells we dug ditches to bed rock in the sand, leading them to the well, and in a year or two I had a series of shallow wells put down up stream from the main well, and connected by a 4 in. pipe line that was made to act as a siphon. With such expedients, and in extreme cases by having the railroad people dump some cars of water into the well, we kept things going until we built the second 40 stamp mill, in 1901, when we bought the O'Neil ranch on Date Creek 8 miles north, where there is an abundant supply, and put in a 4 in. pipe line.

At that time the fiscal year of the company was from Sept. 1 to Aug.31, and the only definite and complete statement I can find fortunately covers parts of 1901 and 1902 when we bought the O'Neil ranch. The statement is seggregated by months, but I will condense to totals.

Year September 1, 1901 - Aug. 31, 1902.

| | | |
|---|---|---|
| Tons milled, | 51,538. | Average battery assay, $13.43. |
| Tons cyanided, | 58,935. | |

| | | |
|---|---|---|
| Net returns, concentrates, | | $362,812. |
| " " cyanide bullion, | | 290,869. |
| Sundry rec'ts., store, boarding house, etc., | | 43,371. |
| | | 697,052. |
| Disbursements: | | |
| Operating, | 515,461. | |
| All other, | 59,940. | |
| | | 575,401. |
| Net, | | $121,651. |
| Included in "other" expense are Capital items: | | |
| Purchase of Date Creek Ranch and improving same for water supply, | 19,292. | |
| Repairs and improvements to No. 1 mill preparatory to starting same, | 17,404. | 36,696. |
| Actual profit above operating expense and mine development, | | $158,347. |

In regard to the actual cost of water supply, I find the following tabulation in one of the old note books:

Water Supply, - Cost.

| Year. | Tons Milled. | Fuel Cost. | Labor and All other Expense. | Total Cost Of Water. |
|---|---|---|---|---|
| 1894 | 22,826. | $2,338.88 | $2,141.23 | $4,480.11 |
| 1895 | 36,623 | 2,433.81 | 3,067.93 | 5,501.74 |
| 1896 | 34,110 | 2,049.58 | 3,072.60 | 5,122.18 |
| 1897 | 36,411 | 1,725.30 | 1,914.40 | 3,639.70 |
| 1898 | 38,336 | 2,131.50 | 3,585.41 | 5,716.91 |
| 1899 | 35,093 | 2,310.64 | 8,407.14 | 10,717.78 |
| 1900 | 35,212 | 3,367.66 | 8,844.82 | 12,212.48 |
| 1901 | 44,868 | 3,691.27 | 2,982.52 | 6,673.79 |
| | 283,479 | 20,048.64 | 34,016.05 | 54,064.69 |

I am unable to account for the two excessively high years, unless it was that we were buying water from the railroad before putting in the Date Creek plant. The average cost, 19¢ per ton, seems very high.

I am told that people who have been recently trying to work the tailings have relied on the mine water. I could have told them that it did'nt amount to anything. I have a memorandum of the amount of water hoisted from the 2050 level of the No. 5 shaft for the year 1906-7, which was 12,983 tons, and that, if I have figured it correctly, corresponds to only about 6 gallons per minute.

With best wishes, I am,

Sincerely yours,

W. F. Staunton

WFS/H

60



# Ore Possibilities at the Congress Mine

*Geological History of This Interesting Old Property Suggests Advisability of Further Exploratory Work and Development*

[redacted]

62

# CONGRESS MINE

# METALLURGICAL REPORT

# CYANIDE PLANT

# FOR

# TAILINGS DUMP

BY

W. A. LEDDELL

March 16, 1936

W. A. LEDDELL
MECHANICAL AND
METALLURGICAL ENGINEER

MILLS BLDG.
EL PASO, TEXAS

511-5th AVE.
NEW YORK CITY, N. Y.

March 16, 1936.

The Congress Mining Corporation,
100 Broadway,
New York City, N.Y.

Gentlemen:

In accordance with your request, I am herewith submitting my report on the cost of rehabilitation of the present cyanide plant for the treatment of the old tailings from the Congress Mills.

In order to obtain this desired information, it was necessary to:

1st - Survey and sample the dump for quantity and value; maps, weighted values of samples and tonnages are submitted in a separate report.

2nd - Using representative samples of the above sampling, make the necessary metallurgical tests in [illegible] oz. in gold and 0.45 oz. in silver per ton. Assays of samples of the brown sand tailings averaged 0.0527 oz. in gold and 0.33 oz. in silver per ton.

2. As a conservative figure a recovery of 85% of the gold and 75% of the silver can be expected on the white sand and 70% of the gold and 65% of the silver can be expected on the brown or roasted tailings.

3. The third item above "Design the Flow Sheet and Calculate the Size of the Equipment" is taken up in detail under the following:

1st - Metallurgaical Treatment in General and the Recoveries to be Expected;

2nd - Explanation of Flow Sheet Selected;

3rd - Estimate of Operating Power Required;

4th - Estimate of Operating Costs;

5th - Estimate of the Cost of Plant Changes and Additions.

METALLURGICAL TREATMENT IN GENERAL:

One of the first conditions to be investigated was to determine, if any of the gold was in a water soluble state. This was not found to be so.

Flotation was tried but resulted in low recoveries.

The cyanide tests made were to determine the fineness of grinding necessary, the time of agitation treatment, the dilution ratio advisable, and the amount of lime required per ton of ore to give protective alkalinity.

Grinding to 78% -200 mesh seems to be the economic limit to which to grind, although the extraction is improved, by finer grinding. Time of agitation treatment is to a certain extent interchangeable for finer grinding and to a certain extent is cheaper than excessive power and steel cost to produce the finer meshes.

The time of agitation treatment is found to be at least 48 hours on the coarser meshes. Extraction may be improved by continuing this treatment for 60 hours which can be done in the same agitators by changing the dilution ratio of sand to liquid.

Agitation will take place at dilution ratios of 2 to 1 or 1-1/2 to 1.

The amount of lime required is found to be from 12# per ton of ore for the white sands to 20# per ton for the brown sands. A high recovery of tailings water may lower this lime consumption.

The settling ratio of the ore is very rapid indeed and no extra lime will be required to assist the settling rate as is often the case.

A summary of the tests on both white and brown sands indicated that recoveries as follows can be relied upon:

| | Gold | Silver |
|---|---|---|
| White Sands | 85% | 75% |
| Brown Sands | 70% | 65% |

EXPLANATION OF THE FLOW SHEET SELECTED:

Recovery of Tailings.- The most economical method of collecting and delivering the tailings to the mill will be by means of a slusher hoist and a belt conveyor.

A 35-H.P. hoist has been recommended by the manufacturers for the work of handling 300 tons in 8 to 10 hours. In order to provide ample power it is proposed to power this hoist with a 50-60-H.P. Diesel engine.

The tailings sands are very clean and have practically no old timber in the sand but the slusher hoist will deliver to a grizzly and screen over the movable hopper feeder over the belt conveyor which will deliver to the present storage bins at the mill.

In addition to the bins the conveyor equipment is on hand for most of the incline conveyor.

Later in the operation a 200 to 300-ft. cross conveyor will have to be added to gather the sands from each end of the tailings deposit. This is provided for in the estimate.

Mill Bins.- The 250-ton mill bin which is already erected on the property is divided and it will therefore have to be provided with 2 feeders to deliver the sands to a common conveyor which will deliver these sands to the bowl of the bowl classifier which is arranged in closed circuit with the ball mill.

Lime will have to be fed to the ball mill in amounts of approximately 1-1/2 to 3 tons per day. Optionally it could be added to the feeder conveyor.

Grinding.- The ball mill at present in the plant is a 6' x 4' Colorado Iron Works mill but is somewhat small for the required tonnage. Therefore, a larger mill (7' x 36") has been provided for in the estimate.

By feeding the sands directly to the bowl of the bowl classifier it is intended to classify out as much as possible of the 55% of -200 mesh material in the original feed.

Theoretically the amount of grinding to be done could be accomplished by about 50 H.P. However, the actual tonnage of -100 mesh which will be required will be affected by so many variables, such as the efficiency of the classifier, the hardness of the ore, etc. that while the present ball mill, which requires when fully loaded about 60 H.P., the actual reduction of 300 tons per day of sands to all minus 100 mesh material would be a little in doubt. Therefore a 100-H.P. motor is recommended.

One other point in favor of more mill capacity is the fact that finer grinding improves the percentage of recovery.

If the present mill should be used it would have to be provided with a new lining, scoop and possibly have the grates in the end removed.

Classification.- In order to get efficiency in the grinding circuit, there should be good classification. This will require a bowl classifier and in order to get good separation on the finer mesh sizes a high dilution must be carried in the classifier. This is the reason for the separate solution circuit of 75 G.P.M. shown in the flow sheet.

Settling.- The capacity of a cyanide plant is also limited by the area of the thickeners. Preliminary tests have been made which show a very high settling rate.

At a plant capacity of 300 tons per day the settling rate in the last thickener will be 2.85 sq.ft. per ton per 24 hours.

Agitation.- The underflow from the primary thickener "U" is delivered by a 4" duplex diaphragm pump to a series of 3 - 36' x 17' Dorr type agitators. The flow sheet indicates agitation at 2 to 1 but the agitators are calculated to give 60 hours agitation at a dilution of 1-1/2 to 1. They will require about 100 cu.ft. of free air per minute at 20 to 25 pounds pressure. This will require about 10 H.P. which with the mechanical power will give a total of 15 H.P. for the agitator power.

Decantation.- The pulp from the agitators is diluted and discharged by gravity to the decantation thickeners. The overflow of thickener "V" is returned by pumping to the mill solution tank for further enrichment in the grinding circuit.

Fresh water and barren solution enter the circuit at either "X" or "Y" as practice may indicate. Discharge from thickeners is at a dilution of 1 to 1. Four-inch Duplex Diaphragm pumps will be required to handle the underflow of each thickener.

Tailings Disposal.- As the proposed tailings pond site is almost level transportation of these tailings should be by means of a pipe line and a Wilfley pump. Water can be added to tailings at this point if found advisable.

About 60% of the tailings water should be recovered for mill use.

Clarification.- Clarification will be by means of duplicate sand filters. These will be provided with wood screens, cocoa matting and burlap, over which is spread the sand.

An optional method of clarification would be a Butters filter with a wet vacuum pump.

Precipitation.- In order to hold down capital costs duplicate zinc boxes are planned for use in precipitation on zinc shavings.

We have in the present mill a Crowe Vacuum tank for use with zinc dust equipment but the use of this might necessitate the purchase of patent rights for a short-lived operation. Also it is understood that the Crowe patent is due to expire shortly.

Zinc box precipitates can be expressed in sealed cans to a refinery or a melting furnace can be installed and the bars shipped to a mint for parting. The refinery treatment charges are usually very reasonable and this method would save the cost of a furnace installation.

Miscellaneous.- Calcium cyanide will be dissolved in a small Pachuca tank which will then be diluted in a stock solution tank for use in the mill circuit.

Four triplex Diaphragm pumps now available at the plant will have one bowl removed from each as they are over-capacity for our circuit. These four bowls will be sufficient to build up two new duplex pumps with timber frames.

As the plant will be located on almost a level site a Wilfley pump will have to be used to lift the classifier overflow into primary thickener "U".

A number of small pumps now at the plant will be used on the most suitable service, after reconditioning.

Four of the present thickeners with their tanks and superstructures will be used in the reconditioned plant. The original superstructures were weak, but can be suitably reinforced.

The original agitator tanks were so small that only a few hours agitation was possible. These tanks can be utilized in some other service.

It is proposed to use the following old equipment which is already on the property:

1 - Conveyor idlers and belt to deliver the tailings from the slusher hoist to the mill bins

2 - The Mill bins of about 250 tons capacity

3 - Four thickeners erected and in place at the present time

4 - Four #4 Triplex Diaphragm Pumps

5 - Precipitation building

6 - Various pumps, some of which can be utilized to eliminate purchase of new equipment

7 - An assay office partially equipped

8 - The Martinez pump station supplying water to the mill. It is intended to install a small pump and pipe line near Date Creek to augment the water supply in dry seasons

9 - Various wood and steel tanks which may be utilized

10 - A 6' x 4' Ball Mill which is somewhat small for the requirements. It may, however, be utilized.

11 - A quantity of pipe and fittings

12 - A number of motors which may be reconditioned for use

ESTIMATE OF OPERATING POWER REQUIRED:

| | Horse Power Requirements |
|---|---|
| Conveyor to Mill Bins | 7 H.P. |
| Feeders and Feed Conveyor | 2 H.P. |
| Classifier | 10 H.P. |
| Ball Mill | 100 H.P. |
| Wilfley Pump | 10 H.P. |
| 5 Tickeners at 2 H.P. | 10 H.P. |
| Primary solution Pump | 3 H.P. |
| Mill Solution Pump | 5 H.P. |
| Barren Solution Pump | 7 H.P. |
| Wilfley Tailings Pump | 15 H.P. |
| Agitators Mech. H.P. | 5 H.P. |
| Agitators Air Compressor | 10 H.P. |
| Tailings Water Return Pump | 10 H.P. |
| Machine Shop | 5 H.P. |
| Extra | 5 H.P. |
| | 204 H.P. Connected Load |

Assume actual horse-power at 80% of connected load.

Then 204 x 0.80 = 163.2 H.P. actual

163.2 x 0.746 = 122 K.W.

ESTIMATE OF OPERATING COSTS:

In estimating the operating costs of the plant, for the labor estimate the prevailing local rates have been used.

The crew for the plant and their rates per day are as follows:

| | |
|---|---|
| 1 Superintendent | $10.00 |
| 1 Bookkeeper | 5.00 |
| 1 Master Mechanic & engineer | 8.50 |
| 3 Mill Shifters at $5.00 | 15.00 |
| 3 Helpers at $4.00 | 12.00 |
| 3 Laborers at $3.50 | 10.50 |
| 1 Assayer at $5.00 | 5.00 |
| Total | $66.00 |

This gives a labor cost for 300 tons per day of

$\frac{66.00}{300}$ or $0.22 per ton for labor,

and the estimate of operating costs will then be:

| | |
|---|---|
| Cost of placing tailings in mill bins, including operating labor on hoist | $0.12 |
| Labor cost in mill proper | 0.22 |
| Supplies: | |
| KCN 0.6# at 0.16 | 0.096 |
| Zinc 0.4# at 0.13 | 0.052 |
| Lime 21# at 80% efficiency at $11.00 per ton | 0.12 |
| Steel | 0.04 |
| Other chemicals | 0.04 |
| Water | 0.03 |
| Tailings Disposal | 0.02 |
| Power - 14 KWH per ton at 0.007 | 0.098 |
| | 0.836 |

348

71

ESTIMATE OF THE COST OF THE PLANT CHANGES AND ADDITIONS:

| | |
|---|---|
| 1 - Sullivan Slusher Hoist driven by 50-60 H.P. Diesel Motor - complete with scraper and ropes | $ 4,270.00 |
| Freight and installation | 300.00 |
| 1 - Grizzley in place | 150.00 |
| 1 - Screen | 350.00 |
| 1 - Hopper (lined) | 100.00 |
| 1 - Belt conveyor (material on hand) Erection only ( 4 M ft. timber at $50.00) | 200.00 |
| 1 - 300' cross conveyor erected complete | 3,600.00 |
| 2 - Feeders | 60.00 |
| | 9,030.00 |

| | | | |
|---|---|---|---|
| | Forward | $ 9,030.00 | |
| 1 - Conveyor under Bins | | 400.00 | |
| Supports for above, 2 M ft. timber at $60.00 | | 120.00 | |
| 1 - Bowl classifier | | 4,975.00 | |
| Supports for above, 3 M ft. timber erected at $60. | | 180.00 | |
| Motor and drive | | 300.00 | |
| 1 - 7' x 36" Hardinge Ball Mill, delivered and erected | | 4,600.00 | |
| Ball Mill Foundation | | 1,000.00 | |
| 1 - Ball Mill motor and control, erected | | 1,850.00 | 7540.00 |
| Foundation for motor | | 50.00 | |
| Belting for above | | 40.00 | |
| 3 - 36 x 17 Dorr agitators, Wt. 6,000# | 1,500.00 | | |
| Freight | 120.00 | | |
| Superstructure | 300.00 | | |
| Piping | 50.00 | | |
| Tank erected | 1,400.00 | | |
| Foundations | 300.00 | | |
| Timber | 200.00 | | |
| Clearing site | 100.00 | | |
| | 3,970.00 | | |
| | x 3 = | 11,910.00 | |
| 1 - Wilfley Pump and Motor for classifer to primary thickener | | 600.00 | |
| 1 - 33' new Dorr thickener, 8,600# | 1,285.00 | | |
| Freight | 170.00 | | |
| Erection | 100.00 | | |
| Piping erected | 100.00 | | |
| Tank erected | 1,400.00 | | |
| Foundations | 300.00 | | |
| Timber | 200.00 | | |
| Clearing site | 100.00 | 3,655.00 | |
| 2 - New Diaphragm pump frames | | 100.00 | |
| 5 - Thickener gear motor drives complete with belts for pump and mechanism | | 900.00 | |
| | | $ 39,710.00 | |

| | | | |
|---|---|---|---|
| | | Forward | $ 39,710.00 |
| 1 - 20' x 10' Barren solution tank | | | 450.00 |
| 1 - 25 x 12 Mill solution tank | | | 650.00 |
| Clearing site and foundations for above 2 tanks | | | 500.00 |
| 2 - 25 x 8 sand clarifier tanks | 1,200.00 | | |
| Foundations for above | 600.00 | | |
| 8 M ft. timber at $50.00 | 400.00 | | |
| Cocoa matting and burlap | 100.00 | | |
| Sand filling | 50.00 | | 2,350.00 |
| 1 - Wilfley tailings pump and motors | | | 650.00 |
| General Mill piping | | | 1,000.00 |
| Cleaning out all present tankage | | | 500.00 |
| Excavation under present tanks | | | 500.00 |
| Machine shop and tools | | | 3,000.00 |
| 50 H.P. Standby Engine and Generator (An old auto engine will do for this) | | | 750.00 |
| New transmission belting | | | 300.00 |
| Shed over ball mill and classifier | | | 1,000.00 |
| Hoppers and chutes under ore bins | | | 500.00 |
| 1 - 110 Cu.ft. low-pressure compressor complete with 10 H.P. motor and Vpbelt drive delivered | | | 760.00 |
| Foundation and erection of above | | | 150.00 |
| 1 - Grinding water circulating pump from thickener "U" to Mill Solution tank - motor pump for 100 G.P.M. installed | | | 300.00 |
| 3 - 2 H.P. gear motors for present thickeners | | | 500.00 |
| 1 - 5 H.P. gear motor for agitator drive | | | 200.00 |
| Repairing all old motors | | | 400.00 |
| Repairing all old machinery | | | 2,000.00 |
| | | | $ 56,170.00 |

| | | |
|---|---|---|
| | Forward | $ 56,170.00 |
| 1 - | Barren Solution Return Pump, 200 G.P.M. motor pump installed | 225.00 |
| | Erection of 3 new agitators and one new thickener | 800.00 |
| | Timber for walkways, launders, etc. | 1,500.00 |
| 2 - | Zinc boxes at $400.00 | 800.00 |
| | Barren solution sump | 400.00 |
| | Clarified solution sump | 250.00 |
| | Wilfley tailings sump | 100.00 |
| | Tailings Pipe Line | 500.00 |
| | Tailings water pump and return pipe line | 600.00 |
| | New tailings launders | 500.00 |
| 1 - | Cyanide dissolver | 75.00 |
| 1 - | 20' x 10' Stock solution tank, erected in place | 1,700.00 |
| 1 - | Lime feeder | 200.00 |
| 1 - | Lime storage bin | 500.00 |
| 1 | All iron valves and fittings | 1,500.00 |
| | Concrete floor under thickeners and agitators | 1,000.00 |
| 1 - | Tilting Furnace | 500.00 |
| | <u>Miscellaneous</u> | |
| 1 - | Mill water pumping plant and pipe line | 15,000.00 |
| 1 - | 200 H.P. Diesel Engine erected in place | 17,000.00 |
| 1 - | Tailings Pond Fence | 1,500.00 |
| | Repairs to plant fence | 500.00 |
| | | 101,320.00 |

| | | |
|---|---|---|
| | Forward | $ 101,320.00 |
| 1 - Telephone Line | | 500.00 |
| 1 - Power house for Diesel Engine | | 1,000.00 |
| Repairs to houses | | 1,500.00 |
| 1 - Truck | | 1,000.00 |
| 1 - Warehouse and yard | | 1,000.00 |
| Additional assay equipment | | 1,000.00 |
| | | $ 107,320.00 |
| Engineering | | 5,300.00 |
| Contingencies | | 7,380.00 |
| | | $ 120,000.00 |

I am of the opinion that an expenditure of $120,000.00 should be ample to construct complete at Congress a cyanide plant of 300 tons daily capacity as described in the foregoing.

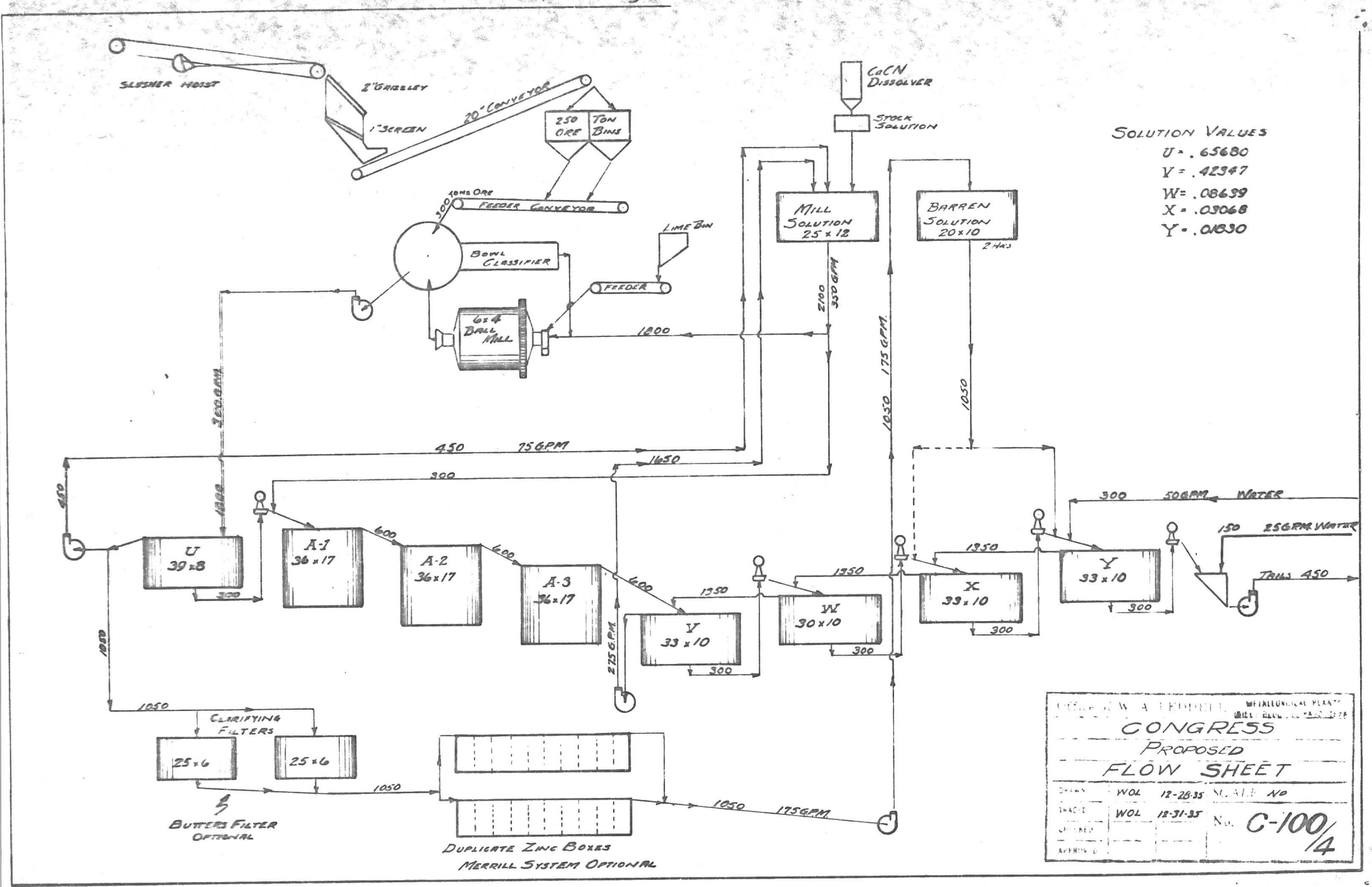
In conclusion, it might be said that equipment vital to good plant operation will be purchased new or will be carefully inspected before purchase if used machinery is selected.

As the details of the plant are taken up carefully in the design of the new plant further economies may be found possible by the use of equipment already on the property.

Respectfully submitted,

W.A. Leddell

W.A. LEDDELL



SLUSHER HOIST
2" GRIZZLEY
1" SCREEN
20" CONVEYOR
250 TON ORE BINS
TONS ORE
300
FEEDER CONVEYOR
BOWL CLASSIFIER
LIME BIN
FEEDER
6x4 BALL MILL
1200
CaCN DISSOLVER
STOCK SOLUTION
MILL SOLUTION 25 x 12
BARREN SOLUTION 20 x 10
2 HRS
2100
350 GPM
175 GPM
1050
SOLUTION VALUES
U = .65680
V = .42347
W = .08639
X = .03068
Y = .01830
450
75 GPM
1650
300
450
1200 GPM
1800
U 39 x 8
A-1 36 x 17
A-2 36 x 17
A-3 36 x 17
600
V 33 x 10
W 30 x 10
X 33 x 10
Y 33 x 10
1350
275 GPM
300
50 GPM
WATER
150
25 GPM WATER
TAILS 450
1050
CLARIFYING FILTERS
25 x 6
25 x 6
BUTTERS FILTER OPTIONAL
DUPLICATE ZINC BOXES
MERRILL SYSTEM OPTIONAL
1050
175 GPM
CONGRESS
PROPOSED
FLOW SHEET
DRAWN WOL 12-28-35
TRACED WOL 12-31-35
SCALE NO
No. C-100/4

WF 597 10M 6-35 HJL PRINTED IN U.S.A.

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Final Report

Cyanidation Tests on Samples of Congress Junction Tailings Submitted by Mr. G. T. Sherman

Introduction:

The following samples designated as "Congress Junction Tailings, Congress Junction, Arizona", were received at the laboratory for metallurgical testing:

An express shipment consisting of four 50 pound sacks and two 5 pint bottles of water. The four sacks containing the samples were marked No. 1, No. 2, No. 3-A, and No. 3-B, respectively. One of the bottles was marked, "Martinez Creek 'A'" and the other "Date Creek 'B'".

Two small samples weighing about 12 pounds each were received by messenger from our New York Office. One of these samples was marked: "Composite 500, Composite of 8 holes" and the other, "Composite #600 Holes Congress Dump - 3-13-17-62-63-64".

The foregoing samples were submitted for cyanidation tests as the results of an arrangement between our Mr. J. T. Sherman and Mr. Gerald T. Sherman.

Description of Samples:

The tailings dump from which these samples were taken was stated to contain two distinct types of material, namely; a roasted and an unroasted material. These products are distinguished from each other by their color and have been classified as "Brownish sands" (roasted) and "White sands" (unroasted).

Samples "No. 2" and "Composite sample 500 composite sample of 8 holes" were classified as brownish sands, whereas, Sample "No. 1" and "Composite #600, holes Congress dump 3-13-17-62-63-64" were designated as white sands.

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Samples Nos. 3-A and 3-B were composites made by mixing Samples 1 and 2.

The tailings pile from which these samples were taken was stated to have been standing in the dump for about 30 years. The samples received for testing purposes were essentially all -10 mesh dry sands. They did, however, contain some soft lumps of the finer material, which were easily broken up by screening.

Purpose of Tests:

We were requested by Mr. Gerald T. Sherman to conduct cyanidation tests on samples No. 1, No. 2, Composite No. 500 and Composite No. 600. No cyanidation tests were run on the composite samples Nos. 3-A and 3-B.

Preliminary Investigation:

Preparation of the Samples for Testing:

Samples No. 1 and No. 2

The samples were treated separately. Each sample was screened thru a 10 mesh sieve to break up the lumps, mixed thoroughly and halved by riffling. One half of each sample was then mixed thoroughly and divided into 600 gram test charges. A head sample of each of the samples was riffled out and submitted for assay. The partial analyses of these head samples are presented in Table 1.

Samples Composite 500 and Composite 600

These two samples were treated separately and each were prepared for testing as follows:

The total amount was screened thru a 10 mesh sieve to break up the lumps, mixed thoroughly by rolling on a rolling cloth. The tailings were then spread out into a flat circular pile. Representative charges of 600 grams each were removed by dipping out small amounts of the tailings with a spatula from

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numerous parts of this pile. A 600 gram charge was ground wet in the laboratory rod mill and a representative amount of this charge constituted the head sample. In case of the composite No. 500 two head samples were cut out, one was pulverized dry in the usual manner and the other was washed to determine the water-soluble values.

The partial analyses of these samples are presented in Table 1.

Table I

Analyses of Head Samples

| Sample No. | Pb % | S % | Orig. Au oz./ton | Duplicates Au oz./ton (a) | (b) | Average Au oz./ton | Ag oz./ton |
|---|---|---|---|---|---|---|---|
| No. 1 | 0.031 | 0.40 | 0.0495 | 0.0685 | - | 0.0590 | 0.500 |
| No. 2 | 0.036 | 0.41 | 0.0535 | 0.0545 | - | 0.0540 | 0.430 |
| No. 3-A | 0.031* | 0.42* | 0.0575 | 0.0635 | - | 0.0605 | 0.450 |
| No. 3-B | | | 0.0600 | - | - | 0.0600 | 0.450 |
| Composite 500 (dry) | | | 0.0453 | 0.0735 | 0.0420 | 0.0536 | 0.441 |
| " 500 washed | | | 0.0459 | 0.0405 | 0.0845 | 0.0569 | 0.441 |
| " 600 | | | 0.0620 | 0.0765 | - | 0.0692 | 0.509 |

* Composite of samples 3-A and 3-B.

It will be observed that the gold values were "spotty" i.e., not uniformly distributed thruout the samples.

The following special method of assaying for gold and silver was used in obtaining the foregoing and all subsequent results:

Two 2-assay ton charges in duplicate, a total of 8 assay tons of each sample, were fused. Two of the charges were inquarted with silver and the other two were not. The lead buttons of the two inquarted charges were combined, scorified, cupelled, parted and assayed in the usual manner for gold. The lead buttons from the other two charges were combined, scorified, cupelled and the doré weighed. This doré was then parted in the usual manner and the gold bead weighed. It will be noted that this method of assaying requires the fusion of at least 8 assay tons of the sample and the weighing of a gold bead representing 4 assay tons. It, therefore, is a means of minimizing the errors introduced by the "spotty" nature of this ore.

Microscopical Examination:

Sample No. 2 (Brownish Sands)

A representative charge of sample No. 2 was ground in the rod mill for 21 minutes and this ground pulp was panned. The panned concentrates were examined with a binocular microscope at 36X. The principal sulfide mineral was pyrite which, in the majority of instances, was tarnished. An occasional particle of what appeared to be fine, free, "rusty" gold, was noted.

Composite Sample #600

A representative sample was panned and the resultant concentrates were examined with a binocular microscope at 36X. Pyrite was the principal sulfide mineral, which, in the majority of instances was tarnished. Some clean bright pyrite was also found, but no free gold was noted.

Analyses of Water Samples:

Partial analyses of the two samples of water are presented in Table II.

Table II

Analyses of Water

| | Martinez Creek "A" | Date Creek "B" |
|---|---|---|
| CaO parts per million | 102.0 | 74.3 |
| MgO " " " | 26.4 | 10.7 |
| pH | 6.8 | 7.4 |
| Remarks | Slight sediment in water. Colorless solution. | Clear and colorless |

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The pH of Warners tap water which was used in conducting the cyanidation tests was 6.6.

We would not anticipate any cyanidation difficulties when using the type of water as represented by either one of these samples. However, if there were a choice, the above analyses would seem to indicate that the Date Creek water might be more suitable than the Martinez Creek water.

Cyanidation Testing Procedure:

The following general testing procedure was used in conducting our cyanidation tests:

The 600 gram charge of -10 mesh tailings sample was ground in a laboratory rod mill at 67 % solids for specified periods of time. Aero Brand cyanide solution, of the same strength as used during subsequent agitation, and dry lime were added to the grinding circuit. The ground pulp was transferred to 5 gallon wide-mouth bottles, which were placed upon revolving wooden rolls. This method of agitation provides excellent aeration of the pulp, because the rotation of the large bottles causes a thin film of solution to be carried around the inside surface of the bottle. After agitation the pulp was filtered and washed twice with tap water. The ratio of water to ore was 2:1 for each wash.

The screen analyses of the various feeds to cyanidation are presented in Table III.

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## Table III

### Screen Analyses of Feeds

| | Brownish Sands | | | | | White Sands | | | |
|---|---|---|---|---|---|---|---|---|---|
| Test Numbers | 1-5-6 & 7 | 10 | 2 | 11 | 3 & 4 | 8 | 12 | 9 | 13 |
| Min. Grinding | 10 | 10 | 16 | 16 | 21 | 12 | 12 | 18 | 18 |
| Mesh Tyler Sieves | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. | % Wt. |
| +100 | 0.53 | 0.40 | Nil | 0.03 | - | 1.33 | 1.66 | - | 0.47 |
| +150 | 8.38 | 7.08 | 1.32 | 0.77 | 0.17 | 14.50 | 18.47 | 2.08 | 6.10 |
| +200 | 14.09 | 12.85 | 7.90 | 5.22 | 1.92 | 14.17 | 15.82 | 10.25 | 14.83 |
| -200 | 77.00 | 79.69 | 90.78 | 93.98 | 97.91 | 70.00 | 64.05 | 87.67 | 78.60 |

**Experimental Data:**

The cyanidation data pertaining to this investigation has been divided into two parts as follows:

Part I - Cyanidation Tests on "Brownish" Sands
Part II - Cyanidation Tests on "White" Sands

### Part I - Cyanidation Tests on Brownish Sands

This data has been divided into two sections as follows:

Section I - Cyanidation Tests on Sample No. 2.

Section II - Cyanidation Tests on Sample "Composite Sample 500 Composite Sample 8 Holes".

### Section I - Cyanidation Tests on Sample No. 2

A series of Tests 1, 2, and 3 was run to show the effect of grinding to -100 mesh, -150 mesh and -200 mesh, respectively. Test No. 4 shows the effect of reducing the time of agitation from 52 hours as used in Test No. 3, to 24 hours in Test No. 4. The detailed data and metallurgical results of these tests are presented in Table IV.

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Table No. IV Test No. 1, 2, 3 & 4

| CONDITIONS AND RESULTS | Test 1 | Test 2 | Test 3 | Test 4 |
|---|---|---|---|---|
| **GRINDING** | | | | |
| Time, minutes | 10 | 16 | 21 | 21 |
| Percent Solids | 67 | 67 | 67 | 67 |
| Barren Solution Added | | | | |
| NaCN, % | 0.112 | 0.112 | 0.112 | 0.112 |
| CaO, % | 0.020 | 0.020 | 0.020 | 0.020 |
| Dry CaO Added, Lbs./ton | 2.0 | 2.0 | 5.0 | 5.0 |
| **AGITATION** | | | | |
| Time, hours | 52 | 52 | 52 | 24 |
| Percent Solids | 33.3 | 33.3 | 33.3 | 33.3 |
| Barren Solution Added | | | | |
| NaCN, % | 0.112 | 0.112 | 0.112 | 0.112 |
| CaO, % | 0.020 | 0.020 | 0.020 | 0.020 |
| Pregnant Solution, Off | | | | |
| NaCN, % | 0.085 | 0.085 | 0.090 | 0.098 |
| CaO, % | 0.054 | 0.054 | 0.044 | 0.031 |
| Au, oz./ton | | | | |
| Ag, oz./ton | | | | |
| **REAGENT CONSUMPTION** | | | | |
| * NaCN, Lbs./ton | 1.05 | 1.14 | 1.12 | 0.63 |
| CaO, Lbs./ton | 15.3 | 14.9 | 15.3 | 11.10 |
| **ASSAYS, Oz./ton** | | | | |
| FEED | | | | |
| Au | 0.054 | 0.054 | 0.054 | 0.054 |
| Ag | 0.43 | 0.43 | 0.43 | 0.43 |
| TAILINGS | | | | |
| Au | 0.0175 | 0.0155 | 0.0145 | 0.0163 |
| Ag | 0.159 | 0.142 | 0.146 | 0.123 |
| **EXTRACTION, %** | | | | |
| Au | 67.59 | 71.30 | 73.15 | 69.81 |
| Ag | 63.02 | 67.59 | 66.05 | 71.40 |

30 - 271



* Low alkalinity during 1st 24 hours agitation likely the cause of high cyanide consumptions.

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The foregoing data show that:

1. The extractions were slightly improved by finer grinding. However, it is doubtful whether grinding finer than 77 % -200 mesh is justified.

2. Increasing the period of agitation from 24 hours to 52 hours improved the gold extractions.

3. The maximum gold extraction did not exceed 74 %.

It has been our experience that occasionally the results of cyanidation can be improved by maintaining a high alkalinity during cyanidation and/or pre-treatment with lime water prior to the addition of cyanide. The next series of three tests was therefore run to show the effect of maintaining a high alkalinity during cyanidation and also to show the effect of aeration with lime before adding the cyanide.

Test No. 5 shows the effect of increasing the amount of lime from 2.0 lbs./ton (in Test No. 1) to 19.4 lbs./ton (Test No. 5) in the grinding circuit.

Test No. 6 shows the effect of diluting the pulp from the rod mill to 3 to 1, then thickening to 1-1/2 to 1 and agitation for 48 hours. In conducting this test the testing procedure was as follows:

The ore was ground in cyanide and saturated lime solution and transferred to a 5 gallon bottle. Enough barren cyanide solution was then added to give a dilution of 3 to 1 after which the pulp was mixed thoroughly and allowed to settle for about 1 hour. The clear effluent solution was then decanted and the thickened pulp agitated for 48 hours.

Test No. 7 shows the effect of aeration with lime prior to the addition of the cyanide. The testing procedure for conducting this test was as follows:

The ore was ground in water containing lime equivalent to 19.4 lbs./ton CaO. This pulp was then transferred to a large conical-bottomed glass tube and diluted with water to a 3 to 1 ratio. Air was bubbled through this pulp, which was maintained practically saturated as regards the lime content, for 2 hours. The pulp was allowed to settle and enough clear effluent solution was decanted to give a thickened pulp of about 1-1/2 to 1. Strong cyanide solution was added and the pulp was then agitated for 48 hours.

The detailed data and metallurgical results of these three tests compared with those of Test I are presented in Table V.

WF 541 1M 4-55 HJL CYANIDATION TEST LOG SHEET



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Table No. V. Test No. 1-5-6 & 7

| CONDITIONS AND RESULTS | Test 1 | Test 5 | Test 6 | Test 7 |
|---|---|---|---|---|
| **GRINDING** | | | | |
| Time, minutes | 10 | 10 | 10 | 10 |
| Percent Solids | 67 | 67 | 67 | 67 |
| Barren Solution Added | | | | |
| NaCN, % | 0.112 | 0.092 | 0.092 | - |
| CaO, % | 0.020 | 0.125 | 0.125 | - |
| Dry CaO Added, Lbs./ton | 2.0 | 19.4 | 19.4 | 19.4 |
| **AGITATION** | | | | |
| Time, hours | 52 | 48 | 48 | 48 |
| Percent Solids | 33.3 | 33.3 | 33.3 | 33.3 |
| Barren Solution Added | | | | |
| NaCN, % | 0.112 | 0.092 | 0.092 | - |
| CaO, % | 0.020 | 0.125 | 0.125 | - |
| Pregnant Solution, Off | | | | |
| NaCN, % | 0.085 | 0.084 | 0.086 | 0.076 |
| CaO, % | 0.054 | 0.095 | 0.096 | 0.093 |
| Au, oz./ton | | | | |
| Ag, oz./ton | | | | |
| Decanted Solution Au oz./ton | | | 0.01005 | 0.00055 |
| **REAGENT CONSUMPTION** | | | | |
| NaCN, Lbs./ton | 1.05 | 0.38 | 0.46 | 0.21 |
| CaO, Lbs./ton | 15.3 | 35.9 | 46.0 | 41.0 |
| **ASSAYS, Oz./ton** | | | | |
| FEED | | | | |
| Au | 0.054 | 0.054 | 0.054 | 0.054 |
| Ag | 0.43 | 0.43 | 0.43 | 0.43 |
| TAILINGS | | | | |
| Au | 0.0175 | 0.0183 | 0.0167 | 0.0185 |
| Ag | 0.159 | 0.167 | 0.120 | 0.153 |
| **EXTRACTION, %** | | | | |
| Au | 67.59 | 66.12 | 69.08 | 65.74 |
| Ag | 63.02 | 61.17 | 72.10 | 64.42 |

30 - 271

The foregoing data show that the gold extractions were not improved by the use of high lime circuit nor by the pretreatment of the pulp with lime prior to the addition of the cyanide. However, the cyanide consumption was appreciably decreased when the pulp was first treated with lime.

The pregnant and wash solutions of Tests 5-6 and 7 were combined and submitted for gold assay, to obtain a more accurate check on the actual gold dissolved in the cyanide solution. The results of these assays are presented in Table VI.

Table VI

Assays of Solutions Tests 5, 6 & 7

| Test No. | Pregnant + Wash Solution | | | | Residues | | **Computed |
|---|---|---|---|---|---|---|---|
| | Dilution Ratio | Assay Au oz./ton | Total Au dissolved oz./ton | % of Total Au | Assays Au oz./ton | % of Total Au | Feed Au oz./ton |
| 5 | 4.817 | 0.00875 | 0.0421 | 69.70 | 0.0183 | 30.30 | 0.0604 |
| * 6 | 1.67 | 0.01005 | 0.0167 | 28.55 | | | |
| 6 | 4.717 | 0.0053 | 0.0250 | 42.74 | 0.0168 | 28.71 | 0.0585 |
| * 7 | 1.67 | 0.00055 | 0.0009 | 1.54 | | | |
| 7 | 4.887 | 0.00805 | 0.0392 | 66.89 | 0.0185 | 31.57 | 0.0586 |

* Decanted solutions.

** Actual average assay was 0.0540 oz./ton.

It will be noted that 28.55 % of the total gold was removed in the pregnant solution obtained by the decantation of the cyanide solution from the grinding circuit. It is estimated from these results that about 40 % of the total gold dissolved in the cyanide solution during the grinding.

Comparing the extractions as reported in Table IV with that actually computed from the assays of the solutions it will be observed that these results check each other. They have been tabulated to show these comparisons as follows:

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| | Test No. 5 | 6 | 7 |
|---|---|---|---|
| Au extractions % (Table V) | 66.12 | 69.08 | 65.74 |
| " " " (Computed from solution assays Table VI) | 69.70 | 71.29 | 68.43 |

Section II - Cyanidation Tests on "Composite Sample 500 Composite Sample 8 Holes"

Two tests were run on this sample of tailings to show the effect of finer grinding. The pregnant and wash solutions were combined and assayed and the feed to cyanidation was computed from the assay of the solutions and residues. The detailed data and metallurgical results of these two tests, Nos. 10 and 11, compared with those obtained when treating the Brownish sands as represented by Sample No. 2, Test No. 5, are presented in Table VII.

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Table No. VII Test No. 5-10 & 11

| CONDITIONS AND RESULTS | Test 5 | Test 10 | Test 11 | | |
|---|---|---|---|---|---|
| GRINDING | | | | | |
| Time, minutes | 10 | 10 | 16 | | |
| Percent Solids | 67 | 67 | 67 | | |
| Barren Solution Added | | | | | |
| NaCN, % | 0.092 | 0.089 | 0.089 | | |
| CaO, % | 0.125 | 0.104 | 0.104 | | |
| Dry CaO Added, Lbs./ton | 19.4 | 18.7 | 18.7 | | |
| AGITATION | | | | | |
| Time, hours | 48 | 46 | 46 | | |
| Percent Solids | 33.3 | 33.3 | 33.3 | | |
| Barren Solution Added | | | | | |
| NaCN, % | 0.092 | 0.089 | 0.089 | | |
| CaO, % | 0.125 | 0.104 | 0.104 | | |
| Pregnant Solution, Off | | | | | |
| NaCN, % | 0.084 | 0.076 | 0.078 | | |
| CaO, % | 0.095 | 0.031 | 0.030 | | |
| Au, oz./ton | | | | | |
| Ag, oz./ton | | | | | |
| REAGENT CONSUMPTION | | | | | |
| NaCN, Lbs./ton | 0.38 | 0.53 | 0.55 | | |
| CaO, Lbs./ton | 35.9 | 20.4 | 20.5 | | |
| ASSAYS, Oz./ton | | | | | |
| FEED | | | | | |
| Au | 0.054 | 0.0516* | 0.0513* | | |
| Ag | 0.43 | 0.441 | 0.441 | | |
| TAILINGS | | | | | |
| Au | 0.0183 | 0.0171 | 0.0163 | | |
| Ag | 0.167 | 0.191 | 0.174 | | |
| EXTRACTION, % | | | | | |
| Au | 66.12 | 66.80 | 68.20 | | |
| Ag | 61.17 | 56.69 | 60.54 | | |

30 - 271



* Computed assay of feed from solution and residue assays. Actual average assay of feed was 0.0536 oz./ton Au.

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## Part II - Cyanidation Tests on "White Sands"

Two series of cyanidation tests were run on samples No. 1 and Composite #600, to show the effect of grinding. Tests numbers 8 and 9 were run on Sample No. 1 and Tests 12 and 13 on the sample Composite #600. The same general testing procedure as described under the heading "Cyanidation Testing Procedure" was used in conducting these tests. The detailed metallurgical results are presented in Table VIII.

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Table No. VIII Test No. 8, 9, 12 & 13

| CONDITIONS AND RESULTS | Test 8 Sample 1 | Test 12 Composite 600 | Test 9 Sample 1 | Test 13 Composite 600 |
|---|---|---|---|---|
| **GRINDING** | | | | |
| Time, minutes | 12 | 12 | 18 | 18 |
| Percent Solids | 67 | 67 | 67 | 67 |
| Barren Solution Added | | | | |
| NaCN, % | 0.094 | 0.091 | 0.094 | 0.091 |
| CaO, % | 0.093 | 0.109 | 0.093 | 0.109 |
| Dry CaO Added, Lbs./ton | 19.4 | 8.85 | 19.4 | 8.85 |
| **AGITATION** | | | | |
| Time, hours | 46 | 49 | 46 | 49 |
| Percent Solids | 33.3 | 33.3 | 33.3 | 33.3 |
| Barren Solution Added | | | | |
| NaCN, % | 0.094 | 0.091 | 0.094 | 0.091 |
| CaO, % | 0.093 | 0.109 | 0.093 | 0.109 |
| Pregnant Solution, Off | | | | |
| NaCN, % | 0.081 | 0.074 | 0.077 | 0.073 |
| CaO, % | 0.073 | 0.057 | 0.077 | 0.034 |
| Au, oz./ton | | | | |
| Ag, oz./ton | | | | |
| **REAGENT CONSUMPTION** | | | | |
| NaCN, Lbs./ton | 0.70 | 0.59 | 0.72 | 0.65 |
| CaO, Lbs./ton | 17.42 | 9.10 | 17.24 | 10.8 |
| **ASSAYS, Oz./ton** | | | | |
| **FEED** | | | | |
| Au | 0.059 | 0.0692 | 0.059 | 0.0692 |
| Ag | 0.500 | 0.509 | 0.500 | 0.509 |
| **TAILINGS** | | | | |
| Au | 0.0127 | 0.0108 | 0.0115 | 0.0099 |
| Ag | 0.059 | 0.083 | 0.078 | 0.077 |
| **EXTRACTION, %** | | | | |
| Au | 78.47 | 84.39 | 80.51 | 85.69 |
| Ag | 88.20 | 83.69 | 84.40 | 84.87 |

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**Summary:**

The results of our cyanidation tests may be briefly summarized as follows:

"Brownish Sands" (Samples No. 2 and Composite 500)

1. The results of a typical cyanide test gave 69.08 % and 70.10 % gold and silver extractions, respectively. The final residues, after regrinding to 77 % -200 mesh and agitating for 48 hours, assayed 0.0167 oz./ton gold and 0.120 oz./ton silver.

2. The additional gold extraction obtained by grinding from 77 % to 98 % -200 mesh does not appear to be sufficient to warrant the finer grinding.

3. The gold and silver extractions were improved by increasing the period of agitation from 24 to 52 hours.

4. The gold and silver extractions were not improved by the use of a high lime circuit during cyanidation.

5. Aeration of the pulp with lime for 2 hours at 3 to 1 dilution, then thickening to 2 to 1, adding cyanide and agitating for 48 hours, did not improve the gold extractions. The cyanide consumption was, however, reduced from 0.46 to 0.21 lb./ton by such a treatment. (See Table V).

"White Sands" Samples No. 1 and Composite #600)

1. Grinding this type of material to 78 % -200 mesh, and cyaniding for 49 hours gave 85.69 % and 84.87 % gold and silver extractions, respectively. The NaCN consumption was 0.65 lb./ton and the lime 10.8 lbs./ton. The final residue assayed 0.0099 and 0.077 oz./ton gold and silver respectively.

**Remarks and Observations:**

1. The head samples of these tailings were "spotty" in gold values and special precautions were used in assaying in order to minimize the errors introduced by such a condition.

(Continued)

AMERICAN CYANAMID COMPANY
ORE DRESSING LABORATORY

(Continued)

2. These tailing samples appeared to be sensitive to the amount of lime added. The use of too small an amount of lime increased the cyanide consumption, even though the solutions had not entirely lost their alkalinity. The consumption of lime appears to be somewhat proportionate to the amount added i.e., the more lime added the higher consumption. The optimum amounts of lime for the Brownish sands appears to be about 20 lbs./ton of CaO and that for the White Sands about 10 lbs./ton.

3. The settling rate of the Brownish Sands and White Sands in cyanide and lime solutions was fairly rapid. No difficulties were encountered in obtaining clear effluent solutions.

AMERICAN CYANAMID COMPANY

Arvid E. Anderson
Arvid E. Anderson
Ore Dressing Laboratory

FJM

September 24, 1935

30 - 271

COLLEGE OF MINES AND METALLURGY
(A BRANCH OF THE UNIVERSITY OF TEXAS)
EL PASO, TEXAS

DEPARTMENT OF MINING AND METALLURGY
JOHN F. GRAHAM
EUGENE M. THOMAS

# REPORT

on

# CYANIDE TESTS

of

# CONGRESS DUMP.

CONGRESS JCT., ARIZ.

made by

JOHN F. GRAHAM

PROF. OF MINING AND METALLURGY,

TEXAS COLLEGE OF MINES AND METALLURGY.

EL PASO, TEXAS JULY 31, 1935

# Cyanide Tests on Congress Dump

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## Preliminary Tests

June 21, 1935 Mr. [illegible] A. Leddell brought in three samples from the Congress Dump upon which were to be made tests preliminary to the receipt of regular dump samples.

| | |
|---|---|
| Sample #1, Composite of 27' hole | Au = .07 oz. |
| Sample #2, White sand | Au = .07 oz. |
| Sample #3, Upper Red Dump | Au = .07 oz. |

## Washing Tests

Washing tests were made on samples #1 and #2. After quartering out a sample, it was divided, one half going to the assayer direct; the other half being well washed before being assayed.

| | |
|---|---|
| Sample #1, Unwashed | Au = .045 oz. |
| Sample #1, Washed | Au = .04 oz. |
| Sample #2, Unwashed | Au = .06 oz. |
| Sample #2, Washed | Au = .08 oz. |

These tests, while not in accord with the head samples, nevertheless do not give any indication that the sands contain any appreciable amount of soluble gold. The fact that the sub-soil under the dump is almost if not as rich as the dump itself would indicate that the dump has been well washed by the rains of the years gone by.

## Flotation

A flotation test was made on sample #2.

| | | | | |
|---|---|---|---|---|
| 12 grams | concentrate | assaying 1.76 oz. Au, | Recovery | = 59.0% |
| 490 " | tails | " .03 " " | " | = 41.0% |

A test made on the final white sand sample gave a tailing value of .02 oz. Ratio of concentration: 25 to 1.

Another test made on the brown sands sample gave a tailing value of .02 oz. Ratio of concentration: 50 to 1.

Inasmuch as better results can be obtained by cyanidation, no further work was done by flotation.

## Cyanidation

A 48 hour agitation test was run of the white sands "as is", without grinding, using .274% KCN solution, and a 2 to 1 solution. This gave a .02 oz. gold tail. No record was made on the silver content.

At this time the two regular samples were received and all work on the preliminary samples was stopped.

---

Cyanide Tests
on
White Sands,
Congress Dump.

---

Sample #100

Assays of alternate quarters of a sample from this ore assayed

| | | |
|---|---|---|
| #100 | Au = .07 | Ag = .45 |
| #100a | Au = .07 | Ag = .49 |

**Sizing Test**

| Material | % of Total | Agitation Hours | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|
| -100, + 150 | 39.2 | 72 | .03 | .008 | .17 | .07 | 73.3 | 59.0 |
| -150, + 200 | 37.2 | 72 | .065 | .006 | .34 | .05 | 90.8 | 85.4 |
| - 200 | 23.6 | 72 | .12 | .005 | .68 | .06 | 95.1 | 91.3 |
| | | | | | | Average = | 89.8 | 83.0 |
| | | using .27% KCN solution, ratio 2 to 1 | | | | | | |
| all -200 | 100 | 72 | .07 | .008 | .42 | .10 | 88.6 | 76.3 |
| | | using .27% KCN solution, ratio 2 to 1 | | | | | | |
| all -200 | 100 | 72 | .07 | .006 | .42 | .10 | 91.4 | 76.3 |
| | | using .11% KCN solution, ratio 2 to 1 | | | | | | |

## Time of Agitation Test

Head Assay: Au = .07  Ag = .41

| Test No. | Grind | Hours Agitated | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|
| 107 | -100 | 24 | .40 | .07 | .005 | .41 | .08 | 93.0 | 83.0 |
| 108 | -100 | 36 | .44 | .07 | .005 | .41 | .08 | 93.0 | 83.0 |
| 109 | -100 | 48 | .52 | .07 | .004 | .41 | .04 | 94.3 | 91.5 |
| 110 | -100 | 60 | .60 | .07 | .004 | .41 | .04 | 94.3 | 91.5 |

Using .27% KCN solution, ratio 2 to 1.

## Sizing Test

Using .28% KCN solution, ratio 2 to 1.

Time of agitation, 45 hours.

| Test No. | Grind | % of Total | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|
| 113 | +35 | 14.4 | .07 | .05 | .45 | .22 | 28.6 | 51.2 |
| 114 | -35+48 | 12.2 | .06 | .03 | .48 | .19 | 50.0 | 60.5 |
| [illegible] | -48+65 | 15.7 | .03 | .02 | .45 | .16 | 33.3 | 64.5 |
| 116 | -65+100 | 15.7 | .03 | .018 | .37 | .12 | 40.0 | 67.6 |
| 117 | -100+150 | 11.6 | .03 | .012 | .27 | .09 | 60.0 | 66.7 |
| 118 | -150+200 | 12.2 | .05 | .008 | .35 | .07 | 85.0 | 80.0 |
| 119 | -200 | 18.1 | .11 | .006 | .79 | .05 | 94.7 | 93.7 |

## Strength of Solution Test

All ground -100 mesh, agitated 48 hours.

Ratio of solution: 2 to 1.

| Test No. | Sol. % KCN | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|
| 120 | .10 | .56 | .07 | .004 | .47 | .04 | 94.3 | 91.5 |
| 121 | .065 | .44 | .07 | .004 | .47 | .05 | 94.3 | 89.3 |
| 122 | .045 | .52 | .07 | .005 | .47 | .05 | 92.8 | 89.3 |
| 123 | .035 | .40 | .07 | .006 | .47 | .05 | 91.5 | 89.3 |
| 124 | .027 | .52 | .07 | .01 | .47 | .07 | 85.7 | 85.1 |

## Summary, White Sand

This ore needs a few more tests to bring out further limits. For instance, while .05% KCN solution is correct for a 48 hour agitation with all ground to -100 mesh, it is very apparent that 48 hours is too long to agitate. Opposite this fact we have that of good dissolution using.27% KCN sol for 24 hours. The determination of the exact relation between time and strength of solution should be determined but was not because it became evident that this ore would give no trouble in cyaniding and I was requested to prepare a composite of the two samples and test that.

---

Tabulation of Cyanide Tests on White Sands

| Test No. | Grind | Sol % KCN | Hours Agitated | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|---|
| 100 | | | | | .07 | | .47 | | | |
| 101 | -100+150 | .27 | 72 | .28 | .03 | .008 | .17 | .07 | 73.3 | 59.0 |
| 102 | -150+200 | .27 | 72 | .48 | .065 | .006 | .34 | .05 | 90.8 | 85.4 |
| 103* | -200 | .27 | 72 | 1.32 | .12 | .005 | .68 | .06 | 95.1 | 91.3 |
| 104 | -200 | .27 | 72 | .84 | .07 | .008 | .42 | .10 | 88.6 | 76.3 |
| 105 | -200 | .11 | 72 | .76 | .07 | .006 | .42 | .10 | 91.4 | 76.3 |
| 107 | -100 | .27 | 24 | .40 | .07 | .005 | .47 | .08 | 93.0 | 83.0 |
| 108 | -100 | .27 | 36 | .44 | .07 | .005 | .47 | .08 | 93.0 | 83.0 |
| 109 | -100 | .27 | 48 | .52 | .07 | .004 | .47 | .04 | 94.3 | 91.5 |
| [illegible] | -100 | .27 | 60 | .60 | .07 | .004 | .47 | .04 | 94.3 | 91.5 |
| 113 | +35 | .28 | 45 | .68 | .07 | .05 | .45 | .22 | 28.6 | 51.2 |
| 114 | -35+48 | .28 | 45 | .60 | .06 | .03 | .48 | .19 | 50.0 | 60.5 |
| 115 | -48+65 | .28 | 45 | .48 | .03 | .02 | .45 | .16 | 33.3 | 64.5 |
| 116 | -65+100 | .28 | 45 | .36 | .03 | .018 | .37 | .12 | 40.0 | 67.6 |
| 117 | -100+150 | .28 | 45 | .40 | .03 | .012 | .27 | .09 | 60.0 | 66.7 |
| 118 | -150+200 | .28 | 45 | 1.16 | .05 | .008 | .35 | .07 | 85.0 | 80.0 |
| 119* | -200 | .28 | 45 | 2.04 | .11 | .006 | .79 | .05 | 94.7 | 93.7 |
| 120 | -100 | .10 | 48 | .56 | .07 | .004 | .47 | .04 | 94.3 | 91.5 |
| 121 | -100 | .065 | 48 | .44 | .07 | .004 | .47 | .05 | 94.3 | 89.3 |
| 122 | -100 | .045 | 48 | .52 | .07 | .005 | .47 | .05 | 92.8 | 89.3 |
| 123 | -100 | .035 | 48 | .40 | .07 | .006 | .47 | .05 | 91.5 | 89.3 |
| 124 | -100 | .027 | 48 | .52 | .07 | .01 | .47 | .07 | 85.7 | 85.1 |

* Tests are -200 products from sizing tests. All other -200 tests are all ground thru 200.

## Cyanide Test on Composite sample, Congress Dump

A sample was prepared by thoroughly mixing 22 parts white sand with 78 parts brown sand.

The sample was divided and opposite quarters assayed.

| | | | |
|---|---|---|---|
| #300 | Composite sample | Au = .06 | Ag = .39 |
| #300a | " " | Au = .06 | Ag = .34 |

| | | | | | |
|---|---|---|---|---|---|
| 78 parts | sample | #200, | assaying | .045 Au = | .0351 |
| 22 " | " | #100, | " | .07 Au = | .0154 |
| | | | | 100 parts = | .0505 = .0505 oz. Au. |

Since the result to be expected from computation is lower than that obtained by sampling, I felt it best to use as the head assay Au = .055 Ag = .37

### Washing

100 grams of the ore was washed and then assayed, assaying .068 oz. gold. Evidently washing removed no gold.

### Screen Analysis of Composite Ore

| | |
|---|---|
| +35 | = 13.0% |
| -35+48 | = 7.9 |
| -48+65 | = 13.0 |
| -65+100 | = 10.7 |
| -100+150 | = 7.5 |
| -150+200 | = 8.3 |
| -200 | = 39.6 |
| | 100.0% |

## Time of Agitation Test

All -100 mesh, agitated with 2 to 1 solution except test #311, which was agitated with 1 to 1 solution. Strength of solution .10% KCN.

| Test No. | Hours Agitated | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|
| 309 | 24 | .055 | .01 | .37 | .13 | 81.8 | 65.0 |
| 310 | 39 | .055 | .008 | .37 | .09 | 85.5 | 75.7 |
| 311 | 39 | .055 | .008 | .37 | .05 | 85.5 | 86.5 |
| 313 | 42 | .055 | .012 | .37 | .14 | 78.2 | 62.0 |
| 312 | 48 | .055 | .006 | .37 | .04 | 89.1 | 89.2 |

## Strength of Solution Test

All -100 mesh, 2 to 1 solution, agitated 42 hours.

| Test No. | % KCN | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|
| 313 | .02 | .055 | .012 | .37 | .14 | 78.2 | 62.0 |
| 314 | .03 | .055 | .01 | .37 | .11 | 81.8 | 70.3 |
| 315 | .037 | .055 | .006 | .37 | .09 | 89.1 | 75.6 |
| 316 | .046 | .055 | .005 | .37 | .09 | 90.9 | 75.6 |
| 317 | .067 | .055 | .004 | .37 | .07 | 92.8 | 81.0 |

## Washing

A further washing test was made on -100 mesh material, using opposite halves of the sample.

| | | |
|---|---|---|
| Unwashed | Au = .07 oz. | Ag = .39 oz. |
| Washed | Au = .065 oz. | Ag = .32 oz. |

## Dissolution in the ball mill

Two samples were ground in solution in the ball mill, and then washed and assayed.

| | | |
|---|---|---|
| #318 | Au = .03 oz. | Ag = .18 oz. |
| #327 | Au = .03 oz. | Ag = .21 oz. |

This would indicate that in grinding

54.5% of the gold dissolved
54.1% " " silver "

## Time of Agitation and Ratio of Dilution Tests

All solution strengths are .10% KCN

| Test No. | | Grind | Dilution | Hours Agitated | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|---|
| 323 | | -100 | 2-1 | 24 | .055 | .01 | .37 | .07 | 81.8 | 81.2 |
| 321 | | -100 | 2-1 | 36 | .055 | .02 | .37 | .08 | 63.7 | 78.5 |
| 322 | | -100 | 1-1 | 36 | .055 | .015 | .37 | .08 | 72.8 | 78.5 |
| 324 | | -150 | 2-1 | 42 | .055 | .02 | .37 | .07 | 63.7 | 81.2 |
| 325 | | -150 | 1-1 | 42 | .055 | .005 | .37 | .05 | 90.9 | 86.5 |
| 326 | | -150 | 2-1 | 48 | .055 | .005 | .37 | .08 | 90.9 | 78.5 |
| 333 | 82% | -200 | 2-1 | 48 | .055 | .008 | .37 | .11 | 85.5 | 70.3 |
| 331 | 91% | -200 | 2-1 | 48 | .055 | .006 | .37 | .10 | 89.1 | 73.0 |

This indicates that the time of agitation should be 48 hours, that the sands should be ground to -150 mesh, that a dilution ratio of 1 to 1 is sufficient, and that the cyanide consumption will be .6 lb.

The gold extraction will be close to 90%, the silver about 75%.

It is well to note here that gold assays given in the third decimal place are only estimates and it is foolish to figure gold extractions too closely when based upon such results. A reading of Au = .008 means that it was somewhere between .005 oz. and .01 oz. and that .008 was the best judgement of the weigher at that time.

### Lime Test

Tests were made using various amounts of lime per ton of ore.

The pH measure of the ore gave an alkalinity of greater than 8 when 7 is taken as neutral and lower than 7 as acid. Despite this fact the ore requires considerable lime to establish a definite alkalinity of solution. Presumably the lime is consumed by absorption on the surface of the ore.

| | | |
|---|---|---|
| With 5# CaO per ton of ore | % CaO in the solution | = .003 |
| " 10# " " " " " | " " " " " | = .013 |
| " 15# " " " " " | " " " " " | = .025 |
| " 20# " " " " " | " " " " " | = .045 |

Since it is advisable to always have a lime content of .05% in the solution, the above test indicates that 20# of lime per ton of ore is required. It is true that the lime will build up in the solution and so the amount introduced after operating conditions become settled can be decreased. Two tests were successfully made using 10# CaO, but five tests were lost because 10# didn't give sufficient alkalinity. So all tests were made using 20# CaO.

**Note:**

This series was begun upon receipt of a telegram of July 5th giving the proportions to be used, and was completed July 15th.

## Tabulation of Cyanide Tests

| Test No. | Grind | Solution % KCN | Hours Agitated | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|---|
| 300 | | | | | .06 | | .37 | | | |
| 301 | +48 | .10 | 46 | .12 | .07 | .03 | .39 | .27 | 57.2 | 30.8 |
| 302 | -48+65 | .10 | 46 | .28 | .05 | .02 | .31 | .22 | 60.0 | 29.0 |
| 303 | -65+100 | .10 | 46 | .24 | .04 | .018 | .28 | .20 | 55.0 | 28.5 |
| 304 | -100+150 | .10 | 46 | .24 | .03 | .016 | .21 | .16 | 46.7 | 23.8 |
| 305 | -150+200 | .10 | 46 | .52 | .06 | .014 | .34 | .15 | 73.3 | 55.9 |
| 306 | -200 | .10 | 38 | .88 | .062 | .01 | .60 | .29 | 83.9 | 51.7 |
| 309 | -100 | .10 | 24 | .36 | .055 | .01 | .37 | .13 | 81.8 | 65.0 |
| 310 | -100 | .10 | 36 | .32 | .055 | .008 | .37 | .09 | 85.5 | 75.7 |
| 312 | -100 | .10 | 48 | .36 | .055 | .006 | .37 | .04 | 89.1 | 89.2 |
| 313 | -100 | .02 | 42 | .32 | .055 | .012 | .37 | .14 | 78.2 | 62.0 |
| 314 | -100 | .03 | 42 | .28 | .055 | .01 | .37 | .11 | 81.8 | 70.3 |
| 315 | -100 | .037 | 42 | .36 | .055 | .006 | .37 | .09 | 89.1 | 75.6 |
| [illegible] | -100 | .046 | 42 | .36 | .055 | .005 | .37 | .09 | 90.9 | 75.6 |
| 317 | -100 | .067 | 42 | .28 | .055 | .004 | .37 | .07 | 92.8 | 81.0 |
| 318 | +100 | .10 | .1 | 3.80 | .055 | .05 | .33 | .18 | 9.0 | 45.0 |
| 321 | -100 | .10 | 36 | .52 | .055 | .02 | .37 | .08 | 63.7 | 78.5 |
| 323 | -150 | .10 | 24 | .48 | .055 | .01 | .37 | .07 | 81.8 | 81.2 |
| 324 | -150 | .095 | 42 | .40 | .055 | .02 | .37 | .07 | 63.7 | 81.2 |
| 326 | -150 | .095 | 48 | .64 | .055 | .005 | .37 | .08 | 90.9 | 78.5 |
| 333 | 4 min. | .095 | 48 | .54 | .055 | .008 | .37 | .11 | 85.5 | 70.3 |
| 331 | 6 min. | .095 | 48 | .63 | .055 | .006 | .37 | .10 | 89.1 | 73.0 |

All above test agitated with a 2 to 1 solution ratio.

---

All tests below agitated with a 1 to 1 solution ratio.

| | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| 322 | -100 | .10 | 36 | .52 | .055 | .015 | .37 | .08 | 72.8 | 78.5 |
| 325 | -150 | .10 | 42 | .64 | .055 | .005 | .37 | .05 | 90.9 | 86.5 |
| 311 | -100 | .10 | 39 | .84 | .055 | .008 | .37 | .05 | 85.5 | 86.5 |

#300 series of tests on composite #1

## Cyanide Test on Brown Sands

July 12th instructions were received to continue the tests on the brown sands only, using a 1½ to 1 ratio of dilution, grinding to -100 mesh and -150 mesh and report results.

In handling this sample a change was made in the procedure of the individual test. In all previous work the ore was ground dry to pass the desired screen and then bottle agitated. This method is liable to give an excess of fines and of course is not the method used in the mill. It is standard practice in testing and is comparatively easy to handle.

From this time on all samples were ground in solution, washed with solution as in the classifier, decanted to the desired thickness as in the thickener, and then agitated in bottles the required time. Cyanide and lime were measured before grinding, after decantation, and after agitation. This method requires considerably more work but approaches closely mill conditions.

By this method no longer could screens be used to control the fineness but the rod mill used for grinding had to be calibrated from the standpoint of time. Following is the quality of grinding referred to when mention is made of"minutes ofgrinding".

| | 2 min. | 2½ min. | 3 min. | 4 min. | 5 min. | 6 min. |
|---|---|---|---|---|---|---|
| +100 | 9 | 3½ | 2 | 1 | 1 | ½ |
| +150 | 18 | 15 | 9 | 3 | ½ | ½ |
| +200 | 14 | 19½ | 20 | 18 | 11½ | 8 |
| -200 | 59 | 62 | 69 | 78 | 87 | 91 |
| | 100 | 100 | 100 | 100 | 100 | 100 |
| Ave. of | 4 tests | 4 tests | 6 tests | 5 tests | 1 test | 1 test |

Three samples were cut and assays made on the original sack of brown ore.

| | | |
|---|---|---|
| #200 | Au = .05 | Ag = .39 |
| #200a | Au = .04 | Ag = .34 |
| #200b | Au = .05 | |

Sizing Test
+100 mesh = 45%
+150 " = 11%
+200 " = 7
-200 " = 37
100

The head value used in this series of tests was

Au = .045 Ag = .37

### Preliminary test

While testing the White Sand, I had started a sizing test on this ore and also what I call an "ideal" test. When orders came to stop testing I did finish the tests I had started on the white sand and also the "ideal" test on the brown sand, but discarded all the others. The "ideal" test was ground all -200 mesh, agitated for 72 hours with .11% KCN solution, 2 to 1 ratio.

| Test No. | Gold | | Silver | | Recovery | |
|---|---|---|---|---|---|---|
| | Heads | Tails | Heads | Tails | Au | Ag |
| 206 | .045 | .003 | .37 | .03 | 93.4 | 91.8 |

A complete tabulation of all tests is given on a later page but some conclusions may be brought out here.

### Time of Agitation

In no combination tried did the tailing value drop below .01 oz. gold until agitation had taken place for 48 hours. Therefor 48 hours will have to be the agitation period.

Checks [illegible]

### Fineness of Grinding

In all cases tried of different fineness of grinding the tailing value dropped below .01 oz. gold. The fineness of grinding finally adopted will be selected because of economic reasons. A 5% increase in gold extraction will realize 8¢ additional. Whether such a gain in extraction is an actual gain financially must be decided.

### Fineness of Grinding Tests

Agitated 48 hours in .1% KCN solution, 1½ to 1 ratio of solution.

| Test No. | Grind | Gold | | Silver | | Recovery | |
|---|---|---|---|---|---|---|---|
| | | Heads | Tails | Heads | Tails | Au | Ag |
| 221 | 2 min. | .045 | .009 | .37 | .12 | 80.1 | 67.6 |
| 215 | 2½ " | .045 | .007 | .37 | .11 | 84.5 | 70.3 |
| 225 | 3 " | .045 | .007 | .37 | .12 | 84.5 | 67.6 |
| 226 | 4 " | .045 | .005 | .37 | .12 | 88.9 | 67.6 |

####

At this time I undertook to make a final series of tests checking the above results. From certain evidence I had obtained I believed that the tests could be run with 10# CaO and come through with protective alkalinity, and I so ran them. The resultant alkalinity was so near nothing that the tests were a failure giving exceedingly poor extraction. I was now out of ore and Mr. Sherman expressed me 25# reaching me July 24th. I put on 7 tests July 25th which came off July 27th and the results are given below.

Two assays of this new ore gave

| | | |
|---|---|---|
| #1 | Au = .04 | Ag = .41 |
| #2 | Au = .05 | Ag = .63 |
| Average | .045 | .52 |

This ore proved not to be at all like the previous brown ore but I did not discover this until the tests were completed.

First, the new ore has an exceedingly low, by comparison with the first brown ore, settling rate.

Second, the new ore has a high cyanide consumption, being from .8# to 1.25# per ton of ore.

Third, the ore is much finer than the first ore. A screen analysis gave:

| | |
|---|---|
| +100 = | 36% |
| +150 = | 9 |
| +200 = | 5 |
| -200 = | 50 |
| | 100 |

Four minute grinding produced:

| | |
|---|---|
| +100 = | 1% |
| +150 = | 1 |
| +200 = | 10 |
| -200 = | 88 |
| | 100 |

Fourth, 20# CaO was not enough for this ore. The end alkalinity at the finish of the agitation was visible but nowhere near enough for safety.

Fifth, The ore carried pyrite, the first I have encountered in these tests.

I feel, however, that this series of tests does show that with the same ore as previously handled the tail value would have been below .01 oz. gold since most of these tests were .01 oz.

It becomes a question of how much of this class of ore you will have.

Page 12

Seven tests were run and the results are tabulated below.

All tests were ground 4 minutes and agitated 48 hours.

| Test No. | Sol ratio | Sol % KCN | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|
| 237 | 1½ to 1 | .098 | 1.7 | .045 | .01 | .52 | .15 | 77.8 | 71.1 |
| 238 | 1½ to 1 | .098 | 1.14 | .045 | .005 | .52 | .13 | 89.0 | 75.0 |
| 239 | 1½ to 1 | .098 | .78 | .045 | .01 | .52 | .14 | 77.8 | 73.0 |
| 240 | 1 to 1 | .098 | 1.08 | .045 | .01 | .52 | .15 | 77.8 | 71.1 |
| 241 | 2 to 1 | .098 | .80 | .045 | .015 | .52 | .16 | 66.5 | 69.2 |
| 242 | 1½ to 1 | .054 | .48 | .045 | .01 | .52 | .15 | 77.8 | 71.1 |
| 243 | 1½ to 1 | .070 | .60 | .045 | .014 | .52 | .15 | 68.7 | 71.1 |

Cyanide Tests on Original Brown Ore

Congress Dump

| Test No. | Grind | Solution % KCN | Hours Agitated | lbs. KCN used | Gold Heads | Gold Tails | Silver Heads | Silver Tails | Recovery Au | Recovery Ag |
|---|---|---|---|---|---|---|---|---|---|---|
| 200 | | | | | .05 | | .39 | | | |
| 200a | | | | | .04 | | .34 | | | |
| 200b | | | | | .05 | | | | | |
| 206 | -200 | .27 | 72 | | .045 | .003 | .37 | .03 | 93.4 | 91.8 |
| 210 | 3 min. | .095 | 14 | .42 | .045 | .03 | .37 | .18 | 33.3 | 59.3 |
| 218 | " | .095 | 24 | .54 | .045 | .01 | .37 | .14 | 77.9 | 62.1 |
| 211 | " | .095 | 30 | .42 | .045 | .03 | .37 | .10 | 33.3 | 73.0 |
| 217 | " | .095 | 36 | .63 | .045 | .01 | .37 | .14 | 77.9 | 62.1 |
| 223 | " | .095 | 36 | .45 | .045 | .03 | .37 | .17 | 33.3 | 54.2 |
| 225 | " | .095 | 48 | .54 | .045 | .007 | .37 | .12 | 84.5 | 67.6 |
| 220 | 2 min. | .095 | 24 | .33 | .045 | | .37 | | | |
| 219 | " | .095 | 30 | .54 | .045 | | .37 | | | |
| 222 | " | .095 | 36 | .30 | .045 | .025 | .37 | .17 | 44.5 | 54.2 |
| 221 | " | .095 | 48 | .72 | .045 | .009 | .37 | .12 | 80.1 | 67.6 |
| 212 | 2½ min. | .095 | 24 | .60 | .045 | .015 | .37 | .06 | 66.7 | 83.8 |
| 213 | " | .095 | 30 | .48 | .045 | .01 | .37 | .07 | 77.9 | 81.0 |
| 214 | " | .095 | 36 | .63 | .045 | .01 | .37 | .13 | 77.9 | 64.9 |
| 215 | " | .095 | 48 | .63 | .045 | .007 | .37 | .11 | 84.5 | 70.3 |
| 224 | 4 min. | .095 | 36 | .45 | .045 | .02 | .37 | .17 | 55.6 | 54.1 |
| 226 | " | .095 | 48 | .63 | .045 | .005 | .37 | .12 | 88.9 | 67.6 |

Aug. 5, 1935

Mr. Gerald Sherman,
New York City, N. Y.

Dear Mr. Sherman:

<u>Re Congress Dump</u>

Variations in procedure between American Cyanamid Co. and Graham:

1. Graham adds lime all at beginning and Am. Cyanamid by stages.

   Total consumption of CaO is about the same in each case, 15#, because Graham has a residual value of lime which will stay in the circuit.

   I have no explanation as to why a difference in place or time of adding lime should make any difference except that I find that when my protective alkalinity at the end of the grinding period was below .05% per ton of solution, extraction went to pieces.

2. Graham agitates a 1½ to 1 solution, the Am. Cyanamid a 2 to 1 solution, the difference being all in favor of the Am. Cyanamid.

3. The Am. Cyanamid agitates in open mouth bottles, Graham in closed bottles.

   The greater aeration of the open mouth bottle is in favor of the Am. Cyanamid Co.

   The greater, more violent agitation of the end over end agitation is in favor of Graham.

Other than the above all conditions are apparently alike in the two tests except - and I would not suggest it were we not trying to find some reason where apparently no reason exists -

| | |
|---|---|
| Am. Cyanamid | Heads = .054 oz. Au, .43 oz. Ag |
| Graham | Heads = .045 oz. Au, .37 oz. Ag |

The above discrepancy ought to be all on the side of the Am. Cyanamid Co. in figuring extraction.

Since it is not, then it is possible that the New York assayer is consistently high in all results, both heads and tails, or the El Paso assayer consistently low.

The El Paso assayer feels that his work in general is being constantly checked by the umpire work he does for the smelters.

How does any assayer turn out a gold result in the fourth decimal place? .0175 gold. I have always doubted the accuracy of the third place, except as a guess.

Saturday morning I put on three samples of composite of three sacks received from Congress on Thursday. They came off today and were assayed and results turned in to-night.

| | | | | | |
|---|---|---|---|---|---|
| Test | #401 | Au = | .015 | Ag = | .17 |
| " | #402 | " | .015 | " | .16 |
| " | #403 | " | .014 | " | .15 |

The above results are disappointing, but this sample also was decidedly different from the first brown ore, in (1) lime consumption using 40# (2) in KCN consumption, using about 1# (3) in settling rate, being considerably slower, although still classed as a quick settling ore.

In my report, every forty-eight hour test on Original Brown Ore gave less than ~~xx~~ .01 oz. gold.

| | | |
|---|---|---|
| #221 | 2 min. grind | .009 oz gold tailing |
| #215 | 2½ min. grind | .007 " " " |
| #225 | 3 " " | .007 " " " |
| #226 | 4 " " | .005 " " " |

My only trouble on this ore came when I tried to cut down on the lime.

In the morning I shall put on a couple more bottles of the latest brown ore, a comple of bottles of a composite of the latest brown ore and white sand, and a couple of repeats of composite #300. These will come off Thursday morning and be assayed by Thursday night.

Repeats on composite #300 will check my former work, although 12 tests on this composite showed .01 oz. of gold or less.

Tests on a new composite will show whether beneficient effects may be ~~xxxx~~ due to white sands.

Repeat on the latest brown ore will simply check the former work, listed at the head of this letter.

Regardless of the American Cyanamid results, there can be no doubt that my samples #100, #200, and composite #300 consistently gave low tails with 48 hour agitation. Not once did they fail. I might easily get a tailing too high but it would be hard to always get too low.

It is to be regretted that my origonal brown ore ore sample was consumed before the test was finished, but on the other hand, if the dump has refractory spots you should know it. Have you no idea where the origenal brown ore came from?

John F. Graham

# CONGRESS MILL TAILINGS

## SUMMARY OF CYANIDE MILL TESTS

White Tailings

Sample #1

**Graham**

Gold 0.07 ozs. per ton Silver 0.47 ozs. per ton

**Sizing Test**

| Test | Size Mesh | Percentage of Weight | Gold | Silver |
|---|---|---|---|---|
| 113 | +35 | 14.4% | 0.07 ozs. | 0.45 ozs. |
| 114 | -35+48 | 12.2% | 0.06 | 0.48 |
| 115 | -48+65 | 15.7% | 0.03 | 0.45 |
| 116 | -65+100 | 15.7% | 0.03 | 0.37 |
| 117 | -100+150 | 11.6% | 0.03 | 0.27 |
| 118 | -150+200 | 12.2% | 0.05 | 0.35 |
| 119 | -200 | 18.1% | 0.11 | 0.79 |

**Tests on Complete Sample**

| Test | Size | % K C N Solution | Hours Agitation | Cyan. Consumption | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|
| 120 | -100 | 0.10 | 48 | 0.56# | 0.07 ozs | 0.004 ozs | 94.3% | 91.5% |
| 121 | | 0.065 | 48 | 0.44# | | 0.004 | 94.3 | 89.3 |
| 122 | | 0.045 | 48 | 0.52# | | 0.005 | 92.8 | 89.3 |
| 123 | | 0.035 | 48 | 0.40# | | 0.006 | 91.5 | 89.3 |
| 124 | | 0.027 | 48 | 0.52# | | 0.010 | 85.7 | 85.1 |

American Cyanamid Company

Gold 0.059 ozs. per ton. Silver 0.50 ozs. per ton.

Ratio of sizes presumably according to Graham's sizing test above.

Grinding Test

| Test #8 | | Test #9 | |
|---|---|---|---|
| 12 Minutes | | 18 Minutes | |
| +100 Mesh | 1.33% | +100 Mesh | 0.0 |
| +150 " | 14.5 % | +150 " | 2.08% |
| +200 " | 14.17% | +200 " | 10.25% |
| -200 " | 70.0 % | -200 " | 87.67% |

Results of grinding should be consistent as between other tests by American Cyanamid on White Tailings, but have no relation to Graham's data on time and fineness of grinding.

| Test | Size | Hours Agitation | Cyanide Consumpt. | Lime Consumpt. | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|
| #8 | As above | 46 | 0.7# | 17.4# | 0.059 ozs | 0.0127 | 78.47% | 88.2% |
| #9 | As above | 46 | 0.72# | 14.2# | 0.059 | 0.0115 | 80.51% | 84.4% |

Using head assay of 0.07 ozs. Gold per ton, as in Graham's sample, and same tails the Gold Extraction would be: -

#8 81.9 %

#9 83.6 %

Lime charged 20# per ton

NaCN 0.094%

Dilution 2 : 1

## Sample #2



**Graham**

Gold 0.083 ozs. per ton. Silver 0.50 ozs. per ton.

Fineness of grinding

| Tests | Grind | +100 Mesh | +150 | +200 | -200 |
|---|---|---|---|---|---|
| #601) 602) 603) | 5 min. | 3% | 5% | 24% | 68% |
| #604 | 7 min. | 2% | 0½% | 10% | 87½% |
| #605 | 9 min. | 4% | 0 | 1½% | 94½% |

| Test | Grind | Hours Agitation | Heads | Tails | Gold Extraction | Silver Extraction |
|---|---|---|---|---|---|---|
| #601 | 5 min. | 48 | 0.083 | 0.008 | 90.4% | 94.0% |
| 602 | 5 min. | 48 | | 0.008 | 90.4% | 94.0% |
| 603 | 5 min. | 64 | | 0.01 | 88.0% | 94.0% |
| 604 | 7 min. | 48 | | 0.012 | 85.5% | 94.0% |
| 605 | 9 min. | 48 | | 0.009 | 89.1% | 96.0% |

**American Cyanamid Company** (Harder Material)

Gold 0.0692 ozs. Silver 0.509 ozs.

| Test | Grind | +100 Mesh | +150 | +200 | -200 | |
|---|---|---|---|---|---|---|
| #12 | 12 min. | 0.66% | 18.47% | 15.82 | 64.05% | (?) 99. |
| #13 | 18 min. | 0.47% | 6.10 | 14.83 | 78.60% | |

| Size | Hours Agitation | Cyanide Consumption | Lime Consumption | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|
| #12 12 min. | 49 | 0.59# | 9.1# | 0.0692 oz | 0.0108 | 84.39% | 83.69% |
| #13 18 min. | 49 | 0.65# | 10.8# | 0.0692 | 0.0099 | 85.69% | 84.87% |

With same tails &) (0.07 ozs.
or) (0.08 ozs. Heads the extraction would be:

| | Heads | | Extraction |
|---|---|---|---|
| #12 | 0.07<br>0.08 | Extraction | 84.6%<br>86.5% |
| #13 | 0.07<br>0.08 | Extraction | 85.8%<br>87.6% |

Gold 0.08 ozs. per ton Silver 0.50 ozs.

| Test | Size | Hours Agitation | Cyanide Consumpt. | Lime Consumpt. | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|
| #1 | 85% -200 M. | 48 | 0.75# | 10# (?) | 0.08 ozs | 0.01 | 86.0% | 77.5% |
| #2 | 85% -200 M. | 48 | 0.53# | | 0.08 | 0.01 | 87.5% | 76.0% |

#1 Agitated in closed bottle

#2 " in open bottle

Difference in extraction apparently due to unaccounted for loss in Gold. Did tailings weigh out exactly 0.01 ozs. Gold?

| | | | |
|---|---|---|---|
| #1 Test | Gold Heads | | 0.08 ozs. |
| | Solution | 0.06 ozs. | |
| | Tails | 0.01 | 0.07 |

Were Tails 0.02 ozs. or Solution 0.07 ozs. or 3 split between them, or were Heads 0.07 ozs.

| | | | |
|---|---|---|---|
| #2 Test | Gold Heads | | 0.08 ozs. |
| | Solution | 0.0695 ozs. | |
| | Tails | 0.01 | 0.0795 |

In the above tests on White Tailings there is a wide range in the Head assays which indicate the effect of particles of fine Gold.

Sample #1

| | HEADS Gold | HEADS Silver | TAILS Gold |
|---|---|---|---|
| Graham | 0.07 ozs. | 0.47 ozs. | 0.004 ozs.<br>.004<br>.005<br>.006<br>.010 |
| American Cyenamid | 0.059 ozs. | 0.50 ozs. | 0.0127 ozs.<br>0.0115 |

Sample #2

| | HEADS Gold | HEADS Silver | TAILS Gold |
|---|---|---|---|
| Graham | 0.083 ozs. | 0.50 ozs. | 0.008 ozs.<br>0.008<br>0.01<br>0.012<br>0.009 |
| American Cyanamid | (0.062 ozs.<br>(0.0765 | 0.509 ozs. | 0.0108<br>0.0099 |
| R. A. Perez | 0.08 ozs. | 0.50 | 0.02<br>0.0105 |

In Perez Tails, it is assumed that unaccounted for Gold appears in the Tails.

## Brown Tailings (Roasted)

Sample #1

Graham

Gold 0.045 ozs. per ton Silver 0.37 ozs.

Grinding Test

| | 2 Min. | 2½ Min. | 3 Min. | 4 Min. | 5 Min. | 6 Min. |
|---|---|---|---|---|---|---|
| +100 | 9% | 3½% | 2% | 1% | 1% | ½% |
| +150 | 18% | 15 % | 9% | 3% | ½% | ½% |
| +200 | 14% | 19½% | 20% | 18% | 11½% | 8 % |
| -200 | 59% | 62% | 69% | 78% | 87% | 91 % |

Preliminary "Ideal" Test. All through 200 Mesh.

Agitated 72 hours 0.11% KCN Ratio 2 : 1

| Test | GOLD Heads | GOLD Tails | SILVER Heads | SILVER Tails | EXTRACTION Gold | EXTRACTION Silver |
|---|---|---|---|---|---|---|
| #206 | 0.045 ozs | 0.003 ozs | 0.37 ozs | 0.03 ozs | 93.4% | 91.8% |

General Test

| Test | Size | K C N Solution | Hours Agitation | Gold Heads | Gold Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|
| #221 | 2 Min. Grind | 0.10% | 48 | 0.045 | 0.009 | 80.1% | 67.6% |
| #215 | 2½ | 0.10 | 48 | .045 | 0.007 | 84.5 | 70.3 |
| #225 | 3 | 0.10 | 48 | .045 | 0.007 | 84.5 | 67.6 |
| #226 | 4 | 0.10 | 48 | .045 | 0.005 | 88.9 | 67.6 |

American Cyanamid Company

Gold 0.054 ozs. Silver 0.43 ozs.

Grinding Test

10 Minutes

| Tests | +100 | +150 | +200 | -200 |
|---|---|---|---|---|
| #1-5-6-7 | 0.53% | 8.38% | 14.09% | 77.0% |
| #10 | 0.40% | 7.08% | 12.85% | 79.69% |

16 Minutes

| Tests | +100 | +150 | +200 | -200 |
|---|---|---|---|---|
| #2 | 0 | 1.32% | 7.90% | 90.78% |
| #11 | 0.03% | 0.77 | 5.22 | 93.98% |

21 Minutes

| | | | | |
|---|---|---|---|---|
| #11 | 0 | 0.17% | 1.92% | 97.91% |

Cyanide Tests

| Test | Size Grind | K C N Solution | Hours Agitation | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|
| #1 | 10 Min. | 0.112% | 52 | 0.054 | 0.0175 | 67.6% | 63.0% |
| #2 | 16 | 0.112 | 52 | .054 | 0.0155 | 71.3 | 67.6 |
| | | 0.112 | 52 | .054 | 0.0145 | 73.15 | 66.05 |

By interpolation it appears that by grinding to 80% -200 Mesh an extraction of 68.2% would be obtained.

Lime in solution 0.02% dry 2# per ton.

Further tests were made on Sample #1 with changes as follows: -

#5 19.4# of lime per ton used fed into mill

#6 Reagents as in #5 Test, but tailings were decanted, recovering 0.0167 ozs. Gold. Solution then brought up to strength getting 0.025 ozs. more.

#7 Tailings ground with lime without cyanide and agitated (2 hours?). Some of lime solution decanted, cyanide added and agitated. Lime solution contained 0.0009 ounces of Gold in water soluble form.

Fineness - Ground 10 Minutes 77% -200 ?

Mesh Agitation 52 hours

| Test | Size Ground * | Hours Agitation | NaCN Consumpt. | Lime Consumpt. | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|
| #5 | 10 Min. | 52 | 0.38# | 35.9 | 0.059 | 0.0183 | 66.12 | 61.17% |
| #6 | 10 Min. | 52 | 0.46# | 46.- | 0.059 | 0.0167 | 69.08 | 72.1 % |
| #7 | 10 Min. | 52 | 0.21# | 41.- | 0.059 | 0.0185 | 65.74 | 64.4 % |

* 77% Minus 200 Mesh

It may be inferred from this test that lime charged in the mill in large quantities reduces cyanide consumption but has little effect on the extraction.

## Sample #2

This sample contained tailings from fewer drill holes, and a considerable proportion came from near the bottom of the pile adjacent to the original granite soil.

Graham

Gold 0.045 ozs. Silver 0.52 ozs.

| Test | Size Ground * | Hours Agitation | Solution Ratio | K C N Solution | Cyanide Consumed | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|---|
| #237 | 4 Min. | 48 | 1½ : 1 | 0.098% | 1.7# | 0.045 | 0.01 | 77.8% | 71.1% |
| 238 | 4 | 48 | 1½ : 1 | | 1.14# | | 0.005 | 89.0% | 75.0 |
| 239 | 4 | | 1½ : 1 | | 0.78# | | 0.01 | 77.8% | 73.0 |
| 240 | 4 | | 1 : 1 | | 1.08 | | 0.01 | 77.8% | 71.1 |
| 241 | 4 | | 2 : 1 | 0.098 | 0.80 | | 0.015 | 66.5% | 69.2 |
| 242 | 4 | | 1½ : 1 | 0.054 | 0.48 | | 0.01 | 77.8% | 71.1 |
| 243 | 4 | 48 | 1½ : 1 | 0.070 | 0.60 | 0.045 | 0.014 | 68.7% | 71.1% |

* 10 Minutes grinding represents approximately: -

| | |
|---|---|
| +100 Mesh | 0.5% |
| +150 Mesh | 7.75% |
| +200 Mesh | 13.5 % |
| -200 Mesh | 77.8 % |

See grinding in Tests #1 - 5 - 6 - 7 and 10

## Sample #3

### Graham

Gold 0.05 ozs. per ton      Silver 0.40 ozs. per ton

| Test | SIZE +100 | +150 | +200 | -200 | Hours Agitation | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|---|
| #503 | 2% | 2 | 13% | 83% | 48 | 0.05 | 0.015 | 70% | 67.5% |
| 504 | 3% | 2 | 17 | 78% | 48 | | 0.017 | 66% | 62.5% |
| 505 | 3% | 2 | 9 | 86% | 60 | | 0.017 | 66% | |
| 506 | 3% | 0 | 4 | 93% | 48 | | 0.015 | 70% | 65.-% |
| 507 | 1% | 1 | 1 | 97% | 48 | 0.05 | 0.014 | 72% | 65.-% |

| | | | | | |
|---|---|---|---|---|---|
| Grinding in Minutes | 4 | 4 | 4 | 6 | 8 |
| Sample | #503 | #504 | #505 | #506 | #507 |

### American Cyanamid Company

Gold 0.05145 ozs. per ton      Silver 0.441 ozs. per ton

| Test | MESH +100 | +150 | +200 | -200 | Hours Agitation | Cyanide Consumpt. | Lime Consumpt. | Heads | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|---|---|---|
| #10 | 0.4% | 7.1% | 12.8% | 79.7% | 46 | 0.53# | 20.4# | 0.0516 | 0.0171 | 66.8% | 56.7% |
| #[illegible] | 0.03 | 0.77 | 5.22 | 94.0 | 46 | 0.55# | 20.5# | 0.0513 | 0.0163 | 68.2% | 60.5% |

### R. A. Perez & Company

Gold 0.045 ozs. per ton      Silver 0.40 ozs. per ton

Sample ground to 85% Minus 200 Mesh

Agitation 48 hours

| Test | Cyanide Solution | Lime Per Ton | K C N Consumpt. | Lime Consumpt. | Heads | Solution | Tails | Gold Extract. | Silver Extract. |
|---|---|---|---|---|---|---|---|---|---|
| #1 | 0.112% | 10# | 0.5# | 9.5# | 0.045 | 0.031 | 0.01 | 75.6% | 55.8% |
| 2 | 0.112 | 10# | 0.5# | 9.5# | | 0.022 | 0.02 | 52.4% | 31.8% |
| 3 | 0.112 | 20# per ton Sol. | 0.68# | | | 0.035 | 0.01 | 77.7% | 60.4% |
| 4 | 0.105% | 20# per ton Sol. | 1.0# | 15.0# | | 0.035 | 0.01 | 77.7% | 59.3% |
| 5 | 0.105% | 20# per ton Sol. | 1.0# | | 0.045 | 0.0375 | 0.01 | 78.9% | 52.8% |

| | Agitation | Lime | Solution R |
|---|---|---|---|
| #1 | Closed Bottle ) | 10# per ton ore | 2 : 1 |
| #2 | Open " ) | | 2 : 1 |
| #3 | Closed Bottle ) | 20# per ton | 1½: 1 |
| #4 | Air Agitation ) | Solution | 2 : 1 |
| #5 | ) | | ½: 1 |

There are no notes on fineness of grinding except that 85% passed a 200 mesh screen. In Test #5 it is noted that the sample was ground for 30 minutes.

The time of grinding is only comparative for the White and Brown Tailings at the same laboratory.

Perez & Company apparently required 30 minutes to make 85% pass 200 mesh. Graham's 5 minute grind put 87% through the same screen.

October 22, 1935

Congress
Mill
Operating Reports
June 1938 to Dec 1940

| DUMP NO. | FLOTATION | | | | | | | | CYANIDE | | | | | REMARKS |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | COMPOSITE HEADS | | CONCENTRATE | | TAILS | | RATIO OF CONCENTRATION | PER CENT RECOVERY | COMPOSITE HEADS | | TAILS | | PER CENT RECOVERY | |
| | OZ | $ | OZ | $ | OZ | $ | | | OZ | $ | OZ | $ | | |
| 1 | .115 | 4.03 | 2.25 | 78.75 | .043 | 1.51 | 31 TO 1 | 62.6 | .115 | 4.03 | .038 | 1.33 | 67.0 | |
| 2 | .099 | 3.47 | 2.10 | 73.50 | .027 | 0.94 | 29 TO 1 | 72.7 | .099 | 3.47 | .028 | 0.98 | 71.8 | |
| 3 | .120 | 4.20 | 1.75 | 61.25 | .027 | 0.94 | 19 TO 1 | 77.4 | .120 | 4.20 | .035 | 1.25 | 70.7 | |
| 4 | .100 | 3.50 | 2.25 | 78.75 | .027 | 0.94 | 38 TO 1 | 73.0 | .100 | 3.50 | .033 | 1.15 | 67.0 | |
| 5 | .110 | 3.85 | 2.50 | 87.50 | .025 | 0.87 | 30 TO 1 | 77.3 | .110 | 3.85 | .022 | 0.77 | 80.0 | |
| AVE | .109 | 3.81 | 2.185 | 76.47 | .030 | 1.05 | 29.5 TO 1 | 72.6 | .109 | 3.81 | .031 | 1.10 | 71.3 | |

# MILL-R ORT

## JUNE - 1938

| Day | tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN 48% | Zn. # | Lead Nitrate | K.W.H. | Gal. Fuel | Lub. Oil | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 160 | .06 | | .04 | | 1000 | 3 | | 100 | | 2900 | 225 | 2 | |
| 2 | 155 | .06 | | .01 | | 1000 | 16 | | | | 2700 | 225 | 2 | Bldg up ball load |
| 3 | 160 | .06 | | .04 | | 1000 | 24 | | | | 2700 | 225 | 2 | |
| 4 | 165 | .12 | | .01 | | 600 | 16 | | | | 2900 | 225 | 2 | |
| 5 | 170 | .12 | | .02 | | 300 | 8 | USED 330 lbs. per day CaCn | | Approx. 1 lb. used per day | 3100 | 225 | 2 | |
| 6 | 175 | .12 | | .01 | | -- | 8 | | | | 2400 | 225 | 2 | |
| 7 | 180 | .10 | | .02 | | 300 | 8 | | 100 | | 2700 | 225 | 2 | |
| 8 | 180 | .10 | | .01 | | 300 | 12 | | | | 2800 | 225 | 2 | |
| 9 | 180 | .12 | | .02 | | 300 | 12 | | | | 2900 | 225 | 2 | |
| 10 | 190 | .10 | | .06 | | 300 | 12 | | | | 2900 | 255 | 2 | Down 2 hrs chg class. |
| 11 | 170 | .08 | | .06 | | 300 | 8 | | | | 2700 | 223 | 2 | |
| 12 | 200 | .10 | | .08 | | 300 | 12 | | | | 3000 | 249 | 2 | |
| 13 | 220 | .14 | | .08 | | 300 | 12 | | 100 | | 3000 | 275 | 1½ | |
| 14 | 190 | .08 | | .06 | | 300 | 12 | | | | 3000 | 297 | 1½ | |
| 15 | 180 | .08 | | .06 | | 400 | 16 | | | | 2500 | 255 | 1½ | |
| | 2675 | 1.44 | | .58 | | 6700 | 154 | | 300 | | 42200 | 3579 | 28½ | |

# MILL REPORT

## JUNE - 1938

| Day | Tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN# 48% | Zn.# | Lead Nitrate | K.W.H. | Gal. Fuel | Lub. Oil | Remarks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| ~~16~~ | | | | | | | | | | | | | | |
| 17 | | | | | | | 12 | | | 400 | 100 | 1½ | | Mill down |
| 18 | 120 | | | | | | 12 | | | 1800 | 174 | 1 | | 36 hours |
| 19 | 180 | .08 | | .02 | | 400 | 16 | | | 2800 | 265 | 2 | | |
| 20 | 230 | .10 | | .01 | | 400 | 16 | | | 2900 | 273 | 1½ | | |
| 21 | 240 | .10 | | .01 | | 400 | 12 | | | 2900 | 272 | 2 | | |
| 22 | 245 | | | | | 400 | 12 | | | 2900 | 276 | 2 | | Assay Office burned |
| 23 | 245 | | | | | 400 | 9 | 100 | | 2880 | 270 | 1½ | | |
| 24 | 245 | .06 | | .01 | | 400 | 16 | | | 2820 | 255 | 1½ | | |
| 25 | 245 | .10 | | .02 | | 400 | 20 | | | 2780 | 261 | 1½ | | |
| 26 | 245 | .06 | | .01 | | 400 | 16 | | | 2800 | 259 | 1½ | | Stopped 15 min. New scoop tip. |
| 27 | 175 | .08 | | .01 | | 400 | 16 | | | 2800 | 264.4 | 1½ | | White Sands |
| 28 | 175 | .10 | | .01 | | 400 | 14 | | | 2700 | 258 | 1½ | | Wilfley sand pump down. Feed off 20min. |
| 29 | 175 | .10 | | .02 | | 400 | 14 | | | 2850 | 267 | 1½ | | |
| 30 | 175 | .08 | | .01 | | 400 | 14 | 100 | | 2500 | 262 | 1½ | | |
| | 175 | .08 | | .01 | | 400 | 16 | | | 2750 | 257 | 1½ | | |
| Total | 2870 | .94 | | .14 | | 5200 | 215 | 200 | | 38880 | 3713 | 23½ | | |
| 1-15 | 2675 | 1.44 | | .58 | | 6700 | 184 | 300 | | 42200 | 3579 | 28½ | | |
| G Tot. | 5545 | 2.38 | | .72 | | 11900 | 399 | 500 | | 81080 | 7292 | 52 | | |
| Aver. | 198 | .091 | | .028 | | | | | | 2703 | | | | |

## ILL RE[illegible]

### JULY - 1938

| Day | Tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN# 48% | Zn. # | Lead Nitrate | KWH | Gals. Fuel | Ursa Lub. Oil | Remarks | Running Time Hrs |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 210 | .07 | | .01 | | 400 | 7 | 400 | 18 | | 2800 | 261 | 1½ | | 24 |
| 2 | 220 | .08 | | .01 | | 400 | 8 | 200 | 18 | | 2700 | 264 | 1½ | | 22-2 |
| 3 | 230 | .08 | | .01 | | 400 | 9 | 400 | 18 | | 2800 | 264 | 1½ | | 24 |
| 4 | 245 | .10 | | .01 | | 400 | 11 | 400 | 18 | | 2900 | 265 | 2 | | 24 |
| 5 | 252 | .08 | | .04 | | 500 | 12 | 400 | 18 | | 2750 | 265 | 1½ | | 24 |
| 6 | 230 | .08 | | .01 | | 500 | 10 | 800 | 18 | | 2750 | 266 | 1½ | | 24 |
| 7 | 224 | .08 | | .01 | | 700 | 12 | 800 | 18 | | 2700 | 264 | 1½ | | 24 |
| 8 | 215 | .06 | | .01 | | 800 | 16 | 800 | 18 | | 2800 | 263 | 1½ | | 24 |
| 9 | 235 | .06 | | .01 | | 500 | 17 | | 18 | | 2800 | 270 | 1½ | | 24 |
| 10 | 230 | .06 | | .01 | | 500 | 16 | 1600 | 18 | | 2800 | 264 | 1½ | | 24 |
| 11 | 195 | .08 | | .01 | | 500 | 15 | | 18 | | 2760 | 264 | 1½ | | 24 |
| 12 | 180 | .06 | | .01 | | 500 | 18 | 800 | 18 | | 2740 | 261 | 1½ | | 24 |
| 13 | 90 | .08 | | .01 | | | 17 | | 18 | | 900 | 139 | 1½ | No Water | 13.5 |
| 14 | 50 | .06 | | .01 | | | 16 | | 12 | | 1900 | 192 | 1 | " " | 8.25 |
| 15 | 140 | .06 | | .01 | | 720 | 14 | | 18 | | 1700 | 199 | 2 | " " | 21.25 |
| | 2946 | 1.09 | | .18 | | 6820 | 198 | 6600 | 264 | | 37800 | 3698 | 23 | | 329-2/3 |

# MILL [illegible] RT

## JULY - 1938

| Day | Tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN# 48% | Zn.# | Lead Nitrate | KWH | Gals. Fuel | Lub Oil | Remarks | Running Time Hours |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 16 | 0 | 0 | | 0 | | 0 | 0 | 0 | 0 | | 500 | [illegible] | 1 | NoWater | |
| 17 | 208 | .10 | | .01 | | 500 | 16 | 800 | 12 | | 2700 | 250 | 1½ | | 23-2/3 |
| 18 | 227 | .08 | | .01 | | 500 | 8 | 200 | 14 | | 2700 | 257 | 1½ | | 24 |
| 19 | 216 | .10 | | .01 | | 500 | 12 | 800 | 12 | | 2800 | 263 | 1½ | | 24 |
| 20 | 240 | .08 | | .01 | | 500 | 16 | 600 | 12 | | 2700 | 260 | 1½ | | 24 |
| 21 | 218 | .10 | | .01 | | 500 | 16 | | 9 | | 2800 | 264 | 1½ | | 24 |
| 22 | 218 | .12 | | .02 | | 500 | 16 | | 9 | | 2700 | 267 | 1½ | | 24 |
| 23 | 228 | .12 | | .02 | | 500 | 16 | | 9 | | 2800 | 272 | 1½ | | 24 |
| 24 | 228 | .12 | | .02 | | 500 | 7 | 800 | 9 | | 2800 | 271 | 2 | | 24 |
| 25 | 233 | .08 | | .02 | | 500 | 15 | | 9 | | 2750 | 264 | 1½ | | 24 |
| 26 | 237 | .10 | | .04 | | 500 | 11 | | 9 | | 2790 | 266 | 1½ | X TankShaft Twisted | 24 |
| 27 | 166 | .10 | | .02 | | 750 | 12 | 800 | 12 | | 1210 | 220 | 1½ | HotMotor | 17½ |
| 28 | 144 | .12 | | .04 | | 250 | 16 | | 6 | | 2000 | 219 | 1½ | Triplex Pump | 16-3/4 |
| 29 | 211 | .14 | | .04 | | 400 | 16 | | 18 | | 2800 | 265 | 1½ | smelting | 24 |
| 30 | 211 | .10 | | .06 | | 150 | 12 | 800 | 18 | | 2800 | 259 | 1½ | | 24 |
| 31 | 213 | .12 | | .04 | | 200 | 12 | | 18 | | 2800 | 265 | 1½ | | 24 |
| | 3198 | 1.58 | | .37 | | 6750 | 201 | 4800 | 176 | | 39450 | 3944 | 24 | | 345.91 |
| | 2946 | 1.09 | | .18 | | 6820 | 198 | 6600 | 264 | | 37800 | 3698 | 23 | | 329.66 |
| | 6144 | 2.67 | | .55 | | 13570 | 399 | 11400 | 440 | | 77250 | 7642 | 47 | | 675.57 |
| | Av.198 | Av. .086 | | Av.018 | | Av. 438# | Av. 12.9 | Av.367# | Av.12.2# | | Av.2492 | Av.246.5 | Av.15 | | Av. 90.8% |

# M I L L  R E [illegible] T

## AUGUST - 1938

| Day | Tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN# 48% | Zn. # | Lead Nitrate | KWH | Gals. Fuel | Lub Oil | Running Hours Remarks | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 215 | .12 | | .02 | | 600 | 8.0 | 1000 | 18 | | 2800 | 267 | 2 | | 24 |
| 2 | 215 | .10 | | .02 | | — | 12.0 | 800 | 18 | | 2800 | 268 | 1½ | | 24 |
| 3 | 220 | .10 | | .02 | | 300 | 7.25 | — | 18 | | 2700 | 260 | 1½ | | 24 |
| 4 | 217 | .12 | | .04 | | 500 | 15.0 | — | 18 | | 2700 | 260 | 1½ | | 24 |
| 5 | 219 | .12 | | .04 | | 500 | 11.5 | 800 | 18 | | 2700 | 254 | 2 | | 24 |
| 6 | 225 | .14 | | .06 | | 500 | 13.0 | 200 | 18 | | 2700 | 266 | 1½ | | 24 |
| 7 | 238 | .14 | | .04 | | 500 | 11.5 | — | 18 | | 2700 | 260 | 2 | | 24 |
| 8 | 222 | .14 | | .02 | | 500 | 15.5 | 1000 | 18 | | 2800 | 266 | 1½ | Shovel moved new place | 24 |
| 9 | 205 | .10 | | .04 | | 500 | 11.75 | 800 | 18 | | 2800 | 265 | 2 | | 24 |
| 10 | 206 | .08 | | .02 | | 500 | 12.0 | — | 18 | | 2700 | 261 | 1½ | #1 Agitator in operation | 24 |
| 11 | 205 | .08 | | .01 | | 500 | 16.0 | 800 | 18 | | 2700 | 262 | 2 | Shovel moved place (old) | 24 |
| 12 | 229 | .15 | | .06 | | 1500 | 12.0 | — | 18 | | 2800 | 257 | 2 | | 24 |
| 13 | 234 | .185 | | .06 | | 1500 | 8.0 | 800 | 18 | | 2800 | 264 | 2 | | 24 |
| 14 | 232 | .145 | | .07 | | 500 | 12.0 | — | 12 | | 2700 | 259 | 2 | Press off 6AM On 12.30PM | 24 |
| 15 | 234 | .17 | | .06 | | 500 | 12.0 | 1200 | 18 | | 2900 | 266 | 1½ | Smelt 6:30AM Complete 8:30P | 24 |
| | T- 3316 | 1.89 | | .58 | | 8900 | 177.5 | 7400 | 264 | | 41300 | 3935 | 26½ | | 360 |

MILL REPORT

AUGUST - 1938

| Day | Tons | Heads Au. | Heads Ag. | Tails Au. | Tails Ag. | Balls | Pump Hours | KCN# 48% | Zn.# | Lead Nitrate | KWH | Gals. Fuel | Lub. Oil | Remarks | Running Hrs. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 16 | 234 | .19 | | .07 | | 500 | 16.0 | — | 18 | | 2800 | 263 | 2 | | 24 |
| 17 | 235 | .19 | | .075 | | 500 | 12.0 | 800 | 18 | | 2800 | 260 | 1½ | raised NACN to 1.50 | 24 |
| 18 | 235 | .18 | | .09 | | 500 | 12.0 | 200 | 18 | | 2700 | 255 | 2 | cleaned NACN tank | 24 |
| 19 | 208 | .14 | | .085 | | 500 | 10.0 | 1000 | 18 | | 2800 | 256 | 1½ | | 24 |
| 20 | 205 | .16 | | .08 | | 500 | 10.0 | — | 18 | | 2800 | 266 | 1¾ | | 24 |
| 21 | 206 | .18 | | .07 | | 500 | 12.0 | 200 | 18 | | 2740 | 256 | 1¾ | | 24 |
| 22 | 206 | .17 | | .08 | | 500 | 12.0 | — | 18 | | 2760 | 254 | 1¾ | | 24 |
| 23 | 207 | .165 | | .07 | | 500 | 12.0 | 800 | 18 | | 2700 | 256 | 2 | | 24 |
| 24 | 146 | .185 | | .08 | | 500 | 10.0 | 1000 | 12 | | 1900 | 176 | 1¾ | Ball Mill Motor down | 17 |
| 25 | 76 | .075 | | .13 | | 500 | 12.0 | 800 | 6 | | 2050 | 197 | 1½ | " | 9 |
| 26 | 204 | .125 | | .07 | | 500 | 10.0 | — | 18 | | 2800 | 258 | 1¾ | | 24 |
| 27 | 205 | .085 | | .015 | | 500 | 10.0 | — | 18 | | 2820 | 256 | 1¾ | | 24 |
| 28 | 205 | .08 | | .045 | | 500 | 12.0 | 800 | 18 | | 2700 | 257 | 1¾ | | 24 |
| 29 | 205 | .125 | | .07 | | 500 | [illegible] | — | 18 | | 2850 | 257 | 1¾ | Mine Pump Started | 24 |
| 30 | 207 | .10 | | .07 | | 500 | 8.0 | 800 | 18 | | 3000 | 270 | 1¾ | Mine Pump ran 12 hrs | 24 |
| 31 | 206 | .125 | | .065 | | 500 | 2.0 | — | 18 | | 2800 | 256 | 1¾ | Press down 9:45 AM On 1:45 PM | 24 |
| Total | 3190 | 2.33 | | 1.11 | | 8000 | 170.0 | 5400 | 270 | | 43050 | 3993 | 28 | | 362 |
| Total 1-15 | 3316 | 1.89 | | .58 | | 8900 | 177.5 | 7400 | 264 | | 41300 | 3935 | 26½ | | 360 |
| G Total | 6506 | 4.22 | | 1.69 | | 16900 | 347.5 | 12800 | 534 | | 84350 | 7928 | 54½ | | 722 |
| Aver. | 209.87 | .13611 | | .05129 | | | | | | | | | | | |

January 1939

| | |
|---|---|
| Average Heads | .0588 |
| Average Tails | .02 |
| Indicated Recovery | 65.9% |
| Total Tons Milled | 10288 |
| Tons Per Day | 331 |
| Percentage of Full Running Time | 98.6 |
| Pounds CaCN Per Ton | 1.147 |
| Pounds Zn. per ton | .078 |
| K. W. H. per Gal. Fuel | 11.85 |
| Pounds Lime Per Ton | 5.07 |
| Pounds Balls per Ton of Ore | 2.3 |
| Average Tons Per Hour | 14 |
| K. W. H. Per Ton | 10.43 |
| Ball Cost Per Ton of Ore | .0427 |
| Cyanide Cost Per Ton of Ore | .0659 |

## MILL DATA FOR FEBRUARY

February 1939.

| | |
|---|---|
| Average Heads | .0573 |
| Average Tails | .0182 |
| Recovery | 68.2% |
| Tons Milled | 8466.0 |
| Tons Per Day | 302.0 |
| % Full Running Time | 97.1 |
| Pounds CaCN Per Ton | 1.209 |
| Pounds Zinc Per Ton | 0.106 |
| Pounds Lime Per Ton | 5.83 |
| Pounds Balls Per Ton | 2.15 |
| Average Tons Per Hour | 12.9 |
| K.W.H. Per Ton of Ore | 11.23 |
| K.W.H. Per Gal. Fuel Oil | 11.97 |
| Balls Cost Per Ton of Ore | .0411¢ |
| Cyanide Cost Per Ton of Ore | .1018¢ |

MILL DATA FOR MARCH

March 1939.

| | |
|---|---|
| Average Heads | .0738 |
| Average Tails | .024 |
| Recovery | 67.4% |
| Tons Milled | 9628 |
| Tons Per Day | 310.6 |
| % Full Running Time | 98.5% |
| Pounds CaCN Per Ton | 1.39 |
| Pounds Zinc Per Ton | .165 |
| Pounds Lime Per Ton | 7.62 |
| Pounds Balls Per Ton | 2.14 |
| Average Tons Per Hour | 13.14 |
| K.W.H. Per Ton of Ore | 11.17 |
| K.W.H. Per Gal. Fuel Oil | 11.81 |
| Balls Cost Per Ton of Ore | .0409 |
| Cyanide Cost Per Ton of Ore | .0897 |

# MILL DATA FOR APRIL 1939

| | |
|---|---|
| Average Heads | .0749 |
| Average Tails | .0246 |
| Recovery | 67% |
| Tons Milled | 8741 |
| Tons Per Day | 291.3 |
| % Full Running Time | 91.[illegible] |
| Pounds CaCN Per Ton | 1.[illegible] |
| Pounds Zinc Per Ton | .185 |
| Pounds Lime Per Ton | 5.[illegible] |
| Pounds Balls Per Ton | 2.1[illegible] |
| Average Tons per hour | 13.23 |
| K. W. H. Per Gal. Fuel | 11.65 |
| K. W. H. Per Ton of Ore | 11.2[illegible] |
| Ball Cost Per Ton | .0355 |
| CaCN Cost Per Ton | .0916 |

# MILL DATA FOR MAY 1939

| | |
|---|---|
| Average Heads | .0668 |
| Average Tails | .0216 |
| Recovery | 67.6% |
| Tons Milled | 9650 |
| Tons Per Day | 311 |
| % Full Running Time | 99.25% |
| Pounds CaCN Per Ton | .99 |
| Pounds Zn Per Ton | .164 |
| Pounds Lime Per Ton | 7.51 |
| Pounds Balls Per Ton | 1.87 |
| Average Tons Per Hour | 13.1 |
| K. W. H. Per Gallon Fuel | 11.78 |
| K. W. H. Per Ton of Ore | 10.94 |
| Ball Cost Per Ton of Ore | .0357 |
| CaCN Cost Per Ton of Ore | .0639 |

## MILL DATA FOR JUNE 1939

| | |
|---|---|
| Average Heads | .0611 |
| Average Tails | .0198 |
| Recovery | 67.6% |
| Tons Milled | 9392 |
| Tons Per Day | 313 |
| % Full Running Time | 95.7% |
| Pounds CaCN Per Ton | 1.13 |
| Pounds Zinc Per Ton | .155 |
| Pounds Lime Per Ton | 5.27 |
| Pounds Balls Per Ton | 2.06 |
| Average Tons Per Hour | 13.63 |
| K. W. H. Per Gallon Fuel | 11.56 |
| K. W. H. Per Ton of Ore | 11.12 |
| Ball Cost Per Ton of Ore | .0393 |
| CaCN Cost Per Ton of Ore | .0731 |

## MILL DATA FOR JULY 1939

| | | |
|---|---|---|
| Average Heads | .0599 | .0599 |
| Average Tails | .0203 | .0203 |
| Recovery | 66.11% | 66.11% |
| Tons Milled | 8826 | 10949 |
| Tons Per Day | 285 | 353 |
| % Full Running Time | 95.61% | 95.61% |
| Pounds CaCN Per Ton | 1.42 | 1.14 |
| Pounds Zinc Per Ton | .179 | .144 |
| Pounds Lime Per Ton | 6.61 | 5.33 |
| Pounds Balls Per Ton | 2.16 | 1.74 |
| Average Tons Per Hour | 12.4 | 15.38 |
| K. W. H. Per Gallon Fuel | 11.28 | 11.28 |
| K. W. H. Per Ton of Ore | 12.31 | 9.92 |
| Ball Cost Per Ton | .04126 | .0332 |
| CaCN Cost Per Ton | .0916 | .0735 |

Please note two columns, figures in one column are based on a tonnage of 10949 tons, the other column is the tonnage arrived at from the bullion returns devided by the indicated recovery. Neither column is correct. The high tonnage reported was due to a change in the motor speed driving the feeder belt and was also due to a higher percentage of moisture due to recent rains.

# MILL DATA FOR AUGUST, 1939

| | |
|---|---|
| Average Heads | .09673 |
| Average Tails | .01986 |
| Recovery | 79.54% |
| Tons Milled | 7664 |
| Tons Per Day | 247 |
| % Mill Running Time | 72.53% |
| Pounds CaCN Per Ton | 1.531 |
| Pounds Zinc Per Ton | .1609 |
| Pounds Lime Per Ton | 5.458 |
| Pounds Balls Per Ton | 2.008 |
| Average Tons Per Hour | 14.2 |
| K. W. H. Per Gallon Fuel | 10.55 |
| K. W. H. Per Ton of Ore | 11.32 |
| Ball Cost Per Ton of Ore | .03015 |
| CaCN Cost Per Ton of Ore | .11936 |

# MILL DATA FOR SEPTEMBER, 1939

| | |
|---|---|
| Average Heads | .07304 |
| Average Tails | .0212 |
| Recovery | 70.97% |
| Tons Milled | 6434 |
| Tons Per Day | 214.5 |
| Full Running Time % | 70.07% |
| 4400 lbs. CaCN & 2912 lbs. NaCN per ton | 1.1364 |
| Pounds Zinc Per Ton | .1417 |
| Pounds Lime Per Ton | 6.321 |
| Pounds Balls Per Ton | 1.9599 |
| Average Tons Per Hour | 12.75 |
| K.W.H. Per Gallon Fuel | 10.40 |
| K.W.H. Per Ton of Ore | 13.59 |
| Ball Cost Per Ton of Ore | .03743 |
| CaCN & NaCN cost per ton of ore | .10165 |

# MILL DATA FOR OCTOBER 1939

| | |
|---|---|
| Average Heads | .06826 |
| Average Tails | .0253 |
| Recovery | 62.9% |
| Tons Milled | 7312 |
| Tons Per Day | 235.80 |
| Average Tons Per Hour | 10.61 |
| % Full Running Time | 92.6% |
| Lbs. NaCN Per Ton | .7352 |
| Lbs. Zinc Per Ton | .1362 |
| Lbs. Lime Per Ton | 6.5477 |
| Lbs. Balls Per Ton | 2.4507 |
| K. W. H. Per Gallon Fuel | 10.427 |
| K. W. H. Per Ton of Ore | 15.06 |
| Ball Cost Per Ton of Ore | .0468 |
| NaCN Cost Per Ton of Ore | .0937 |

## MILL DATA FOR NOVEMBER 1939

| | |
|---|---|
| Average Heads | .0631 |
| Average Tails | .0284 |
| Recovery | 54.99% |
| Tons Milled | 9107 |
| Tons Per Day | 303.57 |
| Tons Per Hour | 12.65 |
| % Full Running Time | 92.11% |
| Lbs. NaCN Per Ton | .9255 |
| Lbs. Zinc Per Ton | .1068 |
| Lbs. Lime Pe$ Ton | 5.498 |
| Lbs. Balls Per Ton | 1.852 |
| K.W.H. Per Gallon Fuel | 11.21 |
| K.W.H. Per Ton of Ore | 12.00 |
| Ball Cost Per Ton of Ore | .03537 |
| NaCN Cost Per Ton of Ore | .1147 |

## MILL DATA FOR DECEMBER 1939

| | |
|---|---|
| Average Heads | .1022 |
| Average Tails | .0253 |
| Recovery | 75.2% |
| Tons Milled | 7300. |
| Tons Per Day | 235.5 |
| Tons Per Hour | 9.82 |
| % Full Running Time | 94.5% |
| Lbs. NaCN Per Ton | .9580 |
| Lbs. Zinc Per Ton | .0645 |
| Lbs. Lime Per Ton | 6.996 |
| Lbs. Balls Per Ton | 2.656 |
| K.W.H. Per Gallon Fuel | 12.09 |
| K.W.H. Per Ton of Ore | 16.77 |
| NaCN Cost Per Ton of Ore | .1139 |
| Ball Cost Per Ton of Ore | .0507 |

## MILL DATA FOR JANUARY 1940

| | |
|---|---|
| Average Heads | .06965 |
| Average Tails | .02072 |
| Recovery | 70.25% |
| Tons Milled | 8032 |
| Tons Per Day | 259.09 |
| Tons Per Hour | 10.80 |
| % Full Running Time | 86.36% |
| Lbs. NaCN Per Ton | .8963 |
| Lbs. Zinc Per Ton | .1855 |
| Lbs. Lime Per Ton | 5.701 |
| Lbs. Balls Per Ton | 2.65 |
| K. W. H. Per Gallon Fuel | 12.047 |
| K. W. H. Per Ton of Ore | 15.034 |
| NaCN Cost Per Ton of Ore | .1145 |
| Ball Cost Per Ton of Ore | .0506 |

## MILL DATA FOR FEBRUARY 1940

| | |
|---|---|
| Average Heads | .06937 |
| Average Tails | .0192 |
| Recovery | 72.32% |
| Tons Milled | 7600. |
| Tons Per Day | 262. |
| Tons Per Hour | 10.91 |
| % Full Running Time | 98.31% |
| Lbs. NaCN Per Ton | .92105 |
| Lbs. Zinc Per Ton | .1942 |
| Lbs. Lime Per Ton | 5.844 |
| Lbs. Balls Per Ton | 2.421 |
| K.W.H. Per Gallon Fuel | 12.143 |
| K.W.H. Per Ton of Ore | 16.58 |
| NaCN Cost Per Ton of Ore | .1187 |
| Ball Cost Per Ton of Ore | .0444 |

# MILL DATA FOR MARCH 1940

| | |
|---|---|
| Average Heads | .0677 |
| Average Tails | .0192 |
| Recovery | 71.64% |
| Tons Milled | 8200. |
| Tons Per Day | 265. |
| Tons Per Hour | 11.04 |
| % Full Running Time | 95.97% |
| Lbs. NaCN Per Ton | .9024 |
| Lbs. Zinc Per Ton | .1904 |
| Lbs. Lime Per Ton | 5.029 |
| Lbs. Balls Per Ton | 2.446 |
| K.W.H. Per Gal. Fuel | 12.058 |
| K.W.H. Per Ton of Ore | 16.32 |
| NaCN Cost Per Ton of Ore | .1163 |
| Ball Cost Per Ton of Ore | .0449 |

## MILL DATA FOR APRIL 1940

| | |
|---|---|
| Average Heads | .0701 |
| Average Tails | .0256 |
| Recovery | 63.4% |
| Tons Milled | 8795 |
| Tons Per Day | 293 |
| Tons Per Hour | 12.2 |
| % Full Running Time | 99.2% |
| Lbs. NaCN per ton | .842 |
| Lbs. Zinc per ton | .1772 |
| Lbs. Lime per ton | 4.13 |
| Lbs. Balls per ton | 2.095 |
| K. W. H. per gallong fuel oil | 11.95 |
| K. W. H. per ton of ore | 15.22 |
| NaCN cost per ton of ore | .1086 |
| Balls cost per ton of ore | .0385 |

MILL DATA FOR MAY 1949

| | |
|---|---|
| Average Heads | .1186 |
| Average Tails | .0303 |
| Recovery | 74.5% |
| Tons Milled | 6089 |
| Tons Per Day | 196.42 |
| Tons Per Hour | 8.19 |
| % Full Running Time | 91.4% |
| Lbs. NaCN Per Ton | 1.015 |
| Lbs. Zinc Per Ton | .2595 |
| Lbs. Lime Per Ton | 5.49 |
| Lbs. Balls Per Ton | 2.828 |
| K. W. H. Per Gallon Fuel Oil | 11.92 |
| K. W. H. Per Ton of Ore | 22.15 |
| NaCN Cost Per Ton of Ore | .1299 |
| Ball Cost Per Ton of Ore | .0519 |

## MILL DATA FOR JUNE 1940

| | |
|---|---|
| Average Heads | .1199 |
| Average Tails | .0325 |
| Recovery | 72.8 |
| Tons Milled | 4985. |
| Tons Per Day | 166.2 |
| Tons Per Hour | 6.93 |
| % Mill Running Time | 79. |
| Lbs. NaCN Per Ton | 1.062 |
| Lbs. Zinc Per Ton | .2772 |
| Lbs. Lime Per Ton | 6.98 |
| Lbs. Balls Per Ton | 2.768 |
| K. W. H. Per Gal. Fuel Oil | 11.73 |
| K. W. H. Per Ton of Ore | 22.60 |
| NaCN Cost Per Ton of Ore | .1359 |
| Ball Cost Per Ton of Ore | .0498 |

## REMARKS ON MILL REPORT FOR JULY 1940

| | |
|---|---|
| 1st. | Jaw crusher down for repairs. Mine pump pumping at 50 gal. per min. |
| 3rd. | Press down 4½ hours to repair press pump. |
| 4th. | Ore bins emptied to repair feeders. |
| 5th. | Press cleaned. Ball mill down 6 hours to change scoop lip and check liners. |
| 6th. | Mill down, out of water. Melted precipitates. |
| 7th. | Mill down, out of water. |
| 8th. | Mill down, out of water. Lowering rakes. J. C. Phelps shipped 22.85 tons of ore. |
| 9th. | Mill still out of water. |
| 10th. | Mill on 11:30 P. M. trouble with pump line to Hi pressure. |
| 12th. | Wm. Hershkowitz shipped two lots of 33.38 tons and 17.3 tons. |
| 15th. | Press off two hours to repair press pump. Press cleaned. |
| 16th. | Trouble with oversized feed. |
| 17th. | Feed off 1 hour to repair chute leading to ball mill. |
| 18th. | Melted precipitates. |
| 22nd. | Jim Reagan shipped 31.74 tons of ore. |
| 23rd. | Press cleaned and zinc feeder repaired. |
| 26th. | Melted precipitates. Wm. Hershkowitz shipped 52.71 tons of ore. |
| 27th. | Andy Bilch shipped 14.45 tons of ore. |
| 28th. | Press cleaned. |
| 30th. | Lester Jaycox shipped 45.75 tons of ore. |
| 31st. | Press cleaned and precipitates melted. |

MILL DATA FOR AUGUST 1940

| | |
|---|---|
| Average Heads | .0717 |
| Average Tails | .0235 |
| Recovery | 67.22 |
| Tons Milled | 7859 |
| Tons Per Day | 253.516 |
| Tons Per Hour | 10.564 |
| % Full Running Time | 95.69 |
| Lbs. NaCN Per Ton | .8906 |
| Lbs. ZINC Per Ton | .2019 |
| Lbs. LIME Per Ton | 4.2589 |
| Lbs. BALLS Per Ton | 1.976 |
| K.W.H. Per Gallon Fuel Oil | 11.909 |
| K.W.H. Per Ton Of Ore | 14.5459 |
| NaCN Cost Per Ton Of Ore | .1139 |
| Ball Cost Per Ton Of Ore | .03779 |

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# MILL DATA FOR SEPT. 1940

| | |
|---|---|
| Average Heads | .0973 |
| Average Tails | .0299 |
| Recovery | 69.27 |
| Tons Milled | 8930 |
| Tons Per Day | 296.6 |
| Tons Per Hour | 12.36 |
| % Full Running Time | 96.4 |
| Lbs. NaCN Per Ton | .779 |
| Lbs. Zinc Per Ton | .198 |
| Lbs. Lime Per Ton | 3.652 |
| Lbs. Balls Per Ton | 1.933 |
| K.W.H. Per Gallon Fuel Oil | 11.842 |
| K.W.H. Per Ton Ore | 12.73 |
| NaCN Cost Per Ton Of Ore | .101 |
| Ball Cost Per Ton Of Ore | .033 |

- - -

# MILL DATA FOR OCTOBER 1940

| | |
|---|---|
| Average Heads | .091 |
| Average Tails | .033 |
| % Recovery | 63.74 |
| Tons Milled | 9000 |
| Tons Per Day | 290.3 |
| Tons Per Hour | 12.09 |
| % Full Running Time | 81.89 |
| Lbs. NaCN Per Ton | .777 |
| Lbs. Zinc Per Ton | .176 |
| Lbs. Lime Per Ton | 3.719 |
| Lbs. Balls Per Ton | 1.725 |
| K.W.H. Per Gallon Fuel Oil | 11.714 |
| K.W.H. Per Ton Of Ore | 12.701 |
| NaCN Cost Per Ton Of Ore | .0985 |
| Ball Cost Per Ton Of Ore | .0277 |

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MILL DATA FOR NOVEMBER 1940

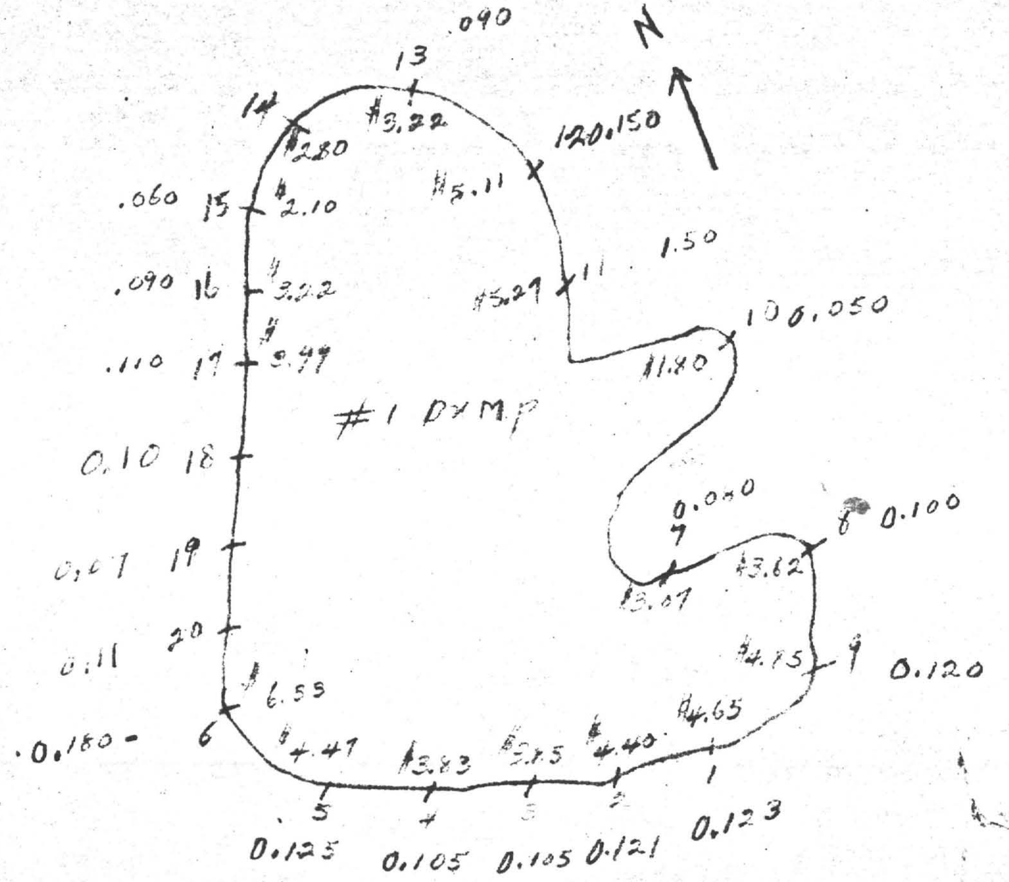
| | |
|---|---|
| Average Heads - - - | .0869 |
| Average Tails - - - | .0325 |
| % Recovery - - - | 62.5876 |
| Tons Milled - - - | 9200.0000 |
| Tons Per Day - - - | 306.6666 |
| Tons Per Hour - - - | 12.7777 |
| % Full Running Time - - | 91.2150 |
| Lbs. NaCN Per Ton - - - | .6929 |
| Lbs. Zinc Per Ton - - - | .1633 |
| Lbs. Lime Per Ton - - - | 3.4109 |
| Lbs. Balls Per Ton - - | 1.8071 |
| K.W.H. Per Gallon Fuel Oil - - | 11.7982 |
| K.W.H. Per Ton Ore - - | 13.2268 |
| NaCN Cost Per Ton Ore - - | .08789 |
| Ball Cost Per Ton Ore - - | .03317 |

- - - -

## MILL DATA FOR DECEMBER 1940

| | | | | |
|---|---|---|---|---|
| Average Heads | - | - | - | .0773 |
| Average Tails | - | - | - | .0316 |
| % Recovery | - | - | - | 59.1227 |
| Tons Milled | - | - | - | 7500.0000 |
| Tons Per Day | - | - | - | 241.9354 |
| Tons Per Hour | - | - | - | 10.0806 |
| % Full Running Time | - | - | - | 81.5860 |
| Lbs. NaCN Per Ton | - | - | - | .9333 |
| Lbs. Zinc Per Ton | - | - | - | .3073 |
| Lbs. Lime Per Ton | - | - | - | 3.4372 |
| Lbs. Balls Per Ton | - | - | - | 2.0300 |
| K. W. H. Per Gallon Fuel Oil | | - | - | 12.0210 |
| K. W. H. Per Ton Ore | | - | - | 15.2780 |
| NaCN Cost Per Ton Ore | | - | - | .1134 |
| Ball Cost Per Ton Ore | | - | - | .0373 |

- - - - -




18,000 Tons Remaining 1974

Average 0.106 oz. Au 140.00 14.84

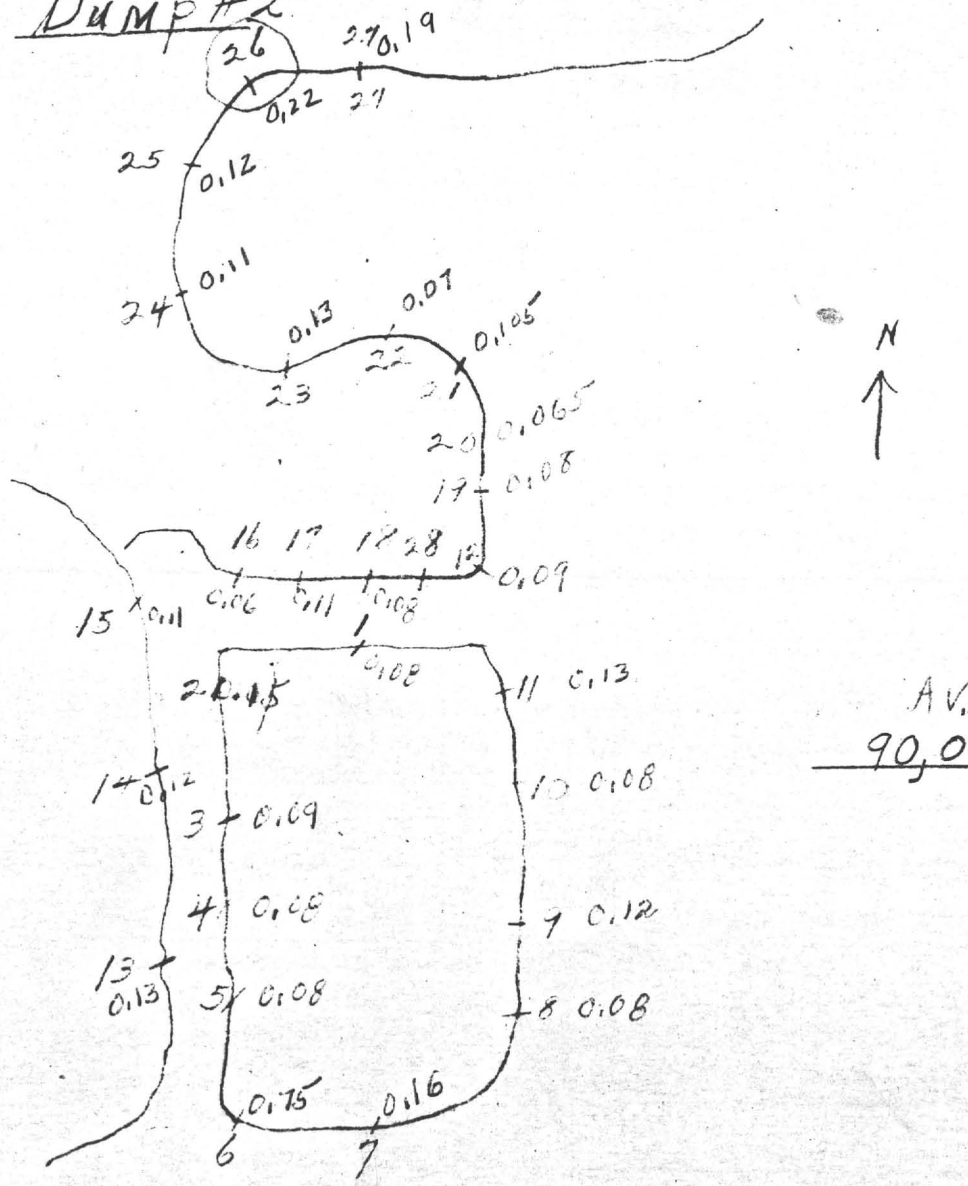
0.20 oz Ag X 4.00 0.80

\$15.64

Congress Dump #1

Congress

Dump #2

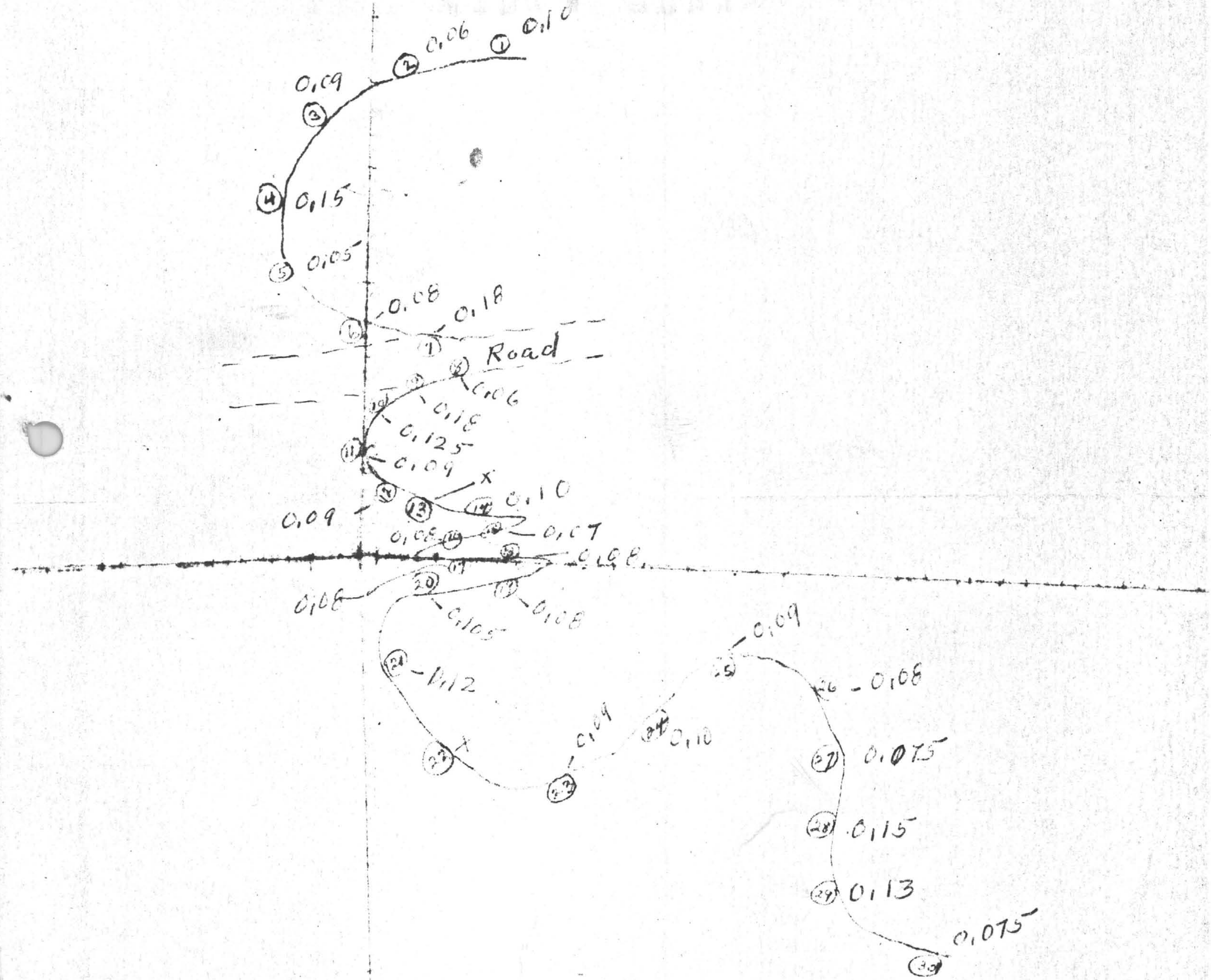



AV. 0.1093

90,000 Tons 1974

Congress

Dump #3

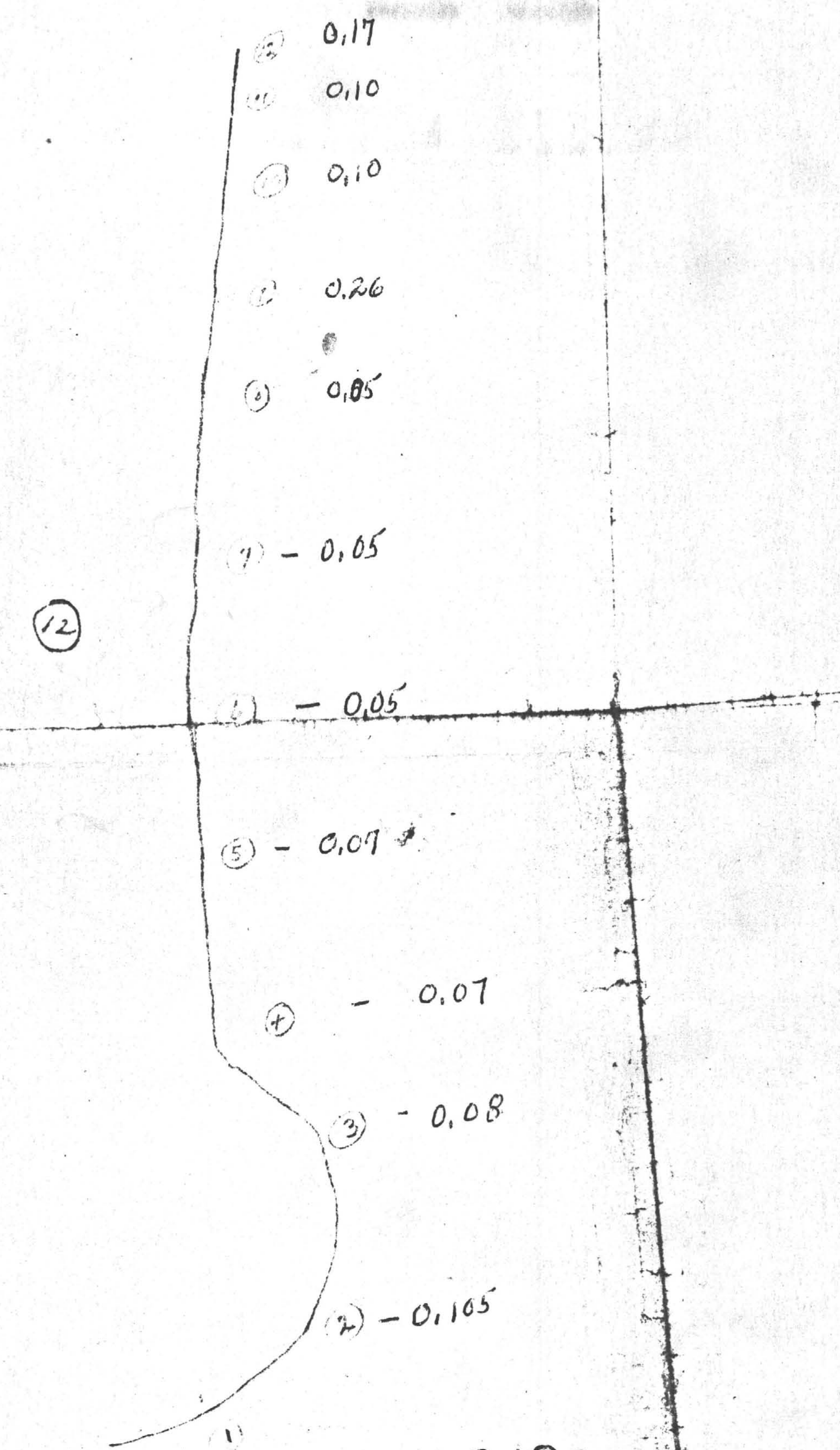



80,000 Tons

Av. 0.10 oze Au.

Niagara

Dump #4

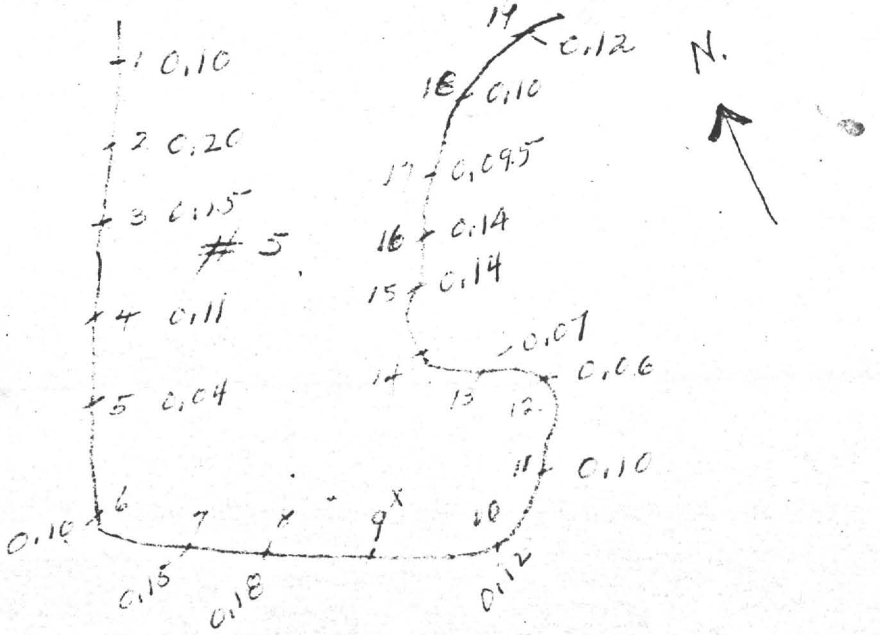



average 0.10 oz

1000 Tons 1974


DuMp #5

Niagara




Average 0.114 oz au

50,000 Tons Remaining 1974

| NAME | BOOK | PAGE |
|---|---|---|
| Jaquays No. 1 Amended | 848 | 855 |
| Jaquays No. 2 " | 848 | 856 |
| Jaquays No. 3 " | 848 | 857 |
| Jaquays No. 4 | 112 | 316 |
| Jaquays No. 9 Amended | 848 | 850 |
| Jaquays No. 10 | 112 | 321 |
| Jaquays No. 11 | 112 | 322 |
| Jaquays No. 12 Amended | 212 | 213 |
| Jaquays No. 13 | 112 | 325 |
| Jaquays No. 16 | 848 | 859 |
| Jaquays No. 17 | 112 | 329 |
| Jaquays No. 20 | 848 | 851 |
| Jaquays No. 21 | 848 | 852 |
| Jaquays No. 22 | 848 | 853 |
| Jaquays No. 23 | 848 | 854 |
| * B & M | 327 | 5 |
| Congress Extension No. 1 | 657 | 875 |
| " " No. 2 | 657 | 876 |
| " " No. 3 | 657 | 877 |
| " " No. 4 | 657 | 878 |
| " " No. 5 | 657 | 879 |
| " " No. 6 | 657 | 880 |
| " " No. 7 | 657 | 881 |
| " " No. 8 | 845 | 332 |
| " " No. 9 | 845 | 333 |
| * Jaquays No. 35 | 904 | 217 |
| * Jaquays No. 28 | 904 | 218 |
| * Congress Extension No. 13-A | 904 | 214 |
| * " " No. 14-A | 904 | 215 |
| * " " No. 15-A | 904 | 216 |
| * " " No. 20 | 904 | 220 |
| * " " No. 20 Amended | 905 | 215 |
| * " " No. 21 | 904 | 221 |
| * " " No. 22 | 904 | 222 |
| * Jaquays No. 5-A Fraction | 904 | 219 |

* Indicates Claims on which Patent has been applied for.