THE CANADIAN MINING JOURNAL

VOL. XXXV. TORONTO, February 1, 1914. No. 3

The Canadian Mining Journal

With which is incorporated the

"CANADIAN MINING REVIEW"

Devoted to Mining, Metallurgy and Allied Industries in Canada.

Published fortnightly by the

MINES PUBLISHING CO., LIMITED

Head Office - - 2nd Floor, 44 and 46 Lombard St., Toronto
Branch Office - - - - - - 34B Board of Trade Building
London Office - - - Walter R. Skinner, 11-12 Clement's Lane London, E.C.

Editor

REGINALD E. HORE

SUBSCRIPTIONS—Payable in advance, $2.00 a year of 24 numbers, including postage in Canada. In all other countries, including postage, $3.00 a year.

Advertising copy should reach the Toronto Office by the 8th, for issues of the 15th of each month, and by the 23rd for the issues of the first of the following month. If proof is required, the copy should be sent so that the accepted proof will reach the Toronto Office by the above dates.

CIRCULATION.

"Entered as second-class matter April 23rd, 1908, at the post office at Buffalo, N.Y., under the Act of Congress of March 3rd 1879."

CONTENTS.

CANADIAN MINING INSTITUTE HEADQUARTERS.

There has of late been considerable discussion, especially among Montreal and Toronto members, as to the advisability of making an effort to raise funds for the purpose of securing permanent headquarters for the Institute. It is argued that the present arrangement, whereby rooms are leased at the Windsor Hotel, Montreal, is not satisfactory and that few members make use of the rooms or library.

There seems to be an impression that members visiting the city would like to have a more attractive place, something more in the nature of a club, with club privileges. To provide and maintain such headquarters, however, a considerable sum would have to be raised.

The question now comes up as to whether the Institute is ready to incur the additional expense and the increased annual dues or other assessments which a more pretentious home would make necessary. Certainly those members who do not frequently visit the city cannot be expected to be very enthusiastic over the proposal. Many of them feel already that they get too little return for their money. If the city members wish, therefore, to add to the expenditures it is only fair that they should bear the whole burden. The city members realize this and the plans so far talked of are based chiefly on the expectation that wealthy members are ready to make donations for the purpose.

Recently the Montreal branch has had under serious consideration a proposal to raise funds for the purpose of securing headquarters for the branch. This proposal seems to have been confused with the other one of securing quarters for the Institute, by the suggestion that subscriptions be accepted from members of the Institute who are not resident in Montreal.

Some members will doubtless object to any plan which calls for expenditure of the funds of the Institute on permanent headquarters.

TREATMENT OF GOLD ORE AT THE DOME MINE.

In this issue we reprint a paper by Mr. D. L. H. Forbes which was read at the 1913 meeting of the Canadian Mining Institute and which has just been published. In the several months which have elapsed since Mr. Forbes' paper was written important additions have been made at the mill along the lines outlined in his paper. Forty stamps have been added and the foundations for a sand leaching plant are in place. It is expected that in a few months a large increase in production will be possible.

In the mill running at present there have been no radical changes made. A coarser screen, about 4 mesh, is now used to advantage since the installation of an additional tube mill. Gradually the output has been increased until now it is over 450 tons per day.

The Dome mill was constructed under circumstances far from favorable. The Merrill Metallurgical Company was called upon to design hurriedly a plant to treat an ore of which little was known. Speed was necessary in order to take advantage of the winter roads for transportation of heavy machinery from Kelso, the nearest point on the railway. The Porcupine branch of the T. & N. O. Ry. had not then been constructed.

Fire destroyed the plant under construction and again a hurry order was sent in for a complete plant.

The mill was not one specially designed for Porcupine ores; but rather an experimental one to be used in testing the ore bodies and in determining the best method of treating the ores. Naturally, therefore, the results at first obtained were not as satisfactory as those obtained later. A great deal of information has been gathered by the metallurgists in charge of the investigations, and some of this information is given in Mr. Forbes' paper.

During the past year the mill has been giving better and better results and is now treating about 450 tons per day and making a very high recovery of the gold contained in the ore.

The Dome Mines Company in enlarging its plant is undertaking still further important investigations. While the present plant is now giving what might be considered fairly satisfactory results, it is thought that still better results can be obtained by separating the sands for treatment without regrinding. Every effort is being made to determine the best method of treating the ore. The men who have done the work, the staff of the Dome Mining Company and the staff of the Merrill Metallurgical Company, deserve much credit for their contributions to our knowledge of Ontario gold deposits and methods of treating the ores.

BRITISH COLUMBIA IN 1913.

The British Columbia Bureau of Mines has published a preliminary review and estimate of the mineral production of the province for the past year, together with some notes on the progress made in the mining and metallurgical industries.

The prompt appearance of the bulletin is creditable to the Bureau and to the Provincial Mineralogist, Mr. Wm. Fleet Robertson.

The estimated production is as follows: Gold, 266,547 oz.; silver, 3,569,642 oz.; lead, 54,205,594 lb.; copper, 46,042,379 lb.; zinc, 7,100,000 lb.; coal, 2,136,694 tons (2,240 lb.); coke, 285,123 tons.

In general, the report agrees with the earlier summary of our correspondent, Mr. E. Jacobs, which was published in our January 15 issue. It contains also considerable additional information, some of which will be found in this issue of the Journal.

The total value of the 1913 production was $30,158,793, as against $32,440,800 in 1912. Copper and coal outputs were considerably smaller than in the previous year. Gold, silver, lead and zinc show an increase.

THE RECOVERY OF SILVER FROM COBALT ORES.

Remarkable progress has been made in the treatment of the silver ores of the Cobalt district. Previous to the discovery of the phenomenal silver deposits in Northern Ontario no ores of this type were being mined on this continent, and little was known of the methods of treating the similar ores mined in Germany.

The ores consist chiefly of native silver, associated with arsenides of cobalt and nickel. The gangue mineral is chiefly calcite. Silver sulphides and antimonides occur also, although in much smaller quantity than the native silver.

Fortunately much of the ore is so rich that large profits were made by simply roasting the ores to drive off the arsenic, and then smelting the product, and refining the base bullion. When the deposits were first opened cobalt and arsenic commanded good prices and the marketing of these constituents helped to defray the cost of recovering the silver. The price soon fell, however, when large quantities were produced.

High grade ore was for some time the only marketable product of the mines, if we except a comparatively small amount of low grade used by some of the smelters.

Soon, however, the lower grade material was investigated carefully and in a few years several mills were erected to crush and concentrate the ore. Straight concentration methods proved very satisfactory and a high recovery was, and is, obtained by the millmen at Cobalt.

Then cyanidation and amalgamation were successfully introduced, and some very remarkable and entirely new methods of treatment were worked out. To the staff of the Nipissing Mining Company and to its consulting metallurgists belongs the credit for many of the more recent marked advances in methods of treating both high and low grade ores.

In this issue we publish a few pages from a very interesting description of the mill and metallurgical methods of the Nipissing mine, by James Johnston. Mr. Johnston's paper was prepared for the annual meeting of the American Institute of Mining Engineers and has been published in the January bulletin. Mr. Johnston is to present a paper on the same subject at the annual meeting of the Canadian Mining Institute in March. The author describes carefully the construction and equipment of the mill and the methods of treating the ores. The costs for several operations and the results obtained are clearly stated. The paper should prove to be one of the most interesting ones presented at the annual meeting.

MINING GOLD ORE AT DOME MINE

View Across Open Pits Towards No. 2 Shaft, Dome Mine.

Good progress is being made at the Dome mine. The ore bodies are being now developed by drifts and crosscuts at the 425 ft. level. Openings have been made underground to permit of a large proportion of the ore being mined there when extreme weather conditions prevail. Nearly all of the ore milled so far has been broken in the open pits. The preparations now made will permit of the discontinuance of work in the open when the weather is very severe.

The men working in the open break a very large tonnage in an 8-hour shift. The practice is to have the miners work one shift per day only. The blasting is done at night. The blasters use one machine to drill the large blocks.

Rand hammer drills, one man machines, are used for this work in the pits. Underground the Ingersoll-Rand No. 43 piston drill is used in crosscuts and drifts. In raises Rand Butterfly stoper drills are used.

Miners Working in one of the Open Pits, Dome Mine.

During the first three months of 1913 there was broken in stoping 43.7 tons per machine-shift. The Rand No. 43 drills averaged 30 ft. per machine-shift, and the Rand hand hammer drills averaged 38.7 ft. per machine-shift. The more recent records show a considerably higher tonnage per machine. For the deeper ore the main hoisting way is to be the No. 2 shaft, a vertical one. The ore from the open pits is now being hoisted up an incline shaft directly to the crushers.

Miners Working in one of the Open Pits, Dome Mine.

It is the intention to avoid the surface tramming of ore by putting in an ore bin at the 100 ft. level of No. 2 shaft. From the lower workings the ore will be raised and delivered to this bin. It will then be run into cars which will be drawn to the incline shaft by mules. These cars will be hauled by engine up into the crusher house as at present.

CANADIAN MINING INSTITUTE ANNUAL MEETING.

The Sixteenth Annual Meeting will be held in the City of Montreal on Wednesday, Thursday, and Friday, March 4th, 5th, and 6th, 1914.

The Institute's headquarters will be the Windsor Hotel. Applications for accommodation should be made through the Secretary of the Institute, or direct to the manager of the Windsor Hotel.

Among the papers already promised for the meeting are the following:

"Mill and Metallurgical Practice at the Nipissing Mining Company, Cobalt, Ont.," by James Johnston, Cobalt, Ont.; "The Sampling of Cobalt Ores," by C. St. G. Campbell, Cobalt, Ont.; "The Veins of Cobalt District, Ont.," by Arthur A. Cole, Cobalt Ont.; "Recent Improvements in Cyanidation," by Herbert A. Megraw, New York; '"Some Notes on Mining and Milling Practice at the Alaska Treadwell Mine," by H. C. Meek, South Porcupine, Ont.; "Ore Dressing Improvements," by Robert H. Richards, Boston, Mass.; "Recent Metallurgical Developments," by A. Stansfield, Montreal, Que.; "Methods of Excavation in the Mount Royal Tunnel," by S. P. Brown, Montreal, Que.; "Factors Influencing the Cost of Power for Mining Purposes," by J. M. Forbes, Montreal, Que.; "High Carbon Steel for Sluice Linings in Hydraulic Mining," by Howard W. Dubois, Philadelphia, Pa.; "Mining in British Columbia" (illustrated by colored lantern slides), by Howard W. Dubois, Philadelphia, Pa.; "Scientific Management," by F. G. Gilbreth, New York; "Efficiency Engineering Applied to Mining, Quarries, and Industrial Plants," by H. M. Payne, New York; "The Chisana Gold Field," by D. D. Cairnes, Ottawa, Ont.; "Coal Resources of the World," by D. B. Dowling, Ottawa, Ont.; "Asbestos Resources of the Thetford Area," by W. J. Woolsey, Thetford Mines, Que.; "Safety in Coal Mines" (illustrated by moving pictures), by a representative of the H. C. Frick Coke Company, Pittsburgh, Pa.

CORRESPONDENCE

Editor Canadian Mining Journal:

Sir.—In a recent issue of your valued paper appeared an item headed "Oxygen Breathing Apparatus," in which item Mr. F. W. Gray, of the Dominion Coal Company, has been given just credit for work he has done in connection with the development of the use of rescue apparatus in this country.

But you state, "He was the first person on this continent to bring over a rescue apparatus of the oxygen type." This statement is at variance with the facts, and if there is any special honor attached to the person having first introduced such an apparatus into Canada, that credit is due to Mr. R. H. Brown, mining engineer, of Halifax, N.S.

Mr. Brown, in the year 1895, while general manager and agent for the General Mining Association, imported a Siebe Gorman type of oxygen rescue apparatus for emergency use in the coal mines then operated by that company.

This apparatus was in the possession of the General Mining Association, and of their successors, the Nova Scotia Steel & Coal Company, Ltd., up to this year.

You will notice that the date of the use of the oxygen type of rescue apparatus antedates by a good many years the arrival of Mr. Gray in this country.

While I am anxious that any person who does anything that is of benefit to the mining interest or fraternity of the province or the Dominion, should receive due credit for same, and while I am not desirous of disparaging Mr. Gray's efforts in this line, having the above knowledge. I feel it is my duty to give the credit to whom the credit is due. There is not the slightest doubt but that Mr. Brown and his officers were quite familiar with and had in use the oxygen rescue apparatus type of machine quite a number of years previous to its coming into general use.

Yours,

THOS. J. BROWN,

General Superintendent N.S. Steel & Coal Co., Ltd.

TREATMENT OF GOLD ORE AT DOME MINE, SOUTH PORCUPINE, ONTARIO*

By D. L. H. Forbes.

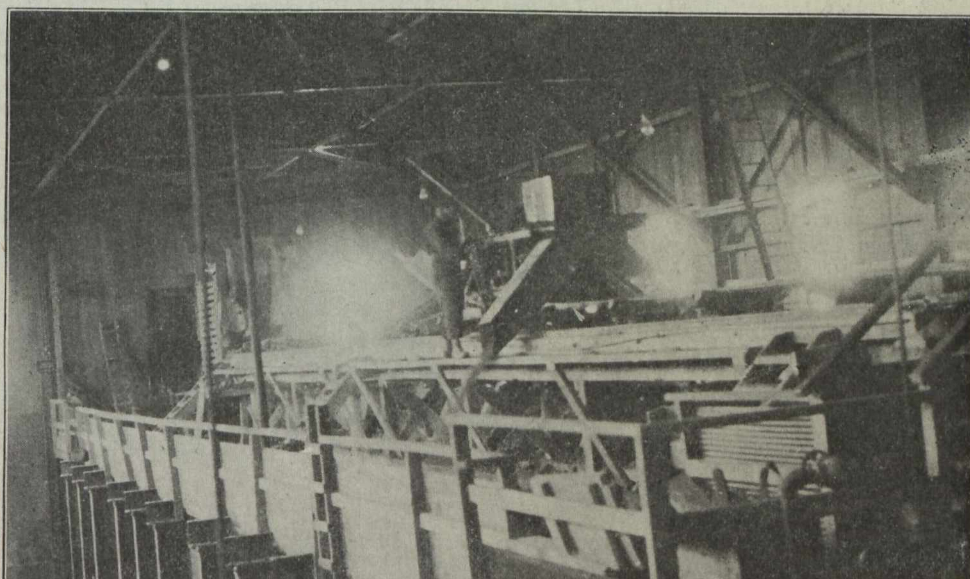
Late in the fall of 1910 it was decided to erect a mill at the Dome mine. The Porcupine branch of the Timiskaming & Northern Ontario Railway had not then been built, so that supplies and machinery had to be transported in winter over rough roads from Kelso. It was a matter of the utmost importance to the Dome Mines Company that the mill should be designed without delay, to insure that all material and machinery for construction could be ordered in time to reach Kelso before the breaking up of the winter roads. Two separate samples of Dome ore were sent to San Francisco for testing; but, as these samples were taken from the mine in its early stages of development and differed considerably in value, the design of the mill had to be of the most flexible nature, to provide for a high extraction even if future development of the mine should disclose ore of different character to that tested.

It was originally planned that the building should be a wooden structure, but after the disastrous fire in the summer of 1911, when the work of construction was already well under way, a change was made to steel construction; thus ensuring a fire-proof building.

In spite of delay caused by the fire, the mill was completed in record time and placed in operation on March 22, 1912. Acting for the Merrill Metallurgical Company, Mr. Henry Hanson supervised the installation. To his energy and ability in overcoming the many obstacles incidental to construction work in a new mining camp, the success achieved was, in a large measure, due.

The Dome ore consists of quartz and schisted slate that has been sheared and metamorphozed to such an extent as to be scarcely distinguishable, except in toughness, from the darker colored schists of the Pearl Lake section of the district. Both the quartz and the slate have been heavily mineralized with pyrite so that the ore is heavier than most quartz gold ores, its specific gravity averaging about 2.8. The ore is reduced under stamps and in tube mills to a granular slime with the production of relatively only a small proportion of colloids, giving a material that is almost ideal for settling and pressure filtration in the cyaniding part of the treatment. While some of the gold is coarse, no large nuggets have been found in the mortars of the stamp batteries, and, in general, a considerable proportion of the gold occurs in such fine particles that tube mill grinding is necessary for its liberation preparatory to either plate amalgamation or cyanide treatment.

In order to be prepared for an acid ore which might come from the lower workings of the mine, as well as to insure efficient amalgamation, with the consequent quick and complete cyanidation of the amalgamation tailings, it was decided to crush and amalgamate in water. This provides against an excessive consumption of cyanide in the cyanide treatment, and also safeguards against losses due to the presence of coarse gold.



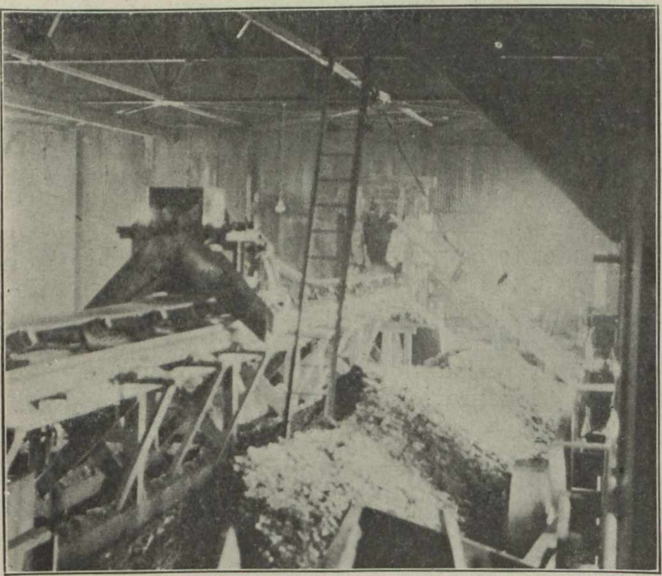
Filling Stamp-Mill Bins with the Crushed Ore, Dome Mill

The mill as first designed was considered a testing unit of a daily capacity of 350 tons to prove the economic value of the ore in the mine, and to indicate the best metallurgical treatment for future adoption, when additions to the milling plant would be required. Regarded as a large experimental mill for Porcupine ores, the results of the operations to date are interesting. It is the purpose of this paper to present a general outline of the plant, and to refer to the results of its operations from the technical rather than from the economic aspect, as many factors, such as power, water supply, and ore tonnage, which affect operating costs, have been of a temporary nature; hence costs based on such conditions might be misleading.

*A paper read at Annual Meeting of C. M. I., 1913.

Crushing.

Preliminary crushing of the ore is done in Kennedy gyratory breakers. The open cut system of mining produces much ore that is over thirty inches in size. To handle such large lumps of ore a crusher with a wide jaw opening is required. A No. 7½ breaker takes the ore dropped from the mine cars and crushes it through an opening of about 6 inches. The broken ore is then elevated by a belt conveyor and screened over a grizzly having 1½ inch openings, from which the oversize goes to two No. 3 crushers. The secondary crushers reduce the ore to a size of 1½ inches, dropping it on a second inclined belt conveyor, where it joins the undersize from the grizzly and is delivered to the stamp mill bin. The crusher plant and conveyors are housed in a wooden building, which is divided from the main steel building of the mill by a fireproof door and partition. The ore is distributed along the top of the battery bin by means of a third belt conveyor, provided with an automatic tripper. The battery bin is of steel construction throughout, is flat-bottomed and has a capacity of about 1,600 tons.



View of Stamp-Mill Bins, Dome Mill. Shows Conveyor and Distributing Apparatus

Stamp Milling.

From the bin the ore is fed to four ten-stamp batteries by means of suspended Challenge feeders. The stamps weigh 1,250 pounds. There are independent battery line shafts behind the stamps, one for each 20 stamps driven by a 75 h.p. motor placed behind the ore bin on the ground level. The stamps make one hundred and two 6½-inch drops per minute, and crush through ten-mesh, rolled, slotted, wire screens. Several screens of different mesh have been used since the plant was started. The following table gives the average results in stamp duty and size of product when using the various mesh sizes:—

Effect of Screen Opening on Stamp Duty and Product.

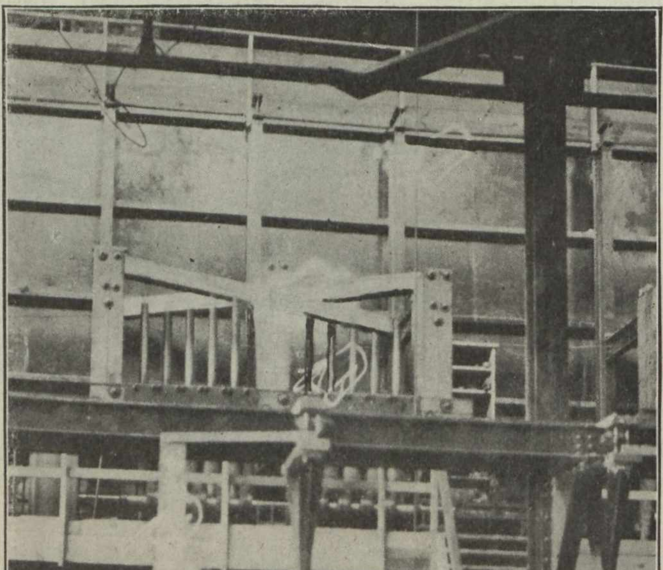
Battery Screen Used	Stamp Duty. Tons per Stamp per Day.	Screen Tests on Battery Discharge. On 60-mesh %	On 100-mesh %	On 200-mesh %	Through 200-mesh %
16-mesh	6.8	41.6	8.8	10.4	38.8
14.mesh	7.9	62.0	6.4	6.8	24.8
12-mesh	9.0	64.0	7.0	6.0	22.8
10-mesh	9.6	59.0	15.0	7.0	19.0

Crushing through 10-mesh screens the stamps give the most economical results. About 7.5 tons of water are fed to the batteries for every ton of ore stamped.

Primary Plates Discarded.—When the mill was started there were eight primary amalgamating plates, one in front of every five stamps, each being 54" x 144" in size with a slope of 1½ inches per foot. It was found when the pulp was passing through 16-mesh screens the primary plates caught more gold than the secondary ones, which follow the tube mills. With 14-mesh screens the amount of gold caught on the primary plates was only slightly greater than that caught on the others, while with 12-mesh screens the secondary plates recovered more of the gold than the primary plates. When 10-mesh screens are used the primary plates recover very little gold and are difficult to keep properly dressed, owing to the scouring action of the coarse sand, hence the primary plates were removed after it was decided to aim at large crushing capacity per stamp.

The relation of the mesh of the screens used on the batteries to the recovery by amalgamation was found to be about as follows:

Using 16-mesh screens, the total recovery by amalgamation was 78 per cent.



View from Front of Batteries, Dome Mill. Note absence of Primary Plates.

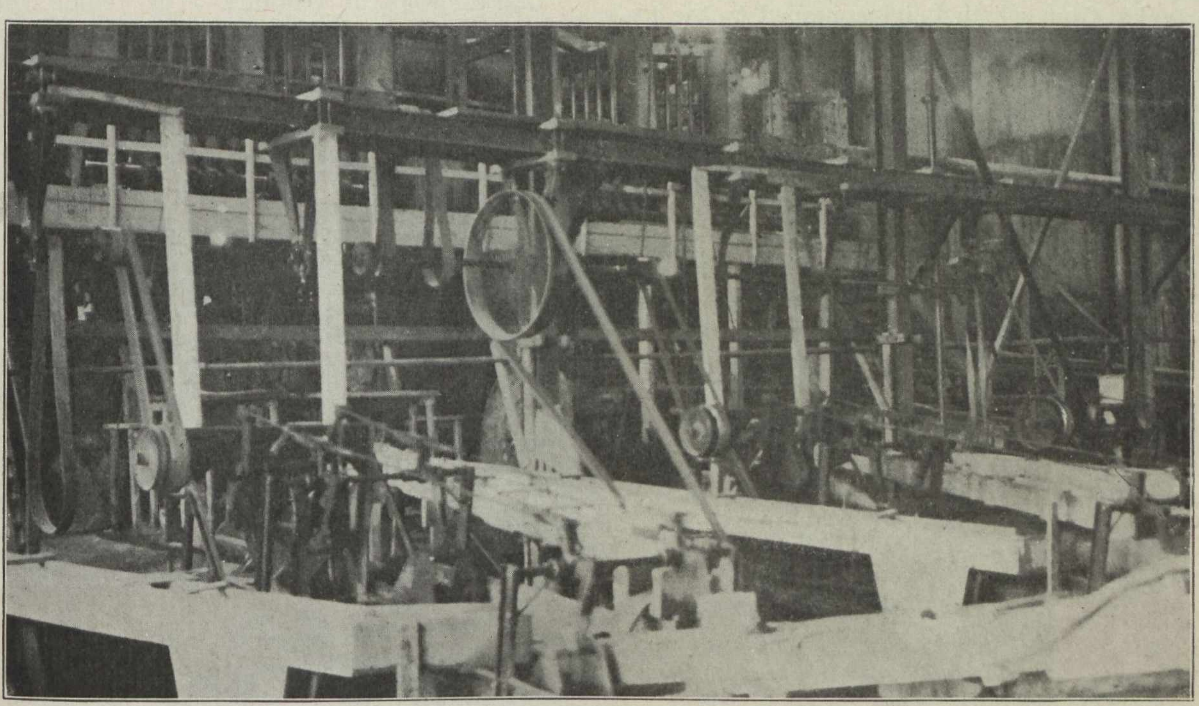
Using 14-mesh screens, the total recovery by amalgamation was 75 per cent.

Using 12-mesh screens, the total recovery by amalgamation was 55 per cent.

Using 10-mesh screens, the total recovery by amalgamation was 46 to 50 per cent.

At the time when the highest recovery by amalgamation was obtained the stamps were giving a duty of only 6.8 tons per day, and the slime treated by the secondary plates was of such a fineness that 85 per cent. would pass a 200-mesh screen.

The lowest recoveries by amalgamation were noted over periods in which the stamp duty was at a maximum, and only 65 per cent. of the slime treated on the secondary plates would pass a 200-mesh screen.

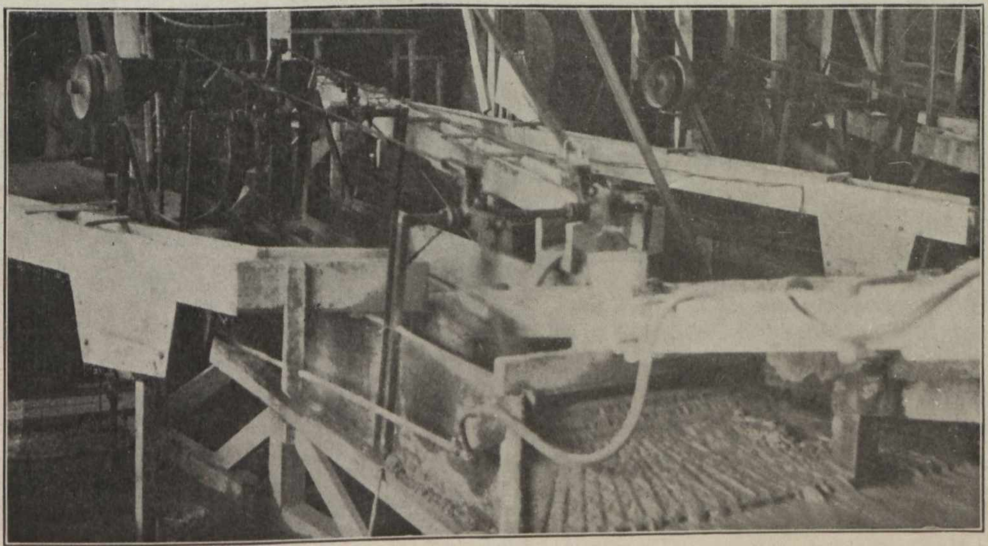


View of Classifiers and Stamps. Dome Mill.

Classification and Tube Milling.

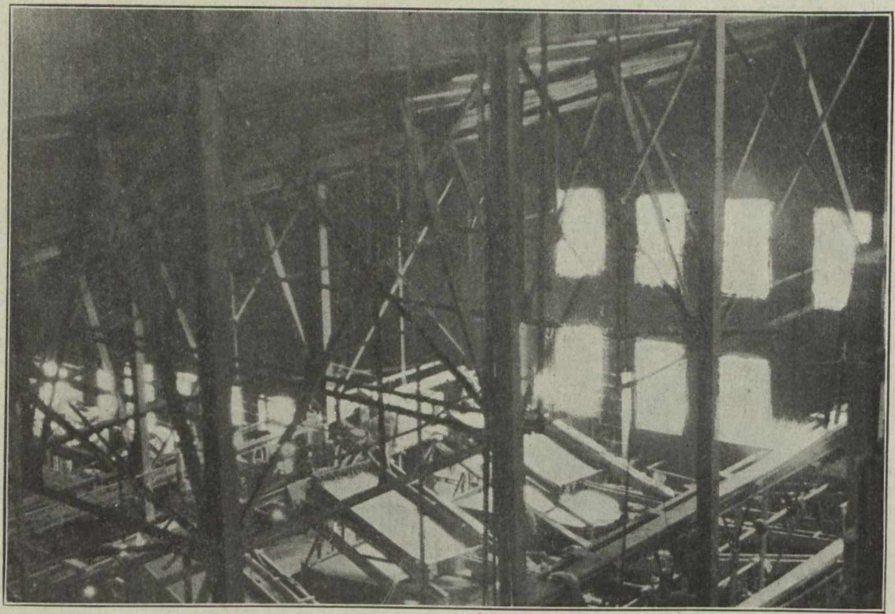
Classification.—The pulp discharged from the batteries goes to four duplex Dorr classifiers, the sand product of which passes to four 5′ x 22′ tube mills. The slime product of the classifiers is carried over four secondary amalgamating plates, one for each classifier. These plates are 108″ x 144″ in size and have a slope of about 1½ inches per foot. After leaving the plates, the slime is passed through sets of Merrill concentrating cones which trap the coarse concentrate and any amalgam or coarse gold that may have escaped from the secondary plates, the coarse concentrate is then returned to the tube mills.

The tube mills are run at 26 to 28 r.p.m. by slip-ring motors coupled direct to the tube mill pinion shafts. Each mill takes approximately 70 h.p. The linings used are of the El Oro type made of chilled cast iron. The discharge of each tube mill is elevated, and returned to the corresponding classifier on the floor above it by means of an 8″ x 40″ Frenier sand pump. When the fineness of the slime is such that 75 per cent. will pass a 200-mesh screen, it has been found that the most economical results are obtained taking into consideration both the total recovery of the gold and the milling cost per ton of ore treated.



View of Dorr Classifier, Dome Mill. Note Slime Overflow Running onto Amalgamating Plate.

Thickening.—The slime is at a dilution of about 10 tons of water to one ton of solids. The water is removed as completely as practicable by settling the slime in three 30′ by 10′ Dorr thickeners, before cyanide treatment is commenced. The thickened slime discharged from these tanks contains about 44 per cent. of moisture. The overflow from the thickeners is usually quite clear, and is collected in 20′ x 10′ storage tanks, from which it is repumped to the tank above the batteries by a triplex plunger pump.



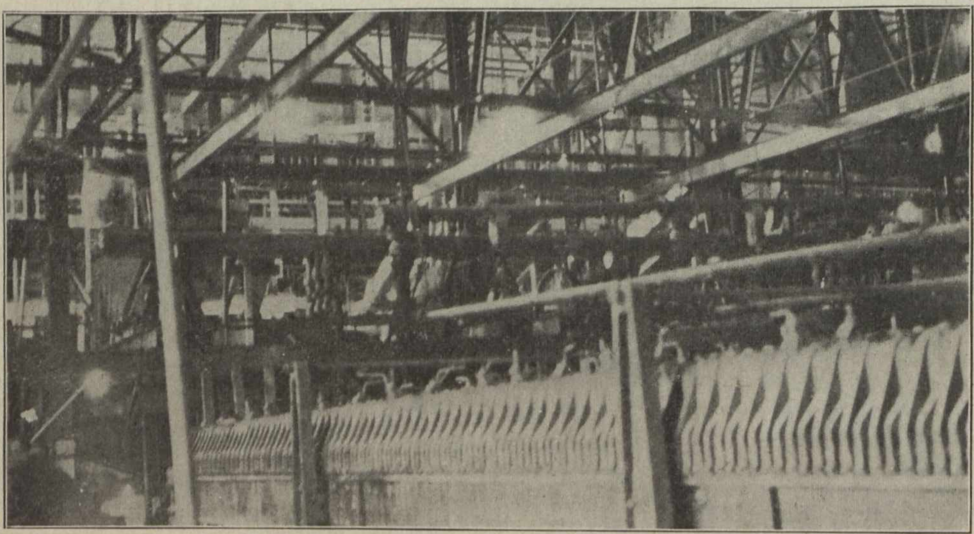
View from above the Stamps, Dome Mill. Shows the Secondary Amalgamating Plates.

Lime is added to the battery water circuit in sufficient quantities to give good settling results in the thickeners, and also to neutralize any latent acidity in the ore. The total consumption of lime averages about 3.2 pounds per ton of ore, and, of this quantity, the larger proportion is used in the battery circuit.

Recovery on Plates.—As might be expected, a certain amount of coarse gold collects with concentrate on the classifier bottoms and remains in the tube mill circuit until ground fine enough to overflow with the slime over the secondary amalgamating plates. These plates, under present practice (January, 1913), recover from 46 to 50 per cent. of the total gold in the ore. The squeezed amalgam taken from these plates carries about $5.50 in gold per oz. The consumption of mercury varies from 0.9 oz. to 1.6 oz. per ton of ore treated.

Cyanide Treatment.

Consumption of Cyanide.—The slime discharged from the Dorr thickeners is ready for cyaniding and is elevated to the treatment tanks by a duplex set of 16-inch bucket elevators which are 70 feet in height between the centres of the pulleys. Cakes of potassium cyanide are dissolved by the pulp as it enters the bottom of the elevators and sufficient cyanide is added to give a treatment solution of about 1 lb. KCN per ton. The average consumption of cyanide varies from month to month between the limits of 0.8 lb. to 1.1 lb. KCN per ton of ore treated. The greater part of this consumption is the mechanical loss due to displacement of weak solution by the water coming in with the slime from the thickeners.



Merrill Slime Filter Press. Dome Mill.

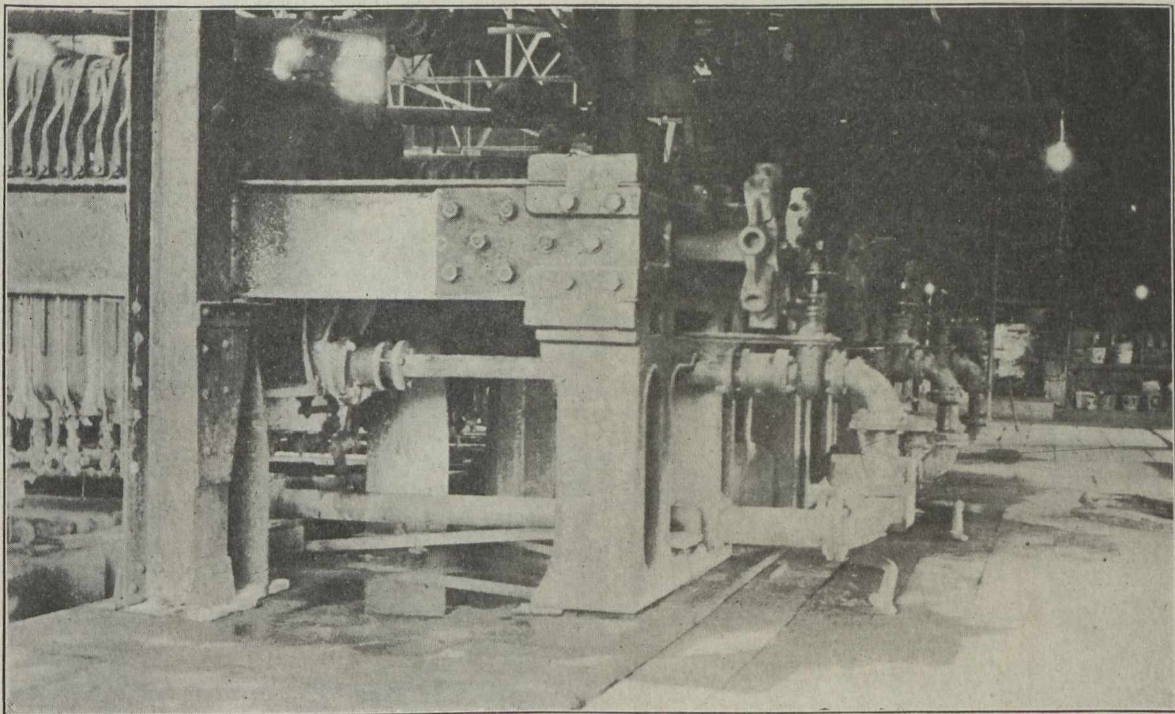
Pachuca Tanks.—The method of treatment in use is that of air agitation by the continuous Pachuca tank system. There are four tanks, 8 feet in diameter by 40 feet in height. The air for agitating the slime in these tanks is compressed to 20 lb. per square inch at the receiver by a 14″ x 16″ single stage, belt driven Rand compressor which is driven at about 240 r.p.m.

From the final treatment tank, the slime is run to a storage tank in which it is mechanically agitated and drawn off intermittently to the filter presses.

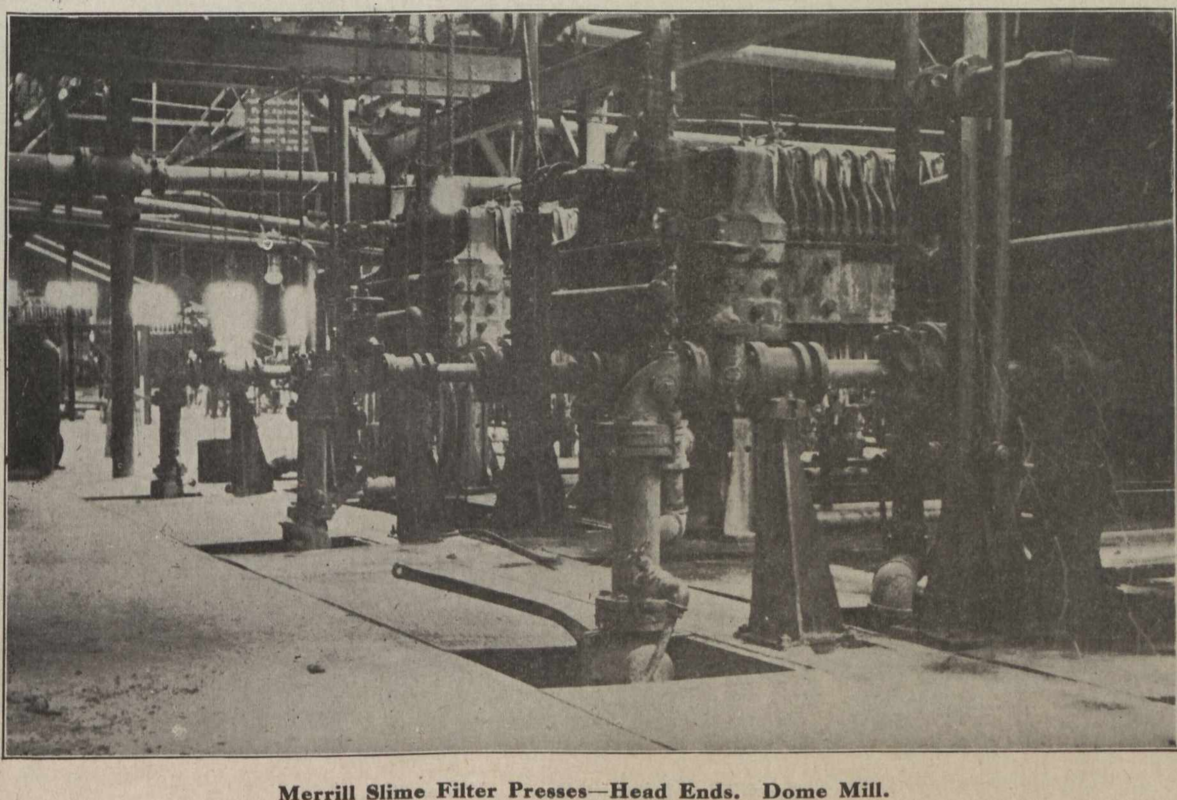
Filtration.

Merrill Presses.—Two Merrill slime filter presses, each having 75 4-in. frames filter the treated slime. At each charging of a press about 20 tons of slime is drawn from the storage tanks at a dilution of 1 to 1, the filling of the presses occupying from 22 to 30 minutes, until the four-in. frames are filled with solid cakes of slime at a final pressure of 20 lb. per square inch.

Washing.—Precipitated solution is then forced

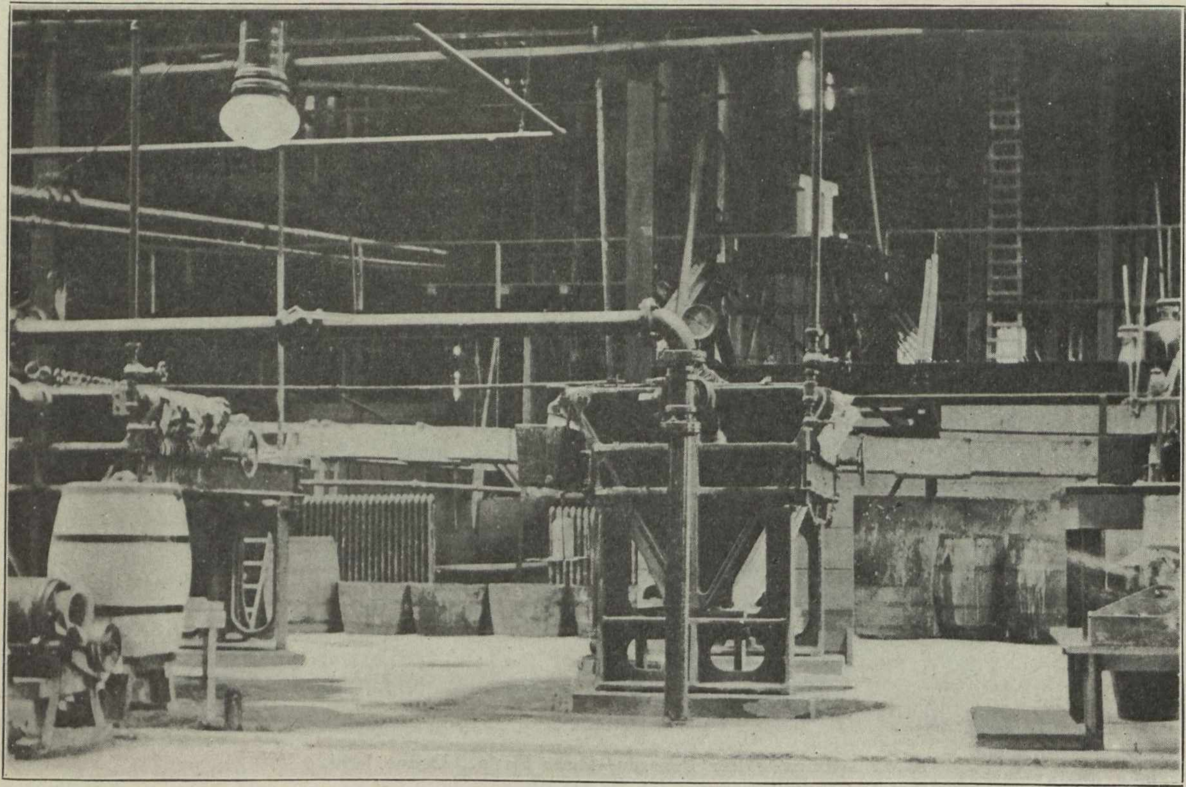


Merrill Slime Filter Presses—Rear Ends. Dome Mill.

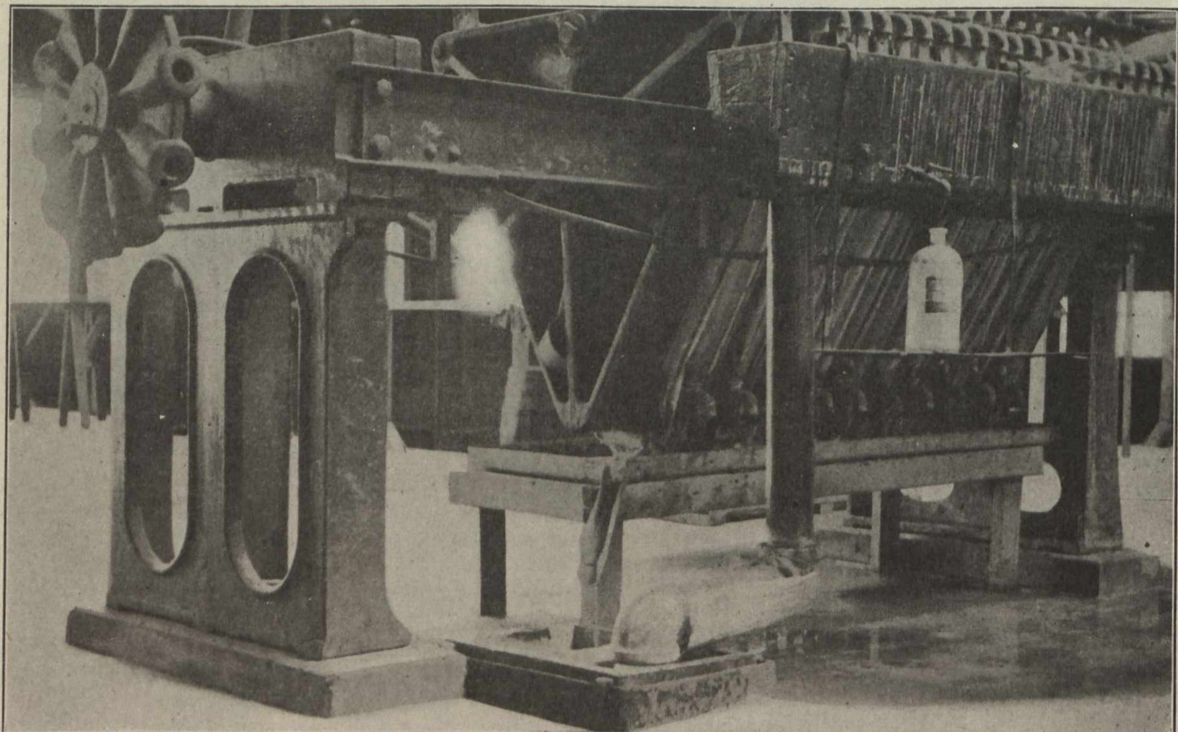


Merrill Slime Filter Presses—Head Ends. Dome Mill.

through the cakes of slime for washing out the dissolved gold remaining in the cakes. During this washing period it has been noticed that a perceptible extraction of undissolved gold takes place. The time allowed for washing is determined by the operators who measure the rate of the solution passing through the presses, so that approximately one ton of wash solution will be passed for every ton of dry slime in the charge. After washing is complete, the discharge valves are opened, and then the mechanism for sluicing out the residues is started. The time required for discharging a press is usually about 50 minutes. The residues discharged from the filter presses with the sluicing water are collected in two 25′ x 10′ Dorr slime thickening tanks, in which the excess water used for sluicing is recovered while the thickened slime residues, at a consistency of about 2 of water to 1 of solids, are allowed to run out to waste.



One of two Merrill 52 in. Precipitating Presses. Dome Mill.



Merrill Precipitating Press. Dome Mill.

Precipitation.

Zinc Dust Used.—The cyanide solution in which the gold has been dissolved flows from the Merrill slime presses in so clear a condition that no clarification is necessary before precipitation. The gold carried by this pregnant solution varies from $2.50 to $6.00 per ton of solution. This gold is precipitated from the solution and recovered in filter presses by the Merrill zinc dust precipitation process. An emulsion of zinc fume and cyanide solution, prepared in a miniature tube mill, is fed in definite amount at a uniform rate to the pregnant solution at the suction side of the Aldrich triplex plunger pumps; these force the solution through precipitation presses in which the gold is trapped in the form of a black mud that has a value of about $40.00 per lb. Two triangular frame 52-in. Merrill precipitation presses are used. One has twenty frames while the other has ten frames—the frames of both presses being 2 inches in thickness. The cyanide solution flowing from the presses carries from a trace to 8 cents in gold, and is returned to the top of the mill for use in washing dissolved gold from the slime filter charges. A certain amount of solution has to be run out to waste from time to time: this solution is carefully precipitated so that its value may not exceed 3 cents per ton. The amount of this solution run to waste is the difference between the number of tons of water entering the system at the treatment tanks with the unthickened slime, and the number of tons of solution retained by the cakes of treated slime that are discharged from the filter presses.

In precipitating gold solutions with zinc dust, more zinc is used immediately after the clean-up of precipitate from a press than at any other time, regardless of the grade of pregnant solution to be treated. The practice at the Dome mill is to feed the zinc dust as follows:

For the first hour after a clean-up, ¾ lb. per ton of solution; for balance of first day after clean-up, ¼ lb. per ton; for second day after clean-up, 1.5 lb. per ton of solution; for third day and until following clean-up, 1.6 lb. per ton of solution.

The average consumption of zinc per ton of ore treated is about 0.37 lb.

Melting Gold Precipitate and Retorting Amalgam.

A separate building, made of brick and situated behind the mill, is used for the melting room and assay offices. The melting room contains an amalgam retort, an acid treatment tank, a briquetting press, a lead smelting furnace, cupel furnace, a tilting oil-fired melting furnace, and a clean-up barrel for pulverizing litharge and cupel bottoms.

The gold from retorted amalgam is melted in the tilting furnace into bars of 990 to 995 total fineness, 890 of which is gold and about 100 silver. The melting of retort gold is done twice every month.

The precipitate, after being acid-treated to remove the large excess of zinc, is dried and then mixed with litharge, borax cake, and a small amount of fuel oil for binding the materials together when the mixture is pressed into briquettes. These briquettes are smelted in the lead furnace and the resulting lead cupelled. The slag, which assays from $40 to $100 to the ton, is re-melted in a later so-called "scavenger" run, together with cupel crusts and scrap metal, during which the slag tapped from the furnace carries only about $3.00 per ton in gold.

The cupel is oil-fired and has a bottom made of a mixture of one part Portland cement with three parts crushed limestone. The operation of cupelling is carried to the point at which about 25 one-thousandths of lead still remain with the precious metals. At this point the metal is brittle when cold, and may be easily broken into pieces for charging the tilting furnace, which melts the gold into bars for shipment. If cupelled beyond this point, however, the gold is tough and has to be divided into sections of convenient size, while molten, by means of iron dividing strips.

Without treating the precipitate by sulphuric acid before smelting, bullion of 900 to 950 fineness is produced, but, as the New York assay office objects to the presence of even 40 one-thousandths of zinc in the bullion, the acid treatment was rendered necessary and it is found that its cost is fully offset by the saving in fluxes and fuel, while the fineness of the bullion can be raised to 990.

Sampling and Weighing.

Daily samples are taken in the mill at the following points: (1) The broken ore delivered to the battery bin; (2) the pulp discharged from the batteries which is called the lip sample; (3) the slime leaving the amalgamation plates which is called "amalgamation tailings" sample; (4) the residues after filter pressing; (5) the sluicing water sent to waste with the residues; (6) the pregnant and barren solutions.

The average assay of the ore delivered to the mill during any month is calculated by dividing the sum of all the gold recovered in the plant, plus the gold lost in residues and sluicing water, by the tonnage treated during the month. The computed average is found to check reasonably well with the average of the battery lip samples but is usually quite at variance with the average of the broken ore samples.

The estimation of weight is calculated each day from a series of hourly measurements made on the flow of slime from the tube mill circuit to the slime thickening tanks. The entire flow of slime is diverted into a small steel measuring tank. The time interval is taken with a stop watch; the volume of the slime collected during this period is accurately measured and samples are carefully taken for the determination of the "solid to liquid ratio" before the slime has had any chance to settle. From these determinations a calculation, simplified by reduction to a formula in which the constants used have been incorporated, gives the rate of flow in tons of dry ore per day. Since this measurement deals with the tonnage of dilute slime, that is ten times the tonnage of dry ore being milled, errors of measurement are divided by ten when dry ore tonnage is deduced from it.

It may seem strange that after a careful consideration of the different methods and apparatus for weighing ore, such a one as the above should have been adopted at the Dome. However, a little consideration of the conditons here shows how unreliable would be the ordinary methods of weighing. Ore coming from the open cuts and delivered in cars to the crushing plant often contains from 5 to 8 per cent. of moisture, and car weights, even when corrected for moisture by taking a moisture sample, would be only as accurate as the number of tons of water estimated to be with the ore, while it is almost impossible to obtain such a moisture sample with any degree of accuracy. The same objection applies to the continuous automatic weighing devices of the Blake-Dennison type, in which a further source of error may be introduced by wet ore sticking to the conveyor belt. With proper care at each observation, the time-volume, specific-gravity determination of weight gives results in which the probable error is less than would occur when weighing wet ore, on either platform or belt conveyor scales, in cases where a large correction for moisture has to be made.

General Conclusions.

The metallurgical extraction obtained on ore of ½-oz. grade is found to be over 95 per cent., of which approximately one-half is made by amalgamation under the present conditions of tube milling and stamping through 10-mesh screens.

Judged by the experience at the Dome it would seem that on low-grade Porcupine gold ore, even when tube mill grinding follows stamping, the recovery by amalgamation alone cannot be carried much above 75 per cent.

The cyanide process is admirably adapted for obtaining a high recovery at low operating cost. It still remains to be determined, however, whether or not the cheaper cost of operating a partial sand-leaching cyanide plant would not offset the additional extraction obtained by reducing all of the ore to slime in tube mills, and then filter-pressing the treated slime so produced. Experiments along these lines are now being conducted in conjunction with experiments to determine the feasibility of leaching slime in the Merrill filter presses, as is done at the Homestake mine where exceptionally low-grade ore can be profitably treated, by reason of the cheapness of operation and the high recovery obtained.

Discussion.

Mr. H. A. Megraw: (Communication to the Secretary:)—Mr. D. L. H. Forbes' paper treating of the mill and process in use at the Dome mine, is an interesting addition to technical knowledge.

The matter of an adequate sampling and weighing system has not, to my knowledge, received any especial treatment at the Dome mill. This was one of the points to which I have called attention, and as I have devoted a special paper to the subject in general, it will not be necessary to go into detailed argument at this time. Evidently it has received the consideration of the Dome operators, as they are not likely to have overlooked so important a detail. It is probable that it is not considered economical to attempt exact sampling, the tailing assay from the filter presses showing the amount of loss and the exact percentage of recovery not being considered essential. Stamp crushing, followed by regrinding in tube mills, is the standard system used at the Dome mill, the only distinguishing feature being that the crushing and regrinding is carried out in water, while at most modern all-sliming mills it is done in solution to avoid the loss involved in changing from water to cyanide solution. The compelling reason in the case of the Dome is the necessity of using plate amalgamation for recovering as great a percentage of gold as possible before beginning cyanidation. The water crushing and plate amalgamation does not seem to be entirely justified at the Dome, but of course the question can only be settled by a comparison of costs. If amalgamation is considered necessary, there would seem to be no reason why it could not be carried out as well in cyanide solution, as is done at the Liberty Bell mill, for instance; or if its primary object is the prompt recovery of coarse gold particles which would unduly delay cyanidation, one has only to consider the system in use at the Hollinger mill, treating ore of similar character, where the pulp is concentrated and the concentrate treated separately by cyanidation. Experience at the Hollinger seems to show that coarse-gold particles entering the tube mills are likely to stay there until thoroughly ground up and, most likely, dissolved. I have already mentioned the remarkable efficiency of the tube mill as a cyaniding machine, due to a combination of advantages, among which may be mentioned its receiving freshly precipitated, barren solution, which is extremely active, the grinding and agitating action and the slightly higher temperature generated by the friction within the mill. The grinding action of the pebbles and quartz sand ought to be particularly effective in abrading particles of gold. Mr. Forbes does not mention this subject in his paper, although it has doubtless been considered.

The Dome mill, while its installation was carefully considered as circumstances would permit, embodies a more or less tentative scheme of treatment. The necessity of promptness and the small amount of underground development, as explained by Mr. Forbes, account for this condition, while personal conversation with Mr. Merrill has confirmed it, he having indicated that with the information gained by the operation of the present mill, further expansion might be made the occasion for a change of the treatment system. Current information shows that the treatment is, in fact, to be substantially changed. The information at my command at this time is, unfortunately, not official or complete, but it is known that 40 additional stamps are to be installed, doubling the present stamping capacity, and that six 40 x 10 ft. leaching tanks will be put into use.

Here, then, is a most significant and far-reaching change; adopting separate treatment in place of total sliming. Hard as the Dome ore is, this seems a most reasonable and beneficial change, as the smaller recovery by leaching, if indeed it is smaller, will probably be more than offset by avoiding the regrinding to slime. It is most probable that the sand in the battery pump will be partially reground in the tube mills, again separated, and the clean sand leached. Whether more tube mills will be used, I do not know, but it seems likely that the present installation will be sufficient to reduce the increased quantity of sand to a size appropriate for leaching.

Whether or not there will be any change of the water-crushing system, does not appear at this time. In view of the fact that Mr. Merrill has secured such uniformly satisfactory results by water crushing at the Homestake, it seems likely that he would prefer to keep to that system. I have already mentioned that the use of water in the crushing system is believed, in some cases, to have advantages other than the presumed amalgamation obtained. In some Western districts, notably at Goldfield and Tonopah, operators believe that when solution is used in the crushing system, chemical combinations prejudicial to good cyanidation are formed which would be avoided were water used. These combinations are said to destroy cyanide. At Goldfield, water is used in crushing, but at Tonopah, where silver ores are treated, the loss of cyanide, due to changing solutions and discarding excess, is believed to be greater than that destroyed when solution is used for crushing. At the Dome, however, where the ore is quite clean, one would not expect any detrimental results from solution crushing.

The very tall and narrow Pachuca tanks used at the Dome mill have always appeared to me expensive to operate, but they have been considered necessary in view of the extremely heavy nature of the pulp. With the change of treatment system, however, these tanks would seem to be made unnecessary, as only the true or natural slime will have to be agitated, the granular, sandy portion going to leaching tanks. A less expensive agitation system ought to be available under the changed conditions.

Whatever details I have discussed here are mentioned with the sole object of throwing more light upon cyanidation in general, using a well-considered, modern plant as a basis for argument. There is no room for doubt that the Dome mill has been successful in the past and will be more so in the future. Mr. Merrill and his associates are quite competent to conceive and produce a successful plant. Their handling of the metallurgy assures thoroughly practical and highly economical results.

Mr. D. L. H. Forbes: I desire to comment briefly upon Mr. Megraw's contribution to the discussion of my paper. The weighing and sampling of ore is a matter on which widely differing views are held. In all engineering measurements "Exactness" is unattainable, and it should be not so much a question of trying to obtain that which is beyond our reach as of obtaining results within limits of accuracy, yet close enough for the purpose to which the results are to be applied. It is true that there are automatic weighing machines which will weigh dry ore to within less than 1 per cent. error, but such a device is out of proportion and even inaccurate when applied to very wet ore coming from the mine. Besides the error introduced by wet ore sticking to the belt of such machines, there is an unavoidable large error in the estimate of moisture contained in the ore. There is no doubt that, in cases where ore coming from the mine is nearly dry, it is most economical and satisfactory to use either automatic continuous weighers of the Blake-Dennison type or registering car scales, but at the Dome mine conditions are such that neither of these methods was deemed advisable.

The matter of sampling ores going to the mill is also one that is a subject of much controversy. In general I am inclined to agree with Mr. Megraw's stand that it pays to sample the mill heads wherever it can be done with a reasonable degree of accuracy and without adding a heavy charge to the cost of milling. That it is possible to sample gold ores such as that of the Dome no one will deny, but an idea of the cost of obtaining results that are reliable can be gained by looking up the rates charged by customs sampling works. No manager who is not engaged in purchasing ores would care to add such heavy operating charges to his milling cost as this class of ore-sampling would involve; yet to install anything short of such methods would be wasting money when sampling ore that is lacking in uniformity of gold distribution.

Contrary to Mr. Megraw's impression, the losses of the mill used in calculating extraction are not alone those in the tailing, but every other possible source of loss is also taken into consideration and estimated as closely as practicable. All waste solution and sluicing water issuing from the mill are regularly sampled and measurements of volume taken.

Mr. Megraw is entirely right in stating that the question of water crushing as compared with crushing in solution is one that can only be settled by a comparison of costs. It is precisely on this basis in combination with the total recovery of gold that the argument for the Dome mill practice rests.

MONELL METHOD OF SEPARATING NICKEL AND COPPER SULPHIDES

In specification of Letters Patent Mr. Ambrose Monell, president of the International Nickel Co., describes the method of separating nickel and copper sulphides.

In the reduction of ores containing nickel and copper where a matte is produced containing sulphides of nickel, copper, and iron, a process has been devised in which a separation of the nickel sulphide is effected by the use of sodium sulphide, advantage being taken of its power of dissolving the sulphide of copper and iron freely and forming a solution of less specific gravity than the nickel sulphide. The matte mixed with coke and sulphate of sodium has been charged into a cupola-furnace. When this charge is smelted the sodium sulphate is reduced by the coke to sodium sulphide and, forming a solution with part of the copper sulphide and iron sulphide, flows with the undissolved and melted sulphides of nickel, copper, etc., through the tap-hole, which is kept constantly open, into moulds, where the molten constituents separate in accordance with their specific gravity, the sodium sulphide containing the dissolved copper and iron sulphides floating on the surface and the undissolved sulphides settling to the bottom. When the contents of the mould have solidified, the parts are separated by fracture and the tops containing the copper and iron are recharged into a smelting furnace, where the sodium sulphide is fluxed off in an iron slag, being then lost. The bottoms contain most of the nickel sulphide of the original matte; but owing to the imperfection of the separation they also contain so much copper sulphide and iron sulphide that it is necessary to resmelt them with fresh additions of coke and sodium sulphate, and thus to repeat the smelting and separation to the fourth or fifth time before the bottoms are brought to sufficient degree of freedom from iron and copper to enable the resultant nickel sulphide to be economically subjected to the succeeding steps of the refining process. The process as thus carried on is slow and wasteful and because of the cost of materials and the amount of labour and handling required adds greatly to the expense of the nickel or nickel oxide which is the final product. I have discovered that these difficulties can be overcome and the separation rendered quick and inexpensive by the following process.

Instead of smelting the compound matte, as heretofore, in a cupola-furnace and running the product continuously into moulds I so smelt the matte that when melted it will remain in a molten state subject to the high temperature of a furnace for a considerable period of time, during which I find that the copper and iron sulphides will be thoroughly dissolved by the sodium sulphide, and in one melting a good separation can be effected, and by two such treatments results are obtained equal or superior to the results of the four or five meltings which have been employed heretofore. For this purpose I employ as the smelting-furnace an open-hearth reverberatory furnace lined with magnesite brick, as I find that silica-lined furnaces are quickly destroyed by fluxing with the sodium sulphide. Into such furnaces I introduce a charge of nickel-copper-iron matte, either solid or molten, together with coke and sodium sulphate, the latter being preferably present in the proportion of sixty per cent. of the weight of the matte and the coke in the proportion of fifteen per cent. of the matte. The sulphate is preferably added in the form of commercial niter-cake. Where, for example, a fifty-ton charge of matte is treated containing, say forty-five per cent. of nickel sulphide and thirty-five per cent. of copper sulphide, it is melted in the furnace and retained subject to the heat for some time—say four to five hours after fusion has occurred—during which time it is preferably "poled"—that is to say, treated by immersing beneath its surface poles of green wood, which evolve hydrocarbon gases and vapours, and thus aid in the reduction of the sulphate and produce an agitation of the material, which facilitates and renders more thorough the solution of the sulphides to be removed. Nearly complete solution of the copper and iron sulphides in the sodium sulphides reduced from the niter-cake is thus effected, and the molten charge may be tapped from the furnace and allowed to separate in moulds; but to get the best results I tap the different strata from the furnace separately, tapping first the solution of copper and iron sulphides floating on the surface of the bath and finally tapping the undissolved nickel sulphide, or the order of tapping may be reversed, the lower stratum of nickel sulphide being removed first. The great proportion of the iron and copper is thus separated, the nickel sulphide obtained being nearly pure. Where greater purity is desired, the nickel sulphide may be recharged into the furnace and treated again in like manner.

The skilled metallurgist will be able to modify the apparatus and also to use other solvent materials. For example, sodium sulphide may be charged into the furnace instead of sodium sulphate, in which case, as no reduction is required, the coke may be omitted or a less quantity of it employed, and even when sodium sulphate is used deoxidation may be performed by the operation of poling without the use of coke or with only a little coke.

Instead of sodium sulphide I may employ the sulphides of other alkaline metals or sulphide of manganese. —*Nickel Industry, by A. P. Coleman.*

ROCK DRILLS*

By W. L. Saunders.

The drills now in use may be classified as follows:

1. The plugger drill. This is of the hammer type. It is used in its smallest sizes for dressing stone, for trimming, cutting hitches, and for block holes. It is a hand-rotated machine.

2. The Jackhammer. A hammer drill with automatic rotation, used for sinking shafts, for down-hole work in stopes, for quarrying, for drilling in coal, and in rock and ore work wherever down-holes are required. It is held in the hand of the operator and in some mines is used for horizontal work, mounted upon some simple form of support.

3. The stoper. A hammer drill with air feed, usually used without mounting and for up-holes. It has a large field of usefulness, mainly in stopes or rooms and in driving raises.

4. The mounted hammer drill, as exemplified in the Leyner type, used mainly for horizontal, or approximately horizontal, hole drilling, for side stoping, and for driving drifts and tunnels. In this type of drill a combined stream of air and water is discharged through a hollow steel at the bottom of the drill hole.

5. The reciprocating drill, used for heavy down-hole drilling where the stopes are large, and for surface work, drilling deep holes of large diameter.

Plugger drills are largely used for pop shooting, breaking up boulders, trimming walls, and for all light work requiring holes not exceeding 5 ft. in depth. Solid or hollow steel may be used. In the latter case the exhaust air is discharged at the bottom of the hole for the purpose of removing the cuttings. This type of machine is equipped with a simple device for cleaning the holes of rock cuttings which may not have been discharged by the normal process of exhausting in the hole. This device consists of a lever, eccentric in its middle portion and located on the side of the valve chest. A portion of the lever forces a plunger against the valve, stopping the action of both the valve and the hammer piston and thus permitting live air to pass to the bottom of the hole. Where steam is used wooden handles replace the metal ones.

The Jackhammer can be operated either by steam or air, it uses hollow steels, is provided with an automatic rotation and it drills holes to a depth of 15 ft. It is essentially an all-steel drill. The cylinder, for instance, is drop forged, made of special steel, treated and hardened in the bore. Its most interesting feature is its automatic rotation.

The stoper hammer drill is so well known and so generally used that further description is unnecessary. It is but a few years since this machine was invented and its application to general mining work has been very rapid. Thousands of them are in use.

Jackhammer Records.

There is no place where this new type of Jackhammer drill shows to better advantage than in shaft sinking and in bench or foot-wall work in mines and tunnels. For shaft work two or three times the number of drills may be used as formerly. Each man in the shaft who is not engaged in mucking is a driller, the two operations being carried on simultaneously.

At the Newport mine in the Lake Superior iron country a shaft is being driven with Jackhammers. Dimensions of shaft, 11 by 18 ft. in the clear. The progress has averaged 20 ft. per week, using five drills; material, hard quartzite. During February last a progress of 107 ft. was reported, and in March 125 ft., this latter being said to be the record of performance in shaft sinking in that district.

At the Lucky Star mine in the Lake Superior district four Jackhammers are working in a shaft 12 ft. 2 in. by 14 ft. 10 in., drilling ten 6-ft. holes for sinking and twenty 5-ft. holes for squaring after blasting; material, hard diorite. The progress has averaged about 100 ft. per month.

The Norrie-Aurora shaft of the Oliver Mining Co. was sunk 64 ft. in February, employing three 8-hr. shifts of eight men, each using Jackhammer drills, the shaft being sunk, steel timbered and concrete lined. The total hours drilling was 794 man-hours, out of a total of 4,202 man-hours. In this performance the man-hours required for mucking were almost twice that of the drilling, whereas the time charged for blasing approximated that of the drilling; the timbering and lathing combined were slightly under the drilling figure. The remainder of the time was charged to squaring, cutting hitches, piping, etc. During March 105 ft. of shaft were driven and

*Extract from paper presented at Butte Meeting, A.I.M.E., August, 1913.

completed. This included not alone the sinking of the shaft, but the heavy steel sets were placed in position, the concrete slab-lathing was put in place for the entire distance, also the back-runners. The magnitude of the work may be better understood when it is realized that 24,600 cu. ft. of rock were broken and hoisted to the surface, and the labour of placing the shaft equipment, consisting of two skip roads, one cage road, ladderway, pipe compartment, counter balance and back-runners, completing a modern five-compartment shaft, was all performed in 26 days. Each piece of concrete slab-lathing weighed 130 lbs. and the steel sets were made of heavy material. Six Jackhammers were used to sink the shaft.

The East Butte Copper Mining Co. is sinking a shaft through a hard granite formation with two Jackhammers, from the 1,200-ft. level to the 1,800-ft. level, and reports show that from Jan. 20 to Feb. 15 of the present year 110 ft. of shaft were driven, including timbering. During this period they lost 11 shifts due to outside causes, no work whatever being done. Here an average of 28 holes was drilled per shift, 8 cuts 9 ft. deep, 8 lifters 6 ft. deep, 6 back holes 6 ft. deep and 6 end holes 6 ft. deep. Occasionally as high as 33 holes were drilled. The average net drilling time was 5 hrs. This gives a total hole footage per shift of 192 ft., or 96 ft. per drill, an average of 19.5 ft. per drill per hour. The previous best progress, made with heavy tripod drills, was 60 ft. in one month. Size of shaft, 19 ft. 6 in. by 6 ft. 10 in.

J. A. McIlwee, the contractor who drove the Laramie-Poudre tunnel, is sinking a shaft for the Silver King Consolidated mine, Park City, Utah. This is a three-compartment shaft and is being driven a distance of 500 ft. from the 1,300-ft. to the 1,800-ft. level. The work began March 22, with four Jackhammer drills, two on each side of the shaft; four drillers were employed. A great many delays and drawbacks were experienced, mainly on account of water. Some boiler difficulties were incurred, which caused a delay of four days in April. Notwithstanding this, 105 ft. of actual sinking or 95 ft. of completed and timbered shaft have been accomplished. Based on the number of days actually at work sinking, this is a progress of about 5 ft. per day. Holes were drilled 5 ft. deep, the usual time employed being 10.5 min. per hole. The best time made in drilling was 33 holes 5 ft. deep, with 6-ft. cut holes, in 2 hrs. 45 min.; three shifts, each shift gang doing the drilling, mucking, and timbering.

Bench or foot-wall work has heretofore been done with large drills, employing from two to three men per drill. Deep holes of large diameter have been drilled, but during recent years the hammer type, both in the stoper and in the Jackhammer, has replaced the heavier machines for this class of work. In one of the Lake Superior copper mines recent extensive tests have shown the following figures:

Type of Drill	Tons per man per shift	Mining Cost per ton.
Stoper	13	$0.41
Piston	12	0.54
Jackhammer	38	0.12

No overhead charges are included in the above figures of cost, nor are charges for air included. This effective saving by the use of stoper drills over the heavier tripod type, and the still further saving by the use of Jackhammers, cannot be maintained in all classes of work. In this particular case the bench or foot-wall afforded the best opportunity for the use of these little Jackhammer drills.

An analysis of the drill situation, in the iron fields of Lake Superior, indicates that four types of drills are necessary for the economical extraction of the ore, their relative proportion being about as follows:

	Per cent.
1. Mounted hammer drill	20
2. Stope drills	20
3. Jackhammer or sinking drills	10
4. Light piston drill	50

There will always remain a small percentage of drilling for which the two-man piston drill will still be the favorite. An instance is the operation of the Soudan mine at Tower, where the rock encountered is very hard; in fact, it is characterized as the hardest in the United States.

At this time the ore is very hard and dry, and the chief problem is not so much one to be met by the use of some particular type of drill, but rather one of getting the steel to stand, as the bits will not hold in this rock. Records show that as many as 80 starters have been used, with a 3.25-in. machine, to drill a hole 6 in. in depth. Light one-man drills have been tried in this formation, but without success.

In the hematite mines of Lake Superior the light reciprocating mining drill is generally used for the actual breaking of probably 90 per cent. of the ore which requires drilling (hand-auger ground excepted). In this class of work the service required of the drill is light, the ground being classed as very soft drilling, but of such a character as to bring it under the classification of "too hard drilling" for hand augers.

In the city of Pittsburgh, Jackhammers are being employed for driving an 8 by 12 ft. sewer tunnel, 1,200 ft. long. The formation is a hard slate and shale with a decomposed rock top. Single cap timbers are installed and it is necessary to use fore-poling to hold the top. Two drills are installed in the heading, drilling eight 3-ft. holes, with a single steel to the hole. The average drilling speed is 12 in. per minute. Owing to the bad top and the fact that the tunnel passes under two bridge piers and the Pennsylvania Railroad tracks, the shooting is necessarily light; about 2.5 ft. are pulled at each round. Ten feet of completely timbered heading is the advance per day of two shifts.

The following is a record of work done with Jackhammers in the excavation for the Grand Central Station, New York City:

Month.	Hours drilled.	Total linear ft. drilled.	Average ft. drilled per hr.
Sept. 22 to 29	60	950	15 5/6
Oct. 1 to 31	255	4,256	16.65
Nov. 1 to 5	48	800	16 2/3
	363	6,006	16.5

Cost for repairs (2 pawls, 50c. each), $1, or about 1 2/3c. per 100 ft. drilled.

Air pressure, 70 to 90 lbs.

Holes drilled, 4 to 10 ft. in hard New York mica schist.

In the lead mines in southeast Missouri they have heretofore drilled the down-holes in the stopes with the large type of two-man drill. Since the Jackhammer has been introduced in these mines the results obtained have been 60 to 80 ft. of 8-ft. holes per man per shift, while with the old system employing two men they were only able to drill 40 to 50 ft. of 8-ft. holes per shift. In many cases where the vein is exceedingly wide they drill 10-ft. holes as easily and readily as an 8-ft. hole can be drilled.

In the Oliver iron mines in Michigan the record is given as 57 stoper drills operated in the year 1912, with a total expense for repairs for the year of $920.23, or $16.14 per machine per year.

THE PERSISTENCE OF GOLD ORE IN DEPTH*

By Malcolm Maclaren.

A study of the recent literature of ore deposits inevitably forces the conclusion that workers in this branch of geology are endeavouring, in their zeal for the advancement of knowledge, to wrest from the scanty data available more than the simple facts warrant. Data that have been garnered from the examination of a given metalliferous deposit, and that have a real value when applied to the construction of a sound theory of deposition for that metal alone, have been transferred to stay and brace the tottering structure built for another metal, with which the first may eventually prove to have only the slightest genetic affinity, however closely allied they may appear to be to-day. When all the known facts concerning the deposition of any one metal have been collected, collated and analyzed, then, and not until then, may comparisons be made with the data of another metal similarly treated. Some metals—e. g., tin and copper, clearly lie so far apart genetically that no confusion of data has resulted, but the general impression appears to be that the data concerning the ores of other metals are interchangeable. They may often indeed be so, but the time has not yet arrived when transfers may be made with safety. For this reason, therefore, the ores of one metal only, viz., gold, are considered in the following brief review.

earth's surface. The ore in this zone may be assumed to have been deposited during a single short geological era, and to owe nothing to accretions of a widely separated and later period. It is probable that the irruption of auriferous solutions was normally paroxysmal in character and indeed was comparable to volcanic eruptions of the present day. Only those fissures and channels that were open at the geological moment, so to speak, were filled with ore. An assumption of this nature may help to explain the vertical variations in the tenor of the ore in the primary zone, where in many mines, horizontal bands of richer and poorer ore alternate. These alternations conceivably represent the varying horizons at which successive upward pulsations of metalliferous solutions either became sufficiently cool to be deposited or met with fluid agents of deposition.

I have elsewhere* attempted to show that the gold deposits of the world fall naturally into well-defined auriferous groups, the members of each group, though widely separated in space, being closely allied in genesis, in character, and in geological age. One of the most important distinguishing characters of the several groups is persistence (or otherwise) of ore in depth. The classification adopted must therefore be outlined.

Classification of the Gold Deposits of the World.

(a) Arising as the end-product (generally following albite-porphyry) of diabasic magmatic differentiation series intrusive into schists.	Pre-Cambrian	Western Australia (Kalgoorlie, etc.), India (Kolar, Hutti, and Dharwar) Rhodesia, Transvaal (Witwatersrand, Pilgrim's Rest, Barberton), Brazil (?), Guianas, Appalachian fields, Eastern Canada (Porcupine, etc.).
	?	(a) Urals.
(b) Arising as the end-product of Granodioritic intrusions.	Permo-Carboniferous to Post-Jurassic	(b) Eastern Australia and Tasmania.
	Middle Tertiary	(c) Western North America (California, Oregon and Alaska).
(c) Associated with andesite volcanic eruptions.		Northern Chili, Peru, Colombia, Mexico, California (Bodie), Nevada, Utah, Colorado, Unalaska, Japan, Sumatra, Celebes, New Zealand, Transylvania.

As a preliminary, a definition of "depth" appears to be necessary. Here it is assumed to cover only the (presumably) primary ore that lies beneath the zones of secondary enrichment (oxide and sulphide) and to extend for a limited depth, say 1,000 feet, below the bottom of the deepest mines, or 6,000 feet in all. To take the enquiry deeper is to enter the barren zone of speculation. The combined depths of the oxide and sulphide zones of enrichment may vary with climatic conditions from a few feet to a few hundred feet. Ordinarily, below 500 feet we are, for most gold ore bodies, in primary zone. A word may be said in regard to the use of the term "primary ore." It comprises that ore for which we know no prior state of combination and no former locus in space. In this review then, depth is understood to be a zone extending downwards from 500 feet to 6,000 feet below the

The above table varies slightly from that originally adopted, but four years' further field experience has enabled me to abolish the former tentative subdivision of the Pre-Cambrian deposits and has given a much clearer view of the general sequence of events leading up to auriferous deposition in that age. These groups, therefore, contain all the important gold deposits of the world. Two of these, viz., the Pre-Cambrian and the Tertiary are extremely well defined; the third, including all apparently dependent granodioritic magmas, is still somewhat indefinite and will certainly be modified with increase of knowledge. Probably, when the exact age of its auriferous impregnation is known, the Ural chain of deposits will be brought into close accordance with the Eastern Australian, while the Californian (Mother Lode, etc.) occurrences may eventually be transferred to the Ter-

***A paper contributed to proceedings of the Twelfth International Geological Congress, Canada, 1913.**
***Maclaren, Gold, London, 1908, pp. 42-75.**

tiary andesitic group, with which they are indeed orographically closely connected.

In any consideration of the question of the persistence of gold ore in depth the foregoing divisions must be kept closely in mind, since the recurrence of the evidence of the complete dependence of gold deposits on geological conditions, both for deposition and for extension, lateral and vertical, is certainly the most salient feature arising from the study of the goldfields of the world.

The Younger Goldfields.

It will probably be most convenient to first consider the younger goldfields. These are the andesitic fields that have furnished some of the greatest bonanzas that have been known. Their petrological range is from pyroxene-andesite to quartz-trachyte, and occasionally to rhyolite, all apparently the differentiation members of dacitic magma. Their geological range is from Eocene to Pliocene with a special development in the Oligocene and Miocene. With one notable exception they follow very closely and are confined to the so-called "Pacific Circle of Fire," with which line of volcanic activity they have clearly a very close genetic connection. The outstanding feature of gold deposition in this group is its modernity and its consequent intimate association with existing volcanic phenomena. The geographical exception is the Transylvanian goldfield of Hungary, the andesites of which were erupted during the Aquitanian stage, and along lines of crustal weakness initiated in the Oligocene and indicated at the present day by the active volcanoes of the Mediterranean.

Auriferous deposition in this group has probably been closely associated with solfataric action. Maclaurin has indeed shown that the hot springs of the solfataric region of New Zealand at the present day bring to the surface and deposit notable quantities of gold and silver in the siliceous sinter that forms on the edges of the boiling springs. A similar deposit is recorded from near the De Lamar mine, Idaho. The New Zealand hydrothermal region is on the same line of crustal weakness as the goldfields of the Hauraki peninsula. On it, only 40 miles away from Rotorua, is the famous Waihi mine, until three years ago one of the greatest of the world's gold mines. The chalcedonic character of the siliceous filling of the veins of many andesite fields also appears to point to a deposition from hot waters. In andesitic and allied rocks in the neighborhood of auriferous veins "propylitization" is universal. In this facies of the original andesite rock the feldspars and ferro-magnesian silicates have been converted to quartz, sericite, calcite, epidote, chlorite, serpentine and pyrite.

The outstanding feature of auriferous ore bodies in andesitic fields is their general irregularity, both in form and in tenor. The great persistent fault fissures so often found in older and deeper seated rocks are unknown, or, at any rate, have not served as loci of deposition. There is nothing in any andesitic field comparable, e.g., with the Mother Lode fracture of California.

It is, of course, conceivable that strong fault fissures could readily have been formed, but it is improbable that in any active volcanic and solfataric region, such fault fissures would remain open for any length when large quantities of cementing igneous and aqueous matter were being brought to the surface along the assumed line of weakness. The Comstock Lode with a total length of two and a half miles is probably the longest fault-fissure lode of economic importance in the andesitic fields. Normally the fissures of andesitic fields appear to be local tension fractures due sometimes to cooling and sometimes to minor local movements. They are therefore limited both in linear and in vertical extension, falling into the group of "gash veins" of an old nomenclature. When two or more local series of fractures intersect, the "stockwerk" so characteristic of many New Zealand and Transylvanian areas results.

Where the veins of the stockwerk are sufficiently close together a great bonanza may result as in the case of the Shotover and Caledonian mines, Thames, New Zealand. The original irregularity of the andesitic fissures is greatly accentuated by the selective action of auriferous solutions that replace the fissure walls with ore.

No andesitic field has as yet carried its bonanzas to great depths. By far the deepest is the Comstock, where shafts were sunk to 3,300 feet, but though ore was found erratically distributed through the lower workings, it was in nowise comparable to the great bonanzas that occurred between the 1,000 ft. and 1,800 ft. levels. Only a very few mines in andesitic regions have carried rich ore below 1,000 feet, and the characteristic feature of even these is uncertainty of persistence in depth. For the lack of persistence a definite reason may very often be given, viz., the change along the downward course of the lode from dacite or andesite to the underlying basement rock, or, in rarer cases, to a member of the andesitic differentiation series unfavourable to gold deposition. Often the mere approach to the basement rock connotes impoverishment of lodes. Instances are numerous, e.g., in New Zealand (at Coromandel and Thames), in Colorado (at Cripple Creek and Telluride), in Transylvania (at Vulkoj, Korabia, and Nagyag); but there are many lodes that persist in a homogeneous rock, which may be either a member of the andesitic differentiation series or may form a member of the basement complex through which andesites have burst, and that nevertheless show a marked diminution in value at comparatively shallow depths, often less than 500 feet. For some of these impoverishments a physical cause may be advanced, viz., approach to the bottom of the fissure of tension; but for others, indeed for the majority, no such explanation is possible. For example, the Comstock fissure is well defined as far as it has been followed downward. The great Martha lode-fissure (Waihi) persists as strongly as ever below 1,000 feet, but whereas above that level the gangue was mainly quartz, below it the matrix is calcite. The Martha Lode appears to have been originally wholly a calcite lode that was attacked by solfataric waters above 1,000 feet, silica with accompanying gold almost completely replacing the calcite. Either, then, 1,000 feet below the surface marks the horizon at which solfataric waters become active agents of solution and deposition, or, and more probably, the percolating waters had no access to a zone of the lode immediately below the 1,000 foot level. Whether at greater depths they used the lode-fissure as a channel and replaced its calcite gangue future exploration alone can show. Here, as has so often been the case, the solution of the question of the persistence of ore in depth depends on economic considerations.

The impoverishment of the veins of the andesitic goldfields in depth is a feature so universal that a general cause for diminution in value must be sought. I have attempted to show elsewhere that the probable form in which gold travels in solution, in depth at least, is not as the chloride, but as an alkaline aurosulphide, and that pyrite and other sulphides are not the natural precipitants in depth, but that precipitation may be due to a more general cause as cooling of uprising solutions.* Recently Lenher,† to whose laboratory researches field workers are deeply indebted, has shown that the alkaline sulphide solutions are highly efficient carriers of gold, that pyrites has no effect on their contained gold and that their gold is readily precipitated by acid waters or by exposure to oxidation. Both these agents may reasonably be assumed to operate only near the surface, especially in volcanic regions. Maclaurin found that the waters of the acid lake on White Island, New Zealand, contained 5.47 per cent. of free hydrochloric acid. Little proof of the existence and wide distribution of acid waters at the earth's surface in solfataric regions is, however, necessary. While the former cause probably operates directly in andesitic regions proper, deposition of gold in the numerous cases in Colorado, Nevada, Translyvania, etc., in which the gold-quartz veins lie in older sedimentary or plutonic rocks, is more likely to be due to oxidizing waters, the influence of which naturally reaches only a short distance beneath the earth's surface.

Recognition of the irregularity and lack of persistence of auriferous ore bodies in andesitic fields is of prime importance to the mining engineer. For such ore bodies not a single ton of ore more than has been actually proved may be assumed.

The Granodiorite Goldfields.

The second group of the classification already outlined includes those goldfields that are apparently genetically connected with granodioritic or closely allied magmas and that occur as a product of their differentiation. This group contains three geographical provinces, viz., Eastern Australia, California, Alaska and the Urals. For the purpose of the present paper they may also be divided simply into (a) lodes in granodiorite and allied rock and (b) lodes in the sedimentary complex through which the granodiorite is intrusive. The relations of the former are simple. Those of the latter are greatly complicated, from the present point of view, by changes in tenor likely to take place when lodes pass in depth from one member of the complex to another.

The deposits of the Eastern Cordillera of Australia are initially dependent on great granodioritic intrusions that have taken place along an axial line of earth folding. Gold-quartz veins may occur either in the igneous rock itself or in the sedimentary strata overlying or adjacent. The habitus of the gold deposits in the north is, in the main, in the granitoid rock; while in the south gold-quartz veins are generally found in sedimentary rocks. Important exceptions to both rules occur and are of special value as evidences of the general genetic connection between the gold deposits of the north and the south, respectively. The general age of the plutonic intrusion is probably late Permo-Carboniferous. All adjacent strata of greater age may therefore carry auriferous veins. Charters Towers in North Queensland, with a production of nearly £29,000,000, is the most important field in the granitoid rocks. Its igneous complex comprises rocks ranging from grey hornblende granite to tonalite, the latter being the predominant rock. The two principal lodes are the Brilliant and the Day Dawn, which have been worked to depths of 2,500 to 2,700 feet. On the whole, the ore has shown a gradual though small diminution of tenor in depth. Similar fields are those of Croydon (Queensland), and Wyalong (New South Wales); neither furnishes any evidence bearing on the point in question.

Considering the number and great importance of the goldfields of Eastern Australia developed in sedimentary rocks the light thrown by them on the general question of persistence of ore bodies in depth is singularly little. Certainly some, as Gympie (Queensland) and Ballarat (Victoria), depend for auriferous deposition on the intersection of lodes or quartz veins and graphitic bands in sedimentary strata, a condition which is not necessarily recurrent at depth. Others, including the majority of the important Victorian fields and the Hargraves field in New South Wales, are developed in tension fractures between unlike beds at the crests of anticlines forming, e.g., the famous "saddle reefs" of Bendigo and Castlemaine. In these fields saddle reefs are successively met with in depth when sinking on an anticlinal axis, so that a condition ensues very different from that met with when considering the filling in depth of a single continuous fissure. But the experience gained on these formations all tends to show that the lower saddles are not nearly so rich or so large as those above. At Bendigo mining operations were carried to 4,614 feet below the surface in the New Chum mine, but it is very probable that, taken as a whole, work on the Bendigo field below 2,500 feet has not been profitable. Certainly the tenor of the ore has decreased in depth.

In the California-Alaska belt of gold lodes, which are apparently dependent on granodiorite magmas, the various Mother Lode mines and the Alaska-Treadwell group in south-eastern Alaska are the most important. The latter are still shallow and are of no help in the present discussion. Many of the Mother Lode mines, especially in Amador county, are nearly 2,000 feet deep, and some, as Kennedy (3,650 feet), Gwin (2,650 feet), and South Eureka (2,850 feet), have reached much greater depths.

The Mother Lode is a fissure zone that may be traced from Bridgeport in Mariposa county to near the northern boundary of Eldorado county, a distance of 120 miles. In many places it is a solid lode 100 feet wide, but often it is merely a shattered zone in which numerous quartz stringers are developed. It is undoubtedly due to major faulting developed along a line parallel with the axis of the Sierras during the uplift of those mountains. The faulting has selected the softest beds (Mariposa slates) of the sediments and has uplifted them for great distances.

So far as my three months' examination of the Mother Lode permitted, I have not been able to make out any appreciable diminution in tenor in depth. Many mines have certainly "bottomed" the ore in given fissures at depths less than 2,000 feet, but it often happens that two or more parallel lodes occur within the Mariposa slates, and that when one becomes barren a hanging wall or footwall lode may carry ore to much greater depths. In few auriferous regions is cross-cutting from wall to wall of the lode channel more necessary; in few has less been done than along the Mother Lode. The mines of Angels Camp are often instanced as evidence of the occasional non-persistence of Mother Lode mines in depth, but, assuming for the moment that no ore occurs there in depth, their evidence cannot be admitted against Mother Lode mines. They are, it is true, on the line of the Mother Lode fissure zone, but from the Hardenburg mines, south of Jackson, to near the Rawhide mine, south of Tuttletown, the Mother Lode fissure zone, keeping a straight course, leaves the Mariposa slates, which curve to the west through the Gwin mine and

*Maclaren, Gold, London, 1908, pp. 38, 78, etc.
†Econ. Geol., Vol. VII., 1912, p. 744.

run parallel for many miles before rejoining the fissure zone south of Stanislaus River. One of the factors (viz., the presence of carbonaceous slates) that makes Mother Lode mine is therefore lacking in Angels Camp.

While, therefore, any given fissure of the Mother Lode series may cease to yield ore in depth, it is probable that ore will be found at greater depth in another adjacent member. Finally, when broadly considered, the Mother Lode may, with unchanged geological conditions, be expected to carry ore with undiminished tenor to and perhaps beyond the limit of "depth" set forth in this paper.

No evidence of value is to be derived from a study of the gold veins of the Urals. They are nearly all small and irregular and no deep mining has been done on them.

Reviewing the scanty evidence furnished by the granodioritic group, we find for Eastern Australia a gradual though small diminution of the tenor of ore bodies in depth, while on the Mother Lode all the evidence points towards a general persistence in depth for typical Mother Lode mines. A mining engineer, dealing with the future of these mines, would not, therefore, unless he had evidence of an approaching change in geological conditions, be justified in disregarding all ore except that "in sight"; some might be expected to occur below the deepest present explorations, and such ore should always be taken into economic consideration.

Pre-Cambrian Goldfields.

The third group of the classification includes all Pre-Cambrian goldfields and comprises the most important now being worked. These lie in two geographical areas, one on the borders of the Indian Ocean, ranging from Western Australia through Southern India and Egypt to Rhodesia and the Transvaal, and the other along the eastern side of America from Eastern Canada through the Appalachian chain and the Guianas to Brazil and Tierra del Fuego. The former is a very well defined group of goldfields that, though geographically widely separated, present so many points of similarity that a geological description of the various rocks and of their internal relations in any given region would serve, with the mere change of place names, for any other region of the group. The members are consequently believed to form a single petrological and metallogenetic province, for which the appellation Erythræan* has been suggested.

A typical Pre-Cambrian field is that of Kalgoorlie in Western Australia. Its total gold production has been more than forty millions sterling. It has been closely studied by the writer and throws considerable light on the general question of auriferous deposition in Pre-Cambrian rocks and on the persistence of ore in depth in those rocks. Briefly, the area is one of ancient schists (mainly calc-schist) through which a quartz-dolerite magma with its differentiation products has intruded. The differentiation sequence appears to have been quartz-dolerite (quartz-diabase) followed first by members as basic as peridotite and then by more acid segregations ranging through porphyrite to final albite-porphyry, the last being often intrusive through the quartz-dolerite. Auriferous impregnation followed closely on the intrusion of the albite-porphyry. Rich lodes have been developed only in shear-zones in a broad dike of quartz-dolerite, the shear-zones being barren when they pass in depth or in linear extension out of the quartz-dolerite. Since the shear-zones are, when considered over depths of 3,000 feet, approximately vertical, and the quartz-dolerite dike, which is parallel to the strike of the shear-zone dips west at 65 degrees, the shear-zones pass in depth out of the dike, the eastern shear-zones with their contained lodes reaching barren ground sooner than the western (see plan and section). Kalgoorlie, therefore, well illustrates an outstanding feature of all goldfields, except indeed some in the Tertiary group, viz., that non-persistence of ore in depth is a function not of depth but of geological structure. In Kalgoorlie three well defined parallel shear-zones may be made out. Taken severally and having regard to the depth factor alone, they show (a) non-persistence of ore in depth (Australia East and Lake View-Perseverance lodes), (b) persistence of ore in depth (Great Boulder and Ivanhoe-Horseshoe lodes), and (c) a probable enrichment in depth (Ivanhoe West lode). Generalizations based on the depth factor alone when geological conditions are unknown are misleading. Rickard,† for example, has relied on the evidence furnished by the failure in depth of the eastern lodes and an impoverishment in the Ivanhoe mine at 2,500 feet to support a general theory of impoverishment in depth. Garrison‡ also quotes the Ivanhoe impoverishment as possible evidence of non-persistence in depth. The Ivanhoe impoverishment does take place, but it is local and is due to the fact that the vertical Ivanhoe lode here passes through a thin albite-porphyry dike dipping west about 65 degrees. The great Boulder lode passed through the same dike with local impoverishment at 2,200 feet. When, however, the latter lode was encountered beneath the albite-porphyry dike it proved as rich as in upper levels, and the same result may reasonably be expected in Ivanhoe deeper levels. So far, then, as the evidence furnished by Kalgoorlie goes, it indicates that, so long as its lodes remain in quartz-dolerite, so long will they furnish ore equal in tenor to that found from the 500 to the 2,000 ft. levels. The Horseshoe-Ivanhoe group of lodes may therefore be expected to carry ore to the 5,500 ft. level, provided always that the quartz-dolerite dike persists, does not flatten in dip, and is not thrown westward in depth by westerly dipping reversed faulting.

Archean strata, from the vicissitudes to which they have been subjected in the course of long geological ages, are normally much folded and disturbed, while lode fissures in them are nearly vertical. It is a fundamental axiom in these older deposits that the nature of the lode wall exercises a vital influence on the richness and sometimes on the mineral character of the ore body. Hence it rarely happens that a great depth is reached before the lode, worked from the outcrop downward, has passed out of the favourable rock. A notable exception is the Champion Reef of the Kolar goldfield, southern India, probably the richest single gold lode ever worked. From 3,200 to 3,800 feet ore as rich as any obtained in the upper levels is now being worked and ore may be expected to persist in this fissure as long as it remains in the favourable hornblende-schist.

The greatest goldfield of the world, viz., the Witwatersrand, responsible for 37 per cent. of the world's gold production, is a Pre-Cambrian goldfield, but the criteria of ordinary Pre-Cambrian fields do not apply to it. Its deposits lie in sedimentary quartzites and conglomerates and are undoubtedly decreasing in tenor in depth. Having regard to all the geological conditions surrounding auriferous deposition on this field, it may be assumed that its gold was deposited relatively near the then existing surface and that deposition was due either to cooling on approach to the

*Maclaren, Trans. Inst. Min. Met., Vol. XVI., 1907, p. 15.
†Min. Sci. Press, Aug. 31, 1912, p. 264.
‡Loc. cit., Nov. 30, 1912, p. 701.

surface or to admixture with oxidizing waters, which in basin-shaped sedimentary areas as those of the Witwatersrand, we know from analogy with artesian areas, may reach to depths of several thousand feet. The surface of most Pre-Cambrian goldfields, on the other hand, has been subjected to erosion during a large portion of geological time, and the locus of gold deposition though now comparatively near the surface, was at the period of impregnation many thousands of feet below the then existing surface and beyond the reach of oxidizing waters, perhaps even beyond the influence of thermal changes.

Summary.

Where auriferous ore bodies have been deposited by the influence of meteoric oxidizing waters or by cooling on approach to the earth's surface, they may reasonably be expected to diminish in tenor with increasing depth and finally to disappear. The deposits to which this generalization appears to apply are those of the Tertiary andesitic group. Even for many of these, non-persistence of ore is more often a function of geological structure than of increase in depth. For all other deposits, and especially for those of the Pre-Cambrian group, ore formed in strong well defined fissures may be expected to persist unchanged (apart from local horizontal variations) in "depth" provided the rock of the lode walls is homogeneous and that the ore-bearing fissure does not pass out of that rock. In all these, therefore, geological structure and not "depth" is the factor controlling the persistence of ore. "Depth" exercises only an indirect control, inasmuch as the greater the depth the more is the likelihood of geological change.

THE USE OF COMPRESSED AIR IN CYANIDATION*

By Herbert A. Megraw.

Compressed air has long been used in various branches of metallurgy for the ordinary mechanical purposes common to every industry, but in the cyanide process of recovering gold and silver from their ores there are several applications peculiar to that system which do not apply to industry in general and which may be of interest. Especially is this true regarding that modern development of the process known as "all sliming," in which ore containing precious metal is ground to such a point that nearly all of it will pass a fine wire screen having 200 openings per linear inch, and is then stirred or agitated with the cyanide solution.

The Cyanide Process.—Primarily the cyanide process is simple. Gold and silver are soluble in aqueous solutions of the alkaline cyanides, so that simple soaking of the metal-bearing material is sufficient to accomplish the dissolving of a portion of the desired metal, which is subsequently recovered by precipitation from the solution. Combinations of gold and silver exist, however, which are not so readily soluble in cyanides, so that stronger chemical forces must be brought to bear upon them, and in other cases the metal is so finely divided and distributed through the rock, being practically sealed into it, that the dissolving solution cannot reach it unless the rock is first reduced to an extreme state of sub-division. Thus complications are introduced, many of which have no proper place of mention in this paper; but others, whose successful overcoming has been accomplished by the use of compressed air, will be mentioned.

Any of the alkaline cyanides may be used for dissolving the precious metals, but in practice either potassium or sodium cyanides are used as they are more stable and more easily manufactured and handled. Potassium cyanide was the first salt used, and there are even now operators who believe it is best adapted to particular ores; but the general opinion is that either of the two mentioned may be used with equal effect. Because sodium cyanide contains more cyanogen in the same weight, it is being more largely used, a consequent saving in transportation expense being obtained. Occasionally other cyanide combinations are more efficacious (in the case of particularly rebellious ores) and bromo-cyanides are made use of, but in general the simple sodium cyanide is all that is required for most satisfactory results.

The process is based upon what is generally known as Elsner's equation, which is the chemical reaction expressing the combination of gold, potassium cyanide, and oxygen producing the soluble potassium-gold salt.

$$2\ Au + 4\ KCN + H_2O + O = 2\ KAu\ (CN)_2 + KOH$$

This reaction exhibits the actively necessary part played by oxygen in the process, and indicates the reason for its introduction by artificial means into its practical application. The reactions illustrating the dissolution of silver are somewhat more complicated, although the result is the same kind of a cyanide compound, but the necessity for oxygen is even more pronounced. Many methods of chemically supplying the element have been proposed and tried, but they have been almost universally set aside as being expensive, complicated, and offering no advantage over ordinary air. The opportunity to combine mechanical and chemical service in one agent has been too apparent to remain overlooked, so that at the present time compressed air is one of the most universally-used aids to the process.

First Use of Compressed Air in Cyanidation.—In the early days of cyanide metallurgy, the approved method of treating an ore was to grind it rather coarsely, place it in a tank containing a filter bottom, and leach it with the cyanide solution. By this method the surface of the charge in the tank—that portion exposed to the action of the atmosphere—gave a good extraction of its contained metal, while the interior of it—that portion with which the air could not come into contact—was found to give up its metal content at a much slower rate and in less degree. Believing that oxygen was what was needed, some operators introduced compressed air under the filter bottom and forced it up through the charge, aerating the material thoroughly. This was one of the first uses, if not the actual pioneer application, of compressed air in practical cyanidation. The idea was even covered by patent, the scheme being to fill the bottom of the tank, under the filter, with a network of stationary pipes perforated with small holes into which air could be forced to find its way through the sand in the tank. This operation was sometimes varied by connecting the bottom of the tank with the intake of a compressor, drawing air down through the charge. This latter procedure was found to induce packing of the charge and was not considered as satisfactory as forcing the air upward, which loosened the sand and made it more permeable.

*From Engineering Magazine, Jan. 19, 1914.

Sand and Slime.—Finding that fine material interfered with rapid and even leaching of the tank contents, the two were separated and the clean sand leached by itself, a much more satisfactory method. The finer material, at first discarded, was later found to contain a good proportion of the valuable metal, often the most of it, and the problem of its treatment arose. It could not be leached, as it was too fine and packed so tightly in the tank as to be practically impermeable, and the only way in which it could be satisfactorily extracted was to keep it in suspension in cyanide solution while the dissolving progressed, and to separate the two after the extraction of the valuable metal was complete. The fine material, by common consent, has been called slime; and while the exact definition of the word has been the subject of much controversy, it is generally taken to mean the extremely fine clayey or colloid portion of the ore which may not be successfully leached. Its treatment bothered operators for many years, and in the devising of an applicable method, the agitation system, now highly perfected, was born.

El Oro Methods.—At the mill of the El Oro Mining & Railway Co., at El Oro, Mexico, one of the early plants in America to attempt the separate treatment of slime, the material was agitated in shallow, rectangular tanks by means of compressed air introduced through a hose moved from point to point by an operator. The system made use of the mechanical efficiency of the air in keeping the solids in suspension, and also the oxygen in the air to supply that necessary in the completion of the chemical reaction. This method was troublesome and expensive, requiring too much labor to be economical, and was soon replaced by round tanks in which the agitation was mechanically done, using a central vertical shaft provided at the bottom with arms for keeping the solids in suspension, and air was added either through fixed pipes in the tank or into the suction of a centrifugal pump used to assist agitation. A later development made use of a hollow vertical shaft through which air was introduced, passing thence through hollow agitating arms and escaping through a series of small holes, thus combining mechanical and air agitation at the same time. These methods gave good results; but the notion of "killing two birds with one stone" had not been forgotten, and agitation by means of compressed air alone was studied until the tank known as the "Brown" or "Pachuca" was developed and became almost universally used.

The "Pachuca" or "Brown" tank is a tall cylinder of steel plate having a cone-shaped bottom and is fitted with a tube vertically placed in the centre. This tube acts as an air lift, raising the mixture of solid and solution and discharging it near the top of the tank, thus maintaining a constant circulation and giving no opportunity for the solids to settle. The tank is very tall in comparison to its diameter, made so in order to minimize the chance of settling; and to further this object, the cone bottom is made so steep that solids can hardly remain on it. The usual dimensions of the standard tank of this kind are about 15 feet in diameter and 45 feet deep, although in some cases the proportions are varied. The central tube reaches from near the bottom of the tank, where compressed air is admitted, to within a few inches of the top. This latter point is the subject of many variations, some operators maintaining that the necessary conditions are complied with when the tube reaches only half way up the tank. It will be seen that this is an air lift working under practically ideal conditions as to submergence, and power requirements are not large. In a standard tank, which will hold about 100 tons of dry slime at the usual dilution, the operation is stated to require about 100 cubic feet of free air per minute, compressed to 25 to 50 pounds per square inch. The higher pressures are required to start operations after the tank has been without agitation for some time, and the lower ones to maintain agitation after it has been started.

The Trent System.—Having thus devised a method of treatment which accomplished a straight air-agitation treatment, operators continued to experiment with means of improving it and reducing its cost. The height of the "Pachuca" tank, as it is usually called in the United States, militated against its economy, as it is generally believed now that it requires more power than should be necessary for the performance of its duties. Many other systems have been invented and put into use, most of which use compressed air as motive. One known as the Trent system operates by the use of a centrifugal pump in a tank of large area, having a flat bottom, and comparatively low. Such tanks may be 20 to 30 feet in diameter and 10 to 15 feet deep. The pump is fed from a point near the surface of the tank and forces the pulp through an appropriate gland in the tank bottom into a system of arms, made of pipe, placed so that they may revolve near the bottom of the tank. The arms are fitted into a grit-proof bearing, and the pulp exits are all curved in one direction, the force of the discharging pulp causing them to revolve in the tank after the fashion of an automatic lawn sprinkler. To supply oxygen to this system, air is introduced into the pump column from a compressor; or this may be done by arranging a sniffle valve in the suction, although this is likely to reduce the efficiency of the pump.

The Dorr Agitator.—Probably the latest claimant for honor as an agitator is that known as the Dorr, which uses a flat-bottomed tank like that in the Trent system. The mechanism is like that used with the well-known Dorr thickener—a central shaft carrying two arms, inclined upward at an angle from the centre. Upon the bottom of these arms are a series of inclined blades, which act as rakes, drawing the settled material toward the centre of the tank. Upon reaching the centre, the slime comes under the influence of an air lift, which circulates it much as it is done in the Pachuca tank. The discharge of this air lift may be free, as in the Pachuca tank, or through a distributing canal fastened to the central shaft; in the latter case the discharged slime is distributed over the entire surface of the tank. Here again the air is made to serve two purposes. The particular advantage claimed for this tank is that it requires very little power, the mechanically moved arms revolving very slowly and requiring only from 1/4 to 1/2 a horse power, while the air lift has no great height and small quantities of air under low pressures may be used. Tanks with flat bottoms and great area, compared with their height, are preferred on account of their low installation cost and also the influence upon the first cost of the milling plant. When tall tanks are used, costly excavations are required in addition to the heavy installation expense of the tanks themselves.

It is, of course, not to be thought that the devices mentioned cover the entire field of slime agitation, as many other ideas have been put into practice; but they do represent the most successful types and indicate the essential fact that air can be made to perform two distinct functions at the same time.

Other Uses of Compressed Air in Cyanide Plants.—In addition to the uses named, compressed air performs many other duties about the cyanide plant, most of which, however, are purely mechanical. The air lift is used for elevating both pulp and solution, and compressed air is often used in connection with filtration, the separating of the solids from their treatment solutions. The precipitate resulting when gold and silver are thrown out of the solution is pumped through a filter press, either by means of direct-action pumps or by a monteju, and the cake formed is often partially dried by passing compressed air, sometimes heated, through it. Compressed air is largely used in the operation of fuel-oil burners to melt the precipitate into bullion, and in many other ways this transformation of energy may be considered almost indispensable about the modern cyanide plant. While all of these uses are important, naturally the one in which both chemical and mechanical properties are utilized is most prominent.

THE TREATMENT OF ZINC ORES*

By A. J. Colclough Nettell.

It will be my endeavor to point out that the deposits of British Columbia are capable of treatment locally, and by methods that are in every-day use in Wales. To this end I will roughly sketch the history of the zinc industry from its inception in England, and the various processes employed, both for the smelting of docile and refractory ores.

The real history of zinc starts in Silesia. It was from the deposits there that the first metal came. The furnaces of to-day have many of the characteristics of the original Silesia furnace. It is my intention, however, to deal with the English history of the metal.

Zinc was first known to English miners in the mines of North Wales, where it exists in mechanical mixture with galena. The charter for the oldest of these mines, "the Halkyn," was granted by Charles I. At this time zinc was unknown to commerce, and for many years after it was of no value to the miners. As time passed along the Silesians began to mine and produce zinc, and it then became necessary for a process to be discovered to separate and concentrate the metals from the above ores.

A Concentrating Method Devised.—After years of trial and experiment, Messrs. Green, of Aberystwyth, to some extent perfected a process sufficiently good to enable the zinc blende to be commercially treated. This process consisted of a water concentration over jigs and buddles and treatment in huge tubs called "Gogwrs" in Welsh, and "dollies" in English, and this process with improved type jigs, percussive tables of the "Wynne type" replacing the "buddles," is the one in general use to-day, and is giving every satisfaction.

The Morriston Spelter Works.—Once it became possible to separate satisfactorily the blende and the galena, it became necessary for a smelting process to be evolved to deal with these products. The lead was already provided for, and the Silesians had been for some time treating blende, and that very big and powerful corporation, known to the world as the "Vieille Montagne," was operating at Liege, in Belgium. The late Lord Swansea, afterwards head of the firm of Vivian & Sons, then turned his attention to this industry, and about halfway through the last century inaugurated the smelter, known from that time to today as the Morriston Spelter Works. He engaged a French metallurgical engineer, named Alfred Borgnet, to take up the control of this department, and work was commenced. The process then used was in general principle that used both in England and elsewhere, and although the furnaces have undergone a very radical and complete change, the general principles are much the same now.

The Welsh Furnaces.—The early furnaces, or, as they are now called, the Welsh furnaces, consisted of huge fire grates built below ground level in pits, known as "caves," and from ground level up of huge brick boxes into which were placed tiers of clay pots to the number in some cases of sixty pots arranged in five tiers. This portion of the furnace was known as the "Block," and in this the flames from the underground fire places circulated around the pots, raising the heat to the necessary point for the distillation of the metal.

Above the "Block" was built a calciner, or roaster, operated by the surplus heat from the block below, the whole of the gases from the spelter furnace and calciner being carried into the smoke stack provided for each furnace.

The charge for the largest of these furnaces was 3,000 pounds and the yield of metal was considered good at 50 per cent. of the metal contents and more frequently was below rather than above this figure.

Gas Producers Used.—This class of furnace continued in operation in all the spelter works in Swansea until Siemens perfected his gas producer, when the spelter-men grasped the possibilities of a furnace in which the heat was obtained from this source. Then came a period of radical change, the calciners were built separately, each smelterman using the one that suited his fancy, the number of tiers of pots was reduced to three and the length of the furnaces increased to give room for the same, and even a greater number of pots, and, consequently, greater charges, and so was gradually evolved the present-day furnace of 154 pots and four-ton charges.

But even whilst the furnaces were being so improved the yield of metal did not increase with the same rapidity, and all the metallurgists were keenly at work trying various fluxes to aid in increasing the yield of the, by this time, very valuable metal. So by slow stages was evolved the present-day process as worked in Swansea, whereby a yield of 90 per cent. is obtained and under good conditions even higher than this.

Welsh Ores Contain Fluorine.—The ores treated now are principally obtained from Spain and Algiers. Although there are still great quantities in the North Welsh mines, they unfortunately now carry fluorides. Until some means is found of eliminating this element, the Welsh can only be worked in very small quantities. Fluorine is fatal to the pots.

Broken Hill Ores.—A period of experimenting was started when the Broken Hill deposits of docile ore were exhausted, and the sulphides or complex ores came before the public. It became necessary to find some method of extracting all the values from these ores, and process after process was evolved for this purpose. Although most were failures, each one taught us some lesson on the treatment of zinc and put us probably one step further toward the goal of treatment whereby all the values of ores of this and other complex class could be recovered.

James Process.—It would take too long to epitomize all the processes brought out, so I will confine myself to those that I have been personally more or less intimately connected with. The first of these being by a metallurgist named Christopher James, who worked on the chemical fact that on the meeting of sulphides and oxides under heat, reduction takes place. A number of experimental charges were worked and some results obtained, but the mechanical difficulties did not warrant any further experiment on these lines.

An Improved Process.—Then came another process, where this principle was acted on, together with the principle employed in Parkes desilverizing process, and some better results were obtained. This process consisted in passing the finely ground blende, or complex ore carrying silver or gold values in addition to the zinc and lead, into a bath of highly heated litharge. Reduction took place and as fast as this occurred the face of the bath, at least, was further oxidized by means of an air blast. When the bath was considered sufficiently enriched or in danger of overflowing the fire was damped and the bath cooled back. While cooling it was agitated with paddles until below the melting point of zinc. The zinc contained rose to the surface of the lead in a crust carrying most of the silver, gold and copper with it, and was skimmed off. The zinc was then distilled off leaving the copper and other values in the distil pot or in the zinc pot residues for subsequent treatment. This process gave good results, and I cannot say why it was not prosecuted further.

The Burnham Process.—Next came the "Burnham" process, so called after the metallurgist who brought it out. It was taken up by a London syndicate, and a lot of money was spent on it. It consisted of an agglomeration of the crude ore with nitre cake whereby, it was claimed, the zinc was converted into zinc sulphate. The soda absorbed the siliceous gangue of the ore forming sodium silicate which passed through the subsequent blast furnace treatment practically without change. The resulting scoria from this treatment was then either treated in a spelter furnace or by leaching. It, however, did not work out as claimed and so was not continued.

An Electrical Process.—Next was an electrical process patented by a metallurgist named Grant. This consisted of the deposition of the zinc from its solutions by electrolysis. It was found so difficult to maintain the necessary absolute evenness of current to this end that this process also was relegated to oblivion.

Sulman-Picard Processes.—Later have come the two processes of Messrs. Sulman & Picard. The first of these was worked at the Emo Works, just outside Swansea. It consisted of the briquetting of the ore with coking coal, pitch being used as the bond, and the subsequent distillation in the spelter furnace. The zinc was distilled off and caught in the usual way. The silver, lead, copper or other metals were left behind in the coke formed. This was the most feasible process up to that time, but unfortunately they reckoned without the silica, and instead of metallic lead carrying the silver or gold values being formed, a basic-silicate of lead was the result. This is exceedingly difficult of treatment and results in heavy losses of the higher values. So the process was abandoned. In the opinion of the writer, it might have been made a good and paying process, by research into the field of fluxes.

The Bi-sulphite Process.—These gentlemen next brought forward the process known as the bi-sulphite process, now the property of, and exploited by the British Metal Extraction Co. The company has spent an enormous sum of money on experiments to perfect the mechanics of the process. The process consists of the elimination of the zinc from the other constituents of the ore by means of its own sulphurous acid. The bisulphite of zinc thus obtained is converted into zinc oxide by heat. The operators have obtained the zinc oxide; but are still faced by the difficulty of smelting to metallic zinc by ordinary methods. As the oxide of this metal is light and bulky it is very difficult, if not impossible, to get anything like a full charge on to the spelter furnace. This will entail increased cost in the treatment and I much doubt if the materials will stand it. However, I wish them every success.

Conclusion on Treatment Methods.—From the foregoing list of failures, and partial failures, it will, perhaps, have occurred to your minds that zinc looks impossible, but this is far from being the true state of the case. Zinc is just as amenable to treatment as any of the other metals, given the necessary environment, and the proof of this is that the metal is being worked at a profit on a very large scale to-day.

It is not my intention to go into figures, as these may be obtained from the Governmental Blue Books of the various countries that work this metal. What I have been trying to impress by the foregoing, is the fact that zinc in some of its phases is not so amenable to chemical treatment as its metallurgy would lead one to believe. But my opinion, based with all respect, on twenty-odd years experience of this metal in both its docile and refractory forms, is that the present method in common use is the best up-to-date, and that consists of an initial concentration of the ore, calcination and subsequent treatment by means of the furnace employed in Swansea which combines all the best points of the Welsh, Silesian and Belgium furnaces, and is, I am firmly of opinion, the best for the purposes.

Application to British Columbia Ores.—The deposits of British Columbia, at least those I have seen, appear to be quite amenable to concentration even when in admixture, as some of them are, with copper. Such being the case, the subsequent treatment should present no difficulty by the ordinary direct smelting method. If copper and other values are present they can be recovered by the subsequent treatment of the residues.

Importance of Mechanical Arrangement of Ore Particles.—There are methods other than chemical, which will do this concentration and separation, but I would like at this juncture to point to one fact which I think many engineers overlook, and that is that although the ores come from the same district and even from the same lode, they vary enormously as to their mechanical arrangement. This is one of the most difficult points to be faced by the concentrator, and although the general principles of any concentration process may be the same, yet the details must be worked out to suit each individual ore.

Reason for Neglect of B.C. Zinc Ores.—We have in British Columbia huge deposits of zinciferous ores which, at present, in all but very isolated cases, are being almost entirely neglected, and why? Because, as far as my information enables me to reply, of the heavy freights necessary to take these ores to the nearest smelters, customs duties, penalties, etc., which eat up all the value of the metal contained in the ore. In one case I saw the smelter returns, and after all these deductions had been made there actually remained for the payment of mining and concentrating charges $7.50 per ton. This was on an ore making 45 per cent. zinc, and zinc was at this time worth about 7c. per pound. Yet there is no move being made to alter this. Around Vancouver there are zinc deposits which could be worked at a considerable profit if there was a smelter within reach. There are others that you gentlemen know better than I do. Yet these products to be got to market must be sent across the line into the United States.

The Market for Zinc.—It is open to proof that there are profits in zinc smelting and mining, and yet no endeavor to cut into the market for galvanized sheets, zinc salts or other form of zinc product has, as yet, been made. What is galvanized iron worth to-day in Vancouver? I am credibly informed $100.00 to $110.00 per ton. The English manufacturer obtains for his about £8, or $40.00, per ton.

Vancouver is nearer and quite as well supplied with bottoms as England, to supply the South American, and Oriental markets. She is better situated than England in that she has within the province the crude material, the coal and even natural gas. England has to import all the ore now treated there.

India presents a big field for the marketing of zinc chloride which is used for the burnettizing of their railway ties; China and Japan for galvanized sheeting and other zinc products.

I have not in the foregoing processes made any very serious mention of electrical ones, as my experience has caused me to be somewhat prejudiced. I am informed that experiments are now being carried out along these lines in the province, so I refrain from passing any opinion.

I am also informed that experiments of this kind were carried out some time since in this province by a metallurgist named Schneider, but I understand they were not wholly satisfactory.

Finally, I would suggest that if a process should be wanted in a young, if one may use the term, province like British Columbia, the Swansea process I have given would be the most suitable, as it has stood the test of time. Other improvements and adaptations, that might from time to time be found necessary to meet requirements, could then be built up around this process.

B. & A. ASBESTOS COMPANY.

B. & A. reports a good profit from operations during 1913.

The mine now operated by the B. & A. Asbestos Company was first discovered in the fall of 1908 by a Mr. Labonte, a French-Canadian prospector, who sold it to the present owners in December of the same year. Construction of the plant was commenced early in 1909. At that time, the mines at Thetford Mines and Black Lake were supplying most of the asbestos produced in the Province of Quebec, and, with the mines at East Broughton, Belmina and Rumpelville, which had then just been opened, competition was so great that the opening of another new mine seemed a foolhardy venture. Undaunted, however, by this and by the discouraging advice of their friends, the directors of the company pushed forward with the mine. Needless to say, they encountered many difficulties. Labor was scarce, on account of lack of accommodation for the men, the initial daily production was small, and, owing to the mine being practically unknown, what little fibre they did produce, they were unable to sell. Early in 1910, the company was re-organized, more capital was added, and operations were resumed on a larger scale. Improved machinery was installed to provide for a larger output and accommodation provided for the laborers. The daily production, under these improved conditions, increased gradually, month by month, during 1910, 1911 and 1912. In 1913, the B. & A. Asbestos Company had an average output of 125 tons of asbestos fibre daily, more than produced in a single day by any two other mines in the world. Nor has quality been sacrificed for quantity. The fibre is clean, long and free from grit. Experience and constant supervision has reduced the cost of production to a minimum. The percentage of fibre obtained from the rock on this property is far above the average.

The company is enlarging the pit and installing additional equipment with a view to increasing the efficiency.

The company is not contemplating an amalgamation with any East Broughton mining companies, as might be inferred from a paragraph in our last issue.

MAGNETIC IRON ORE SURVEYS.

The Department of Mines has published reports by E. Lindeman on Austin Brook Iron-Bearing District and Magnetite Occurrences along the Central Ontario Railway. The reports are accompanied by a number of maps prepared by Mr. Lindeman and his assistant, Mr. W. M. Morrison.

HAILWOOD LAMPS PASS BELGIAN TEST.

The Belgian Government has signed the authorization approving of the use in Belgium of the Hailwood oil lamp and also the Hailwood naptha lamp, as manufactured by Ackroyd & Best, Ltd., Morley, England.

The Belgian Government tests are far more severe than the British Government tests, natural gas from the Grand Trait mine being employed in the gas tests. The lamp is subjected to most explosive mixtures of this gas and air at velocities of 5, 7, 9, 11, 13 and 15 metres per second, which is equal to from 984 feet per minute to 2,950 feet per minute. The whole series of the tests being repeated in horizontal currents, downwardly descending currents at an angle of 45 deg., upwardly ascending currents at an angle of 45 deg., vertical ascending currents and vertical descending currents.

These authorizations are especially interesting as the Belgian Government will not now add new lamps to their list unless they show special merit, and the details of the lamp are scrutinized very severely.

PERCENTAGE RECOVERY IN ORE DRESSING

By Edgar P. Anderson, Cobalt.

In the recovery of metals from the rocks in which they occur, it is generally necessary to concentrate them in the form of metallic minerals before the metal is extracted by smelting operations. In thus concentrating the mineral a loss always occurs, so that the actual mineral recovered for smelting is a portion only of the original content. The ratio of the metal in the recovered portion to the metal contained in the original ore when multiplied by one hundred is called the Percentage Recovery.

In practice this factor cannot as a rule be determined directly, as the concentrate may be wet or oily and is usually inconvenient to weigh. The metal in the ore delivered for concentration may be more easily determined, since its dry or actual weight can be found, by sampling for moisture and deducting the weight of water. Generally weights of the various products can be accurately determined at a testing works, where comparatively small parcels of ore are treated as a guide to practical work. For testing works the only percentage recovery which should be taken as correct is the following:

$$\text{Percentage recovery} = \frac{100 \times \text{wt. of concentrate} \times \text{assay value of concentrate}}{\text{weight of ore} \times \text{assay value of ore.}}$$

It is the recovery obtained that is wanted, not the loss sustained. The latter may serve as a check, since the tailing loss plus the metal recovered should not be greater (nor much less) than the metal in the original sample.

To obtain the percentage lost in the tailing the following calculation must be carried out.

$$\text{Percentage loss in tailing} = 100 \times \frac{\text{metal in tailing} (= \text{assay} \times \text{weight})}{\text{metal in ore } (= \text{assay} \times \text{weight})}$$

and metal in ore should be equal to metal in concentrate+metal in tailing.

In practice there are two methods available (both dependent on sampling) supposing that the total weights of products cannot conveniently be obtained.

1. The ore is sampled and its average metal content obtained as a percentage. The concentrate is also carefully sampled and the assay value determined. The samples taken should be large and are taken as a certain fraction of the whole over a considerable period, say one shift. These samples are dried, so that water is driven off and also oil, if used, but nothing else. If any solid, added during treatment, is found in the concentrate, such as carbon, magnetite, iron, etc., the weight of added material must be deducted from the concentrate. The percentage recovery can then be estimated as for a test sample.

This method is very good but gives unnecessary work for comparative day to day runs.

2. The ore feed is carefully sampled periodically and the assay value in metal returned as a percentage. The concentrate obtained is also sampled over the same period and the percentage assay value determined.

The following formula gives the percentage recovery quite close enough for practical work, the errors being due to sampling and assaying only.

$$E = 100 \frac{c\,(a - b)}{a\,(c - b)}, \text{ where}$$

E=extraction rate per cent.
a =assay of ore per cent.
b =assay of tailing per cent.
c=assay of concentrate per cent.

This formula is made up from the assay values and the concentration rate.

The formula may be proved thus:

$$\frac{c}{a} = \text{concentration rate or the ratio } \frac{\text{assay value of concentrate}}{\text{assay value of ore sample.}}$$

$(a - b)$=assay of ore—loss in treatment=proportion of metal taken out of the ore.

$(c - b)$=assay of concentrate—loss in treatment=proportion of metal concentrated.

Formula says:

$$\text{Extraction} = \frac{\text{proportion of metal in concentrate} \times \text{proportion of metal taken out}}{\text{proportion of metal in ore} \times \text{proportion of metal concentrated.}}$$

That is, if there were no loss in the tailing metal extracted=metal in ore, and metal in concentrate=metal concentrated, dividing out in the above equation the total metal extracted=1, which is correct.

To obtain the percentage loss in the tailing the following formula may be used, the symbols having the same significance as above:

$$\text{Loss in concentration per cent.} = 100 \times \frac{b - \frac{b}{R}}{a} = 100 \times \frac{b - \frac{ab}{c}}{a}$$

where R=concentration rate $= \frac{c}{a}$

This result may be added to the extraction and should give 100, serving as a check on the extraction determined from the concentrate value. For convenience the loss may be determined very quickly because of the smallness of the figures involved and subtracted from 100 to give the approximate extraction, say when in a works and wishing to know "if the treatment is going satisfactorily this morning."

That is

That the addition of the extraction and the loss give the metal contained, is shown by the following:

Using the same symbols, and R=concentration rate.

$$\text{then } R\,c + (1 - R)\,b = a \quad (1)$$

$$\text{for } R = \frac{a - b}{c - b}$$

$$\text{and } 1 - R = \frac{c - a}{c - b}$$

Substitute values for R in formula (1)

$$\frac{a-b}{c-b}\times\frac{c}{a}\quad\frac{c-a}{c-b}\times\frac{b}{a}=1$$

That is metal extracted+metal not extracted=1 =metal in ore.

In ore dressing operations there are often intermediate products, or 'middlings,' to be treated again. Such products will complicate the formulae given above. Mr. C. W. Wright in the Mining Magazine of July, 1911, gives the following derived formula.

$$E=100\times\frac{Wc\times Ac}{(Wa\times Aa)+(Wb\times Ab)+(Wd\times Wd)}$$

where E=extraction per cent.

Wa=weight of ore feed, of assay value=Aa.

Wb=weight of tailing, of assay value=Ab.

Wc=weight of concentrate, of assay value=Ac.

Wd=weight of middling, of assay value=Ad.

Percentage recovery may be checked graphically, and the graphs or curves, obtained by several successive recovery tests, will enable percentage recovery to be determined directly. Graphical treatment has been dealt with by Mr. Sharwood in the Mining Magazine for December, 1910.

True extractions are usually difficult to obtain, and, when submitted, are frequently misleading, owing either to omission of products or by the intention of interested parties.

ORE SHIPMENTS FROM COBALT SILVER DISTRICT IN 1913.

According to the Cobalt Nugget, the shipments from Cobalt mines during 1913 amounted to 20,916.26 tons, a decrease of 500 tons from the shipments for 1912. Taking into consideration the amount of concentration and bullion reduction of the Cobalt mines this reduction is noteworthy, and, as compared with a 500-ton reduction in the 1912 figures from the 1911 figures, it shows a steady shipment from Cobalt during the last year.

The list shows two new shippers during the year, the Lumsden and the Gould, although a number of old faces have passed away and new ones substituted. In place of Silver Queen, Aladdin appears. Caribou-Cobalt appears for the Drummond shipments for the year Orion supplants Silver Cliff, and York Ontario the old King Edward. In addition Lost and Found or General mines made a shipment from a North Cobalt group of properties early in the year. The silver Bar shipped its first carload, although on two previous occasions the name has appeared with test shipments.

The general average of the various mines was about on a par with last year, with one or two specific decreases and a corresponding number of increases. Seneca-Superior, which stepped into the shipping class late in 1912, appears last year with a heavy total. Buffalo, always a heavy shipper in the past, now reduces the silver into bars and ships by express, showing a decrease in the tonnage from the camp. Of the larger mines, Nipissing shipped slightly in excess of the previous year residue from the high-grade mill, rich in cobalt and nickel oxides, but low in silver values, predominating. La Rose shows a decrease of some 200 tons, while McKinley shows an increase of a couple of hundred tons. Coniagas' decrease was 500 tons, but to offset this, Cobalt Townsite increased shipments by 800 tons.

Other mines show slight fluctuations, both decreasing and increasing tonnage, and this, together with the fact that the bullion shipped was much greater and increased concentration, argues well for 1913 as a banner year in Cobalt.

Last year 27 mines appeared on the shipping list; this year there are 31 mines, an increase of four. Mines which appeared this year and not on last year's figures included Green Meehan, Lumsden, Gould, Silver Bar, Orion (Silver Cliff), and York Ontario (King Edward), while the Hargraves and Provincial do not appear on the list.

The following figures compare the shipments by months of 1912 and 1913:

	1913.	1912.
January	1,657.83	1,235.07
February	1,718.06	2,063.63
March	1,980.09	1,678.13
April	1,601.76	1,782.79
May	1,839.50	1,928.72
June	1,649.07	1,707.37
July	1,494.77	1,669.55
August	1,586.30	1,980.12
September	1,603.56	1,871.48
October	1,423.11	1,775.61
November	2,268.48	1,608.48
December	2,093.73	2,380.84
	20,916.26	21,631.79

By years the tonnage from the Cobalt camp is as follows:

	Tons.
1904	158.55
1905	2,336.01
1906	5,836.59
1907	14,851.34
1908	25,362.10
1909	29,942.99
1911	33,976.97
1911	24,921.71
1912	21,631.79
1913	20,916.26
	179,934.31

PEAT.

The peat deposits of Canada are quite extensive and constitute an important reserve of fuel that has as yet been but little utilized. The most important areas so far as known are those found in the Provinces of Quebec and Ontario. A number of these have been systematically examined and surveyed by the Mines Branch with a view to determining their character and extent. The Branch has also carried out a comprehensive investigation of the fuel values of peat, having built a plant in Ottawa for demonstrating the feasibility of the manufacture and use of peat gas in gas engines. During the past two years air dried peat fuel from the Government bog at Alfred was sold in Ottawa, and peat from a privately owned bog at Farnham, Quebec, was disposed of in Montreal. In both cases the fuel was in considerable demand for use in open grates and in kitchen ranges. The Alfred bog is now being operated as a private enterprise and a considerable production is anticipated.

CYANIDING SILVER ORES AT NIPISSING MINE, COBALT*

By James Johnston.

The cyanide plant is placed in a building apart, but connected to the battery and tube mill building. It is arranged in two floors, the upper one containing all the slime collecting and the cyanide treatment vats and the lower floor containing all the solution vats, slime filters, and pumps.

There are on the upper floor:

Three 34 ft. diameter by 13 ft. deep slime collecting vats,

Seven 34 ft. diameter by 13 ft. deep slime cyanide treatment vats,

Two 34 ft. diameter by 13 ft. deep stock slime pulp vats for charging filters,

One 34 ft. diameter by 13 ft. deep barren solution vat.

These vats are placed in two parallel rows, and, with the exception of the solution vat, they are all fitted with a mechanical stirring apparatus, driven by a line shaft placed over the vats on a wooden bridge or trestle. The paddles in the vats are made of 4 by 6 in. pine on edge and the two longest arms are 27 ft. in diameter and revolve at 8 rev. per minute, giving a speed on the end of 678 ft. per minute; paddles are located at 2 ft. from the bottom of the vat.

These 12 vats are driven by a 125 h.p. motor, and as on an average 8 of them are agitating at one time, the power necessary to drive vats and shafting is 55 kw. Each vat has an average working load of about 140 tons of dry slime plus about 280 tons of solution.

The slime collecting vats are arranged with a circular overflow around the top on the inside, 6 in. wide by 8 in. deep, over which the clear battery solution flows into the launder leading to the lower battery solution vat. The bottoms of the vats are connected to a 7 in. diameter and to a 4 in. diameter centrifugal pump for transferring the thickened pulp, and a 6 in. diameter decanter is fitted to the side of the vat.

The battery pulp is run into one of these vats for about 9 hr., collecting about 92 tons (dry weight) slimes, and during this collecting period the excess solution overflows to the lower battery solution vat, from which it is repumped for circulation in the battery solution circuit. When the necessary charge is collected, the pulp is switched to collect in the next vat and in the meantime the slime is settling, excess solution being decanted until the pulp in the vat represents about 1.5 solution to 1 slime. The pulp is now agitated for about 1 hr., the depth of pulp measured, sampled, and specific gravity determination made for slime tonnage. The specific gravity of dry slimes on this ore is 2.7. The calculation made here, as previously noted, works out at practically the same tonnage as the weighed-in weight of the ore, after due allowance has been made for the addition of lime and tube mill pebbles.

The thickened pulp of 1.5 caustic soda solution at 0.25 per cent. strength and 1 of slime is now pumped to the desulphurizing treatment.

Desulphurizing Process.

The problem of working out a successful all-cyanide treatment for the Cobalt ores, in which there is such a varying amount of complex minerals, led to the discovery, during the period of experimental work, of what is now known as a wet desulphurizing process. The details of the reactions involved and the necessity for such a treatment will be obtained by referring to J. J. Denny's article. (See page 711, Nov. 15, issue.) Briefly explained, the preliminary desulphurizing treatment breaks up the refractory silver minerals when the slime pulp is brought into contact with aluminum in a caustic soda solution, the silver being left in a spongy metallic state readily amenable to cyanide treatment.

The desulphurizing is accomplished by passing the slime pulp through a revolving tube mill, in which there is a quantity of aluminum, and a further treatment is given in a vat lined with aluminum plates, in which the pulp is slowly agitated.

The practical effects of this preliminary treatment on the ores has resulted in the same and sometimes a better extraction being obtained in 48 hr. cyanide treatment than was otherwise obtained in 120 hr. cyanide treatment with no desulphurizing. In some of the ores which contained a greater proportion of the refractory minerals a better extraction of from 1 to 4 oz. per ton is obtained when desulphurized and cyanided, as compared with a cyanide treatment and no desulphurizing.

A tube mill, 4 ft. in diameter by 25 ft. long, lined with 2 in. thick Silex blocks, is used for the first stage of the desulphurizing treatment. This mill revolves at 10 rev. per minute and carries a load of about 4,000 lb. of aluminum ingots, cut up into cubes about 1.5 to 2 in. The slime pulp is fed through this mill at the rate of 14 tons of dry slimes per hour, diluted with 1.5 caustic soda solution to 1 dry slime. The pulp then gravitates into the 34 ft. diameter by 13 ft. deep alkali stock pulp vat, which is arranged with mechanical agitation and lined around the side with aluminum plates. The pulp is agitated in this vat about 24 to 36 hr. or until it is gradually drawn off in about 40-ton charges to the dewatering filter box. On account of keeping the mill cyanide solution in balance, it is necessary to eliminate as much as possible the crushing caustic soda solution from the slimes before they are transferred into the cyanide vats for treatment, and this is done in a Butters filter plant equipped with 60 leaves, which is capable of dewatering 270 tons of slimes per day, the cake when discharged carrying 26 per cent. alkali solution as moisture.

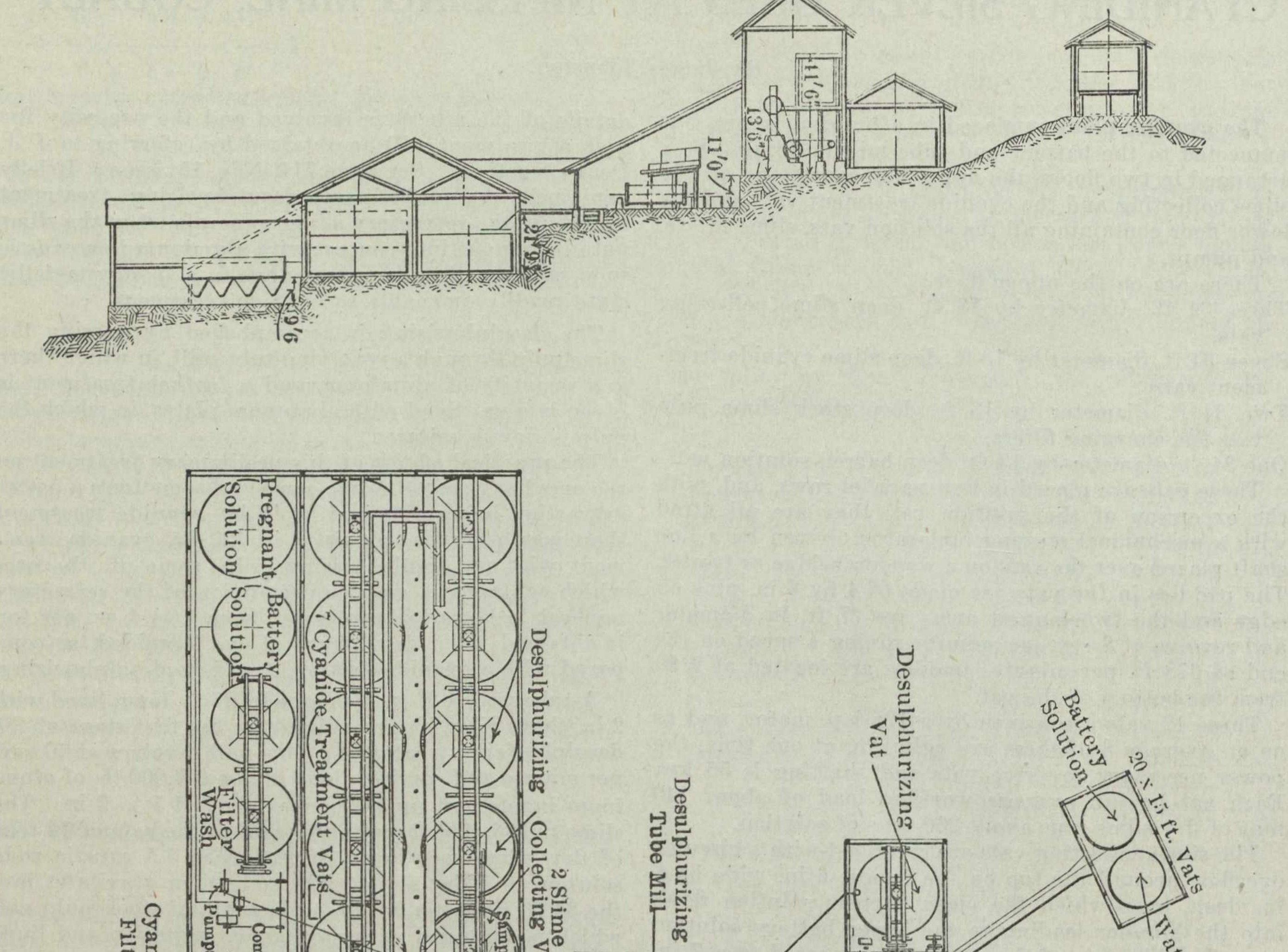
The slimes after dewatering flow into a 16 ft. by 5 ft. deep pulping vat, where they first come in contact with cyanide solution, being diluted with about 2 of cyanide solution to 1 of slime. A 4 in. diameter centrifugal pump is used for transferring the pulp into the cyanide treatment vats.

Cyanide Department.

There are seven, 34 ft. diameter by 13 ft. deep slime treatment vats, fitted with mechanical stirring arrangement. Each vat has a 6 in. diameter decanter, for draining off the clear settled cyanide solution; they are also fitted with a 6 in. diameter air lift operated by a 0.75 in. diameter pipe with air at 20 lb. pressure.

The slimes are treated in charges of about 130 tons dry slimes in a vat, with a dilution of 2 of solution to 1 of dry slimes in a 0.25 per cent. cyanide solution and 0.20 per cent. alkali, agitation being maintained for 48 to 60 hr. Repeated testing in the mill has demonstrated that equally as good an extraction of the silver values

*Extract from a paper on the Mill and Metallurgical Practice of the Nipissing Mining Co., published in Bulletin of the A.I.M.E., January, 1914.



Plan and Section of the Low-Grade Ore Mill of the Nipissing Mining Co., Limited.

is obtained with a 2 to 1 dilution as with a 3 to 1 dilution. The air lift is operated all the time the charge is being treated, so that there is a possible turning over of the charge about 16 times by the air lift.

After the agitation treatment, the charge is allowed to settle, so that the clear solution may be decanted to the pregnant solution vat. The pulp is then agitated and transferred by a 7 in. diameter centrifugal pump to the 34 ft. diameter by 12 ft. deep cyanide stock pulp vat, in which it is kept agitated until drawn off into the filter box.

A Butters filter plant equipped with 80 leaves handles

35 to 40-ton charges in about 3 hr. each, the slimes being discharged with 26 per cent. of moisture. The alkali and the cyanide filters are arranged alongside of each other, so that the same operator handles both boxes from the switchboard placed between them. The filter plants are arranged on the semi-gravity plan; a 10 in. diameter centrifugal pump with 12 in. diameter suction and delivery pipes and hydraulically operated valves is connected with the cyanide box for transferring the excess pulp and solution wash. The clear solution from the cyanide filter box is delivered to the pregnant solution vat and the residue slime is dropped into the empty box, discharging by gravity to the residue dump.

The alkali filter box is connected to an 8 by 10 in. duplex double-acting piston vacuum pump with a displacement of 340 ft. per minute at 160 rev. per minute.

	Heads. per Ton.	Total Silver.		Extraction.	Average Residue per Ton.	Total Residues.
79 tons of	2,648.47 oz.=	209,229.13 oz.	at	99%	26.484 oz. =	2,092.29 oz.
7,320 tons of	26.00 oz=	190,320 oz.	at	92%	2,080 oz. =	15,225.60 oz.
7,399 tons	54.00 oz.			95.66%	2.34 oz.	

The cyanide box has a 12 in. diameter by 10 in. stroke duplex double-acting piston vacuum pump with a displacement of 770 ft. per minute at 160 rev. per minute.

The pregnant solution vat is 34 ft. diameter by 8 ft. deep and the barren solution vat is 34 ft. diameter by 13 ft. deep.

Precipitation Department.

The original plans of this section of the mill were laid out with the intention of using zinc dust for precipitation and an equipment was installed of one 20-frame and one 10-frame 52-in. Merrill precipitation presses with 3-in. frames, together with the other necessary machinery.

Before the mill was ready to start, the experimental work on the use of zinc dust as a precipitant had disclosed that the cyanide solutions after precipitation rapidly deteriorated and fouled, so as to lose their dissolving efficiency. The investigation on this subject has been very fully explained in E. M. Hamilton's and J. J. Denny's articles and proves how impossible it would be to use zinc dust and get good results. After considering various other methods of precipitation, it was decided to use aluminum dust as the precipitant and modify the equipment accordingly.

The practical benefits gained by this change are:

No fouling of cyanide solutions, with a corresponding reduction in dissolving efficiency.

The working mill cyanide solutions are in a more active condition to dissolve silver than if they were freshly made up solutions which had not yet been used, showing in their favor an increased dissolving power of 0.35 oz. of silver per ton of ore.

A regeneration of 0.608 lb. of cyanide per ton of solution precipitated, equal to about 1.67 lb. of cyanide per ton of ore treated, or 408 lb. per day.

The recovery of silver precipitates averaging about 27,000 oz. per ton, or about 93 per cent. silver.

The pregnant solution is pumped to the precipitation room by a 6 by 9 in. vertical triplex pump, running at 54 strokes per minute, where it is clarified in a sand filter before flowing to the special tank arrangement for mixing the aluminum dust in the cyanide solution. It is then pumped into the precipitation presses by another 6 by 9 in. vertical triplex pump. The 20-frame press takes about four days to fill up with precipitates when handling about 550 to 600 tons of solution per day with a head assay running about 8.25 oz. and a tail assay 0.10 oz. The aluminum dust consumption over a nine months' period averages about 0.556 lb. per ton of ore treated, or 1 lb. (avoir.) dust=45.26 oz. (troy) silver precipitated, or 1 lb. (avoir.) dust= 3.104 lb. (avoir.) silver precipitated. The precipitates are sent to the refinery, where they are melted and refined in a reverberatory furnace, as described in R. B. Watson's article, and eventually shipped as bullion at 997 to 999 fine.

Extraction.

The following figures represent the average extraction obtained from a run-of-mine ore of 54 oz. of silver per ton, in the combined metallurgy of the high-grade and low-grade ore mills:

To this will be added the further recovery of 85 per cent. of the silver, made on the sale of the above 26.484 oz. of residue.

The working mill costs are here represented in the percentage that the various most important expenses bear to the total cost.

Labor	28.207
Cyanide	19.943
Electric power	14.542
New construction supplies	5.586
Aluminum dust	4.761
Aluminum ingots	4.709
Caustic soda	1.811
Aluminum plates	1.401
Refinery fluxes, fuel oil, coke, etc.	1.332
Pebbles	1.284
Battery supplies	1.013
Lime	0.579
Sundry supplies	14.832
	100.000

The annual meeting of the Western Coal Operators' Association was held at Fernie, Crowsnest district, British Columbia, on January 9. Those present were Messrs. Lewis Stockett and W. F. McNeill (Secretary), Calgary; John Brown, Hillcrest; A. C. McGibbon and O. E. S. Whiteside, Coleman; Jas. Howard, Taber; J. C. Reid, Lethbridge. These are all Alberta members of the Association. Owing to illness, Mr. W. R. Wilson, manager of the Crowsnest Pass Coal Co., was unable to attend.

A press despatch from Washington, D.C., January 15, says: Preliminary tests of Alaskan coal from the Bering River district have been very discouraging to officials who hoped they might develop a new fuel supply for the navy. Rear Admiral Griffin, in charge of the investigation, has reported to the House Naval Affairs Committee, that the Bering river coal tested so far has fallen so far under expectations in practical use as to be of no value, but from the Matanuska fields and other sections of the Bering district from which coal is yet to be tested, the navy is hoping for better results.

BOOK REVIEWS

HEATON'S ANNUAL—Heaton's Agency, Toronto. Price $1.00. Postage 12c.

The 1914 edition of Heaton's Annual has come to hand. With this issue, the Annual arrives at the end of its first decade. The first edition was compiled in 1904 for the Department of Commerce of the Dominion Government, to meet the requirements of British firms doing business with Canada, and 15,000 copies were purchased and distributed by the Government in Great Britain and continental Europe. Year by year the Handbook has been gradually developed with special regard to its original purpose, and to meet the requirements of those who want a book of easy reference to answer questions regarding the Dominion. In the first 214 pages will be found information on the Customs Tariff and Regulations, the digest of the Customs Laws and Regulations, includes all the memoranda and bulletins that are issued by the Department of the Customs Officials. This information cannot, so far as we know, be found collected in any other single volume. The Shippers' Guide, giving population, banking accommodation, and railway connections in every banking town in the Dominion is a new feature.

The second half of the book contains a concise, up to date description of all the towns in Canada of any commercial importance, including the leading hotels in order of merit, the existing industries, and special opportunities for new industries. To this is added a section covering such subjects as agriculture, fur-farming, commerce, education, finance, fisheries, forests, immigration, mining, population, professions, railways, game laws, water powers, etc.

FIRST AID IN MINING (Metalliferous Mines)—by Louis G. Irvine—Published by the South African Red Cross Society, Johannesburg, S.A., 1913.

This little book of 115 pages has been issued as a supplementary to the First Aid Manuals in general use. It contains specific information regarding accidents in mines. Suggestions are given regarding the equipment of Emergency Stations and of the First Aid Box. A resume of the treatment of wounds, hemorrhage and fractures is given. Methods of transport of injured persons are described. The discussion of accidents resulting from the fumes of explosives and other forms of "gassing" is especially noteworthy.

The several chapters are devoted to, materials provided for First Aid Treatment in Mining Accidents, First Aid Treatment of Ordinary Emergencies and Accidents Underground, Transport of the Injured in Mining Accidents, Accidents from Poisonous Gases, Accidents due to Electricity, Cyanide Poisoning.

Copies of the book may be obtained from the McGraw Hill Co., 239 West 39th Street, New York, U.S.A.

FIRST AID (Miners' Edition)—American Red Cross Abridged Text-book on First Aid—By Major Charles Lynch and Lieut. M. J. Shields—P. Blakiston's Son & Co., Philadelphia—Price 30 cents net.

The needs of miners, so far as might be, in respect to first aid instruction, were met in an earlier book of this series which was called the "Industrial Edition." This, however, was for other industries as well as for mines. Now, the spread of the first aid movement in a vast number of mines has seemed to demand a special book for miners. Furthermore, experience has shown that instruction in safety and in first aid should go together even more than was thought at first, though in all the Red Cross first aid manuals the importance of the subject of prevention as a part of first aid instruction has always been insisted upon. The reception of the former editions of the Red Cross first aid books was very gratifying. It is hoped that a larger experience will make the present edition more useful and practical.

The chapters are headed, What First Aid to the Injured Is; Directions for Giving First Aid; Shock, Bandages, Injuries, Bleeding, Suffocation and Artificial Respiration, Drowning, Electric Shock, Gas Poisoning, Unconsciousness or Insensibility, Poisoning, How to Carry Injured, Safety, Care of Injured, and Organization of First Aid Instruction. While the book seems to have been specially written for coal miners much of the information contained is quite applicable to metal mines as well.

The volume is a neat paper bound one of 181 pages and numerous illustrations.

PERSONAL AND GENERAL

Mr. W. E. Segsworth is in England.

Mr. Donald G. Forbes, of Victoria, B.C., is spending the winter in Great Britain.

Mr. C. L. Dennison and other directors of the Buffalo mine, spent a week at Cobalt and Porcupine during January.

Mr. Chas. F. Caldwell, managing director of the Utica Mines, Ltd., has returned to Kaslo, B.C., from a visit to Ontario.

Mr. W. R. Wilson, general manager for the Crowsnest Pass Coal Co., Ltd., has been sick at his home in Fernie, B.C.

Mr. John D. Galloway, has been gazetted assistant to the Provincial Mineralogist for British Columbia. His headquarters will be at Victoria.

Mr. Jay P. Graves, vice-president of the Granby Consolidated Mining, Smelting & Power Co., is at Pasadena, California, for the winter months.

Mr. Oscar Lachmund, general manager for the British Columbia Copper Co., was at Butte, Montana, last month, returning to Greenwood about the middle of January.

Mr. A. Hickling, director of the Princeton Coal and Land Co., has left Princeton, Similkameen district of British Columbia, on his homeward journey to England.

Mr. C. A. Foster is reported to have completed the purchase of the shares of Messrs. Tough & Oakes in the Tough-Oakes mine at Kirkland lake for the Kirkland Lake Proprietary, Ltd. Mr. Foster has returned to London.

Mr. Charles Emmerson, formerly superintendent for the West Canadian Collieries, Ltd., at its Bellevue colliery, western Alberta, recently returned to Bellevue from Peru on a visit.

Mr. H. A. Guess, brother of Professor Guess, of Toronto University, has been made consulting engineer for the American Smelting and Refining Co. His headquarters will be in New York City.

Messrs. C. E. Smith, of Toronto, P. A. Robbins, of Porcupine, and O. E. S. Whiteside, of Coleman, have been nominated by the Council of the Canadian Mining Institute to fill vacancies on the Council.

Mr. John Brown, general manager of the Hillcrest Collieries, near Frank, Alberta, was taken seriously ill about the end of December. He was taken to Calgary to have an operation performed to give him relief.

Mr. Oscar V. White, superintendent for the Slocan Star Mines, Ltd., in charge of the well-known Slocan Star group of mines near Sandon, Slocan, British Columbia, has been spending a few days with relatives in Spokane, Washington.

Mr. J. Thomas, mine superintendent at Passburg, Alberta, is spending a vacation at the mineral water spring near Frank, for the benefit of his health. In his absence Mr. N. Howells is in charge of the coal mine at Passburg.

Mr. M. S. Davys, of New Denver, B.C., managing director of the Silverton Mines, Ltd., operating the Hewitt-Lorna Doone group of silver mines and a concentrating mill near Slocan lake, has been on a visit to Spokane, Washington.

Mr. Frederic Keffer, of Greenwood, engineer and geologist for the British Columbia Copper Co., has been on a visit to family connections in Cleveland, Ohio, whence he went to New York before returning to British Columbia.

Mr. Wm. Fleet Robertson, Provincial Mineralogist for British Columbia, has been appointed Provincial Assayer as well, the latter office having been vacant ever since the resignation of Mr. Herbert Carmichael, who retired at the end of 1912 after having been fully twenty years in the service of the Provincial Government. Mr. D. E. Whittaker remains Assistant Assayer, and as well he is now Public Analyst for British Columbia.

Press despatches during the week indicate that Raoul Madero, a younger brother of the late President Francisco Madero, of Mexico, is now the chief adviser of General Villa, commander of one of the warring factions in Mexico. Raoul recently assisted General Villa at the capture of Chihuahua. Raoul Madero is a graduate from the Michigan College of Mines, as is his brother Julio.

Mr. John Hopp, who has for years been operating several hydraulic placer-gold mines in the vicinity of Barkerville, Cariboo district, B.C., has staked part of the ground near Quesnel Forks, also in Cariboo district, worked for years by the Consolidated Cariboo Hydraulic Mining Co. with the late Mr. John B. Hobson as general manager. Six or seven years ago Guggenheim interests acquired the property, but after spending between \$200,000 and \$300,000 on extension of the water supply system, there came a period of financial stress, and this enterprise was practically abandoned and eventually the right to the ground lapsed.

Arrangements are being made for holding a meeting of the Western Branch of the Canadian Mining Institute in Vancouver, B.C., on Thursday, February 19.

Canadian Allis-Chalmers, Ltd., has issued a bulletin on jaw crushers.

The Herbert Morris Crane & Hoist Company has issued a bulletin describing Morris Overhead Runways recently installed.

The Short Session for mining men at the College of Mines, University of Washington, Seattle, Washington, opened on January 5 with a registration of 44 mining men from 10 States, Alaska, and 5 foreign countries. The course lasts three months and is arranged in sections for quartz, placer and coal miners.

The Roberts and Schaefer Company, engineers and contractors, Chicago, have just secured a contract from the Paint Creek Collieries Co., Charleston, W.Va., for the building of a new coal mining plant in which the Marcus Patent Picking Table Screen will be installed. This new tipple will be built at Olcott, W.Va. Approximate contract price \$27,000.

The Harlan Coal Mining Company, of Louisville, Ky., has also awarded a contract to the Roberts and Schaefer Company for a new Marcus coal tipple and retarding conveyor from mine, which plant will be built at Coxton, Ky. Contract price \$25,000.

SPECIAL CORRESPONDENCE

BRITISH COLUMBIA

Only occasionally is there available much information concerning mining in Cassiar district. Quite recently there was issued by the British Columbia Department of Mines, Victoria, a "Preliminary Review and Estimate of the Mineral Production, 1913." The following summary of the year's progress in the several divisions of Cassiar district has been taken from that bulletin:

CASSIAR DISTRICT.

The extensive area known as the Cassiar District includes the following mining divisions: Atlin, Stikine, Skeena, Queen Charlotte, and Portland Canal.

Atlin Mining Division.

Atlin has become the most important placer-gold district in the province, and is estimated to have produced this past season about \$320,000 worth of gold, which represents about 60 per cent. of the provincial output of that metal. This is an increase of 10 per cent. over last year's production, and is the greatest output the camp has made since 1908.

It is encouraging to note this increase in a placer camp, for, while it was assisted by a favorable season as to weather and water supply, it was chiefly obtained by increments on old properties and the development of new producers.

Pine Creek.—The Ruffner properties on Pine creek still continue to be the most important in the camp. The Guggenheim properties on Tar flats, held under lease by the Ruffner companies, are reported to have done better than usual, the output, in default as yet of official figures, being estimated at about \$75,000. The North Columbia Gold Mining Co. is likewise estimated to have produced about \$45,000, probably a little less than last year. There are few individual miners on Pine creek, and they are not expected to have made any important output.

Boulder Creek.—The Black claims are the most important on this creek, which, together with the individual miners, are expected to have produced about \$10,000 worth of gold. Gold Run did not make any material production this year.

On Birch creek, H. P. Pearse was the most important producer, being estimated to have obtained about \$16,000, while individuals obtained from \$2,000 to \$3,000.

On Otter creek, J. E. Moran is estimated to have recovered some \$4,000.

The miners on Wright creek are said to have obtained from \$4,000 to \$5,000.

On Ruby creek, T. M. Daulton's company, the Placer Gold Mines, is reported to have done exceptionally well, having, after some years' preliminary development, struck very rich ground. Of this fact there seems little doubt, but as to the amount of gold recovered there is a wide margin between the various unofficial estimates obtained; but it seems safe in this preliminary estimate to credit the company with $35,000. The company is said to have had delivered in Atlin some $12,000 worth of high-carbon steel plates for lining its sluiceways, so it would appear as though the company not only had money in hand, but also high expectations from its property. The development of this property is one of the new features of the camp this past season.

Spruce Creek.—It is reported that the Spruce Creek Power Company did not operate. On the Gladstone lease, James McCloskey, though said not to have taken out quite as much as in 1912, is still credited with some $45,000. Individuals working on Spruce Creek—of whom there are quite a number—are reported to have realized in the neighborhood of $44,000; included in this are material outputs by McPherson, Matthews, and others.

Only a small output is expected from Wilson creek, probably not over from $1,000 to $2,000.

On McKee creek, the individual holdings have been pretty well absorbed by the one large company operating on the creek, the Pittsburg-British Gold Company, which is reported to have had a successful season and to have recovered about $35,000, from which it is unofficially reported dividends were paid to the extent of $10,000.

O'Donnell river seems to have proved attractive to J. M. Ruffner, who has been developing property there, since he had not "come out" by the end of the year, and, consequently, definite information is as yet lacking. He is said to have been meeting with fair success, although his output this year is not expected to be much over $3,000.

The area of the Atlin placer-gold field seems to be gradually extending and promises well for a continuance of its production.

Mineral Claims.—The only lode mine producing in this division is the property of the Northern Partnership—formerly the Engineer group—on the east side of Taku arm. The property is and has been equipped for some years with a 2-stamp mill, which has been in operation for a considerable part of this past season, producing a certain amount of gold—how much is not known, although it is claimed that the quartz treated was quite rich.

Mr. O. H. Partridge and Hon. M. Edgerton were developing the Ben M'Chree at the southern end of Taku arm, but appear to have at least temporarily abandoned that work, and have been engaged in development work on the White Moose, on the west side of the arm—not on the old location on the beach, but some distance up the hill to the westward—with what they claim to be very satisfactory results.

The claims in Rainy Hollow, on the headwaters of the Klehini river, do not appear to have had much attention this past season, further than the assessment work necessary to keep them alive.

Good roads have been built into the district, but no serious work has resulted on the claims, nor any output recorded.

Stikine and Liard Mining Divisions.

In the Stikine division proper there is no mining going on, and as far as is known very little prospecting; all that has been heard of is a little on the Iskut river by the Iskut Mining Company. There is no placer-mining in this division.

The southeasten portion of the division includes a large part of the Groundhog coalfield, which was described in last year's report, and in which during this past season work has been confined to prospecting, with no serious development, and nothing further has been learned that would indicate the future of the camp.

In the Liard division the only work going on is a certain amount of placer-mining in the vicinity of Dease lake. The only important workings there are those of the Boulder Creek Mining Company, operating a hydraulic plant on Thibert creek, fully described in last year's report. This company has been at work all season, working in the new pit, which is reported as proving very satisfactory and producing a fair amount of gold, the exact amount not yet being known.

The flats at the mouth of Dease creek, described in the report mentioned, have this year been the scene of extensive drilling operations by two companies, for the purpose of testing the gold-tenure of the gravels, which, if these preliminary operations prove satisfactory, will lead to the establishment of a dredging plant, to which mode of working the ground is eminently suited.

Queen Charlotte Division.

As yet, no report of actual production this year has been received from the Queen Charlotte division. On Graham Island, prospecting for coal has been carried on by several parties in the interior of the island, but the results so far obtained have not as yet proved coal of commercial importance.

Boring for oil has been going on at the north of the island, without having met with success.

The British Pacific Coal Mines, which partly equipped a colliery on Skidegate inlet last year, has apparently not continued work to any great extent, nor become a producer.

The very interesting discovery was made that a part of the seam opened up at this place consisted of a peculiar form of carbon, which has been classed by a Russian geologist as schungite, and is described as an intermediate state between anthracite and graphite.

The practical peculiarity of the mineral is that, while it has all the appearance of being good coal, it will not burn and cannot be ignited in a blast-gas flame.

On Moresby Island, on the east coast, the usual amount of development and assessment work has been done on the claims near Lockeport and at Ikeda bay, while work has also been done on Huston inlet claims and on Copper Island, Kunghit Island and Collison bay.

On the west coast of the island, on Tasu harbor, important work has been carried on all year by Mr. R. R. Hedley and associates in developing a copper property containing magnetite impregnated with copper-pyrites. A crosscut adit tunnel has been driven in for 300 feet, which, according to Mr. Hedley's sampling and assays, cuts several bands or zones of mineralization. One band 40 feet thick assayed 2.5 per cent. copper; another, 13 feet thick, 2.85 per cent. copper; and the balance about 1.5 per cent. copper, with a little gold and silver. On account of the excess of iron in the ore, it proved so attractive to the smelter that Mr. Hedley has been offered smelting rates practically free of charge. Mr. Hedley hoped to be able to ship several hundred tons before the close of the year, but it is not known whether he succeeded in doing so.

Skeena Mining Division.

The most important development in this division and on the coast has been the progress made at Granby bay, on Alice arm of Observatory inlet, by the Granby Consolidated Company in equipping its Hidden Creek mines with a mining and smelting plant capable of treating 2,000 tons of ore a day.

A review of this sort will not permit of a detailed description of the plant; it is sufficient to say that it will be up to Granby standard, and will include a most up-to-date mine equipment, while the smelting plant will include blast-furnaces, converters, etc. A great part of the plant has already been erected, and it is almost certain that the summer of 1914 should see it in operation.

The mine has been already described in the reports of this department, but it might be said that the company estimates it has blocked out above the working-adit 8,000,000 tons of 2.2 per cent. copper-ore, which estimate is practically endorsed by the mining engineer sent by this Bureau to examine the property. An average of about 1,000 men has been employed during the summer of 1913.

On Alice arm, in addition to the Granby Company's holdings, this Bureau has official reports showing that some twenty miles up the Kitsaulte river, which flows into Alice arm, there are deposits of copper-ore of a workable grade which the present development would give good reason to believe are of very considerable extent, rendering it probable that this section may also develop into an important copper camp.

Near the Granby Company's property—but not included in it—seemingly extensive bodies of copper-ore have been found on the Bonanza and Groundhog claims. It certainly would appear as though the mineral wealth of this section was just beginning to be discovered.

At Kwanitza, forty-five miles up the Grand Trunk Pacific Railway from Prince Rupert, salt has been discovered, and a small quantity extracted in an experimental way. A large basin occurs at this point on the creek, and on the edges of the basin brine was found to be seeping out. Some four boreholes have been put down in the basin, and each yielded a heavy brine, carrying about ¾ lb. salt to gallon of brine. One of the holes had at the bottom crystalline salt mixed with clay. While much yet remains to be proved, it seems probable that the basin contains a large deposit of salt. The quality of the salt evaporated in a crude evaporating pan on the claim is very pure and quite up to commercial standard.

The properties on Princess Royal Island, owned by the Surf Inlet Gold Mines, Limited, described in last year's report, and which were expected to produce this season, have not yet been equipped and made no output.

Portland Canal Mining Division.

As far as can be learned, there was no ore shipped from this division in 1913.

In the Glacier Creek section of the Bear River district, the properties are all awaiting the development of the 2,800-ft. tunnel being driven by the Portland Canal Tunnels Company. This tunnel had at the end of the year just struck the crush zone in which the ore was found 2,000 feet higher, but as yet no development has been done at the tunnel level.

No further development has taken place regarding the Red Cliff and no ore has been produced.

Mr. D. G. Forbes reported on the Salmon River claims for this Bureau, and considers that the present development gives substantial expectation of the development of large ore bodies of workable grade.

KIRKLAND LAKE AND PORCUPINE

Kirkland Lake Proprietary Limited.—The flotation of the Kirkland Lake Proprietary, Limited, in London, has aroused much interest. The prospectus has been criticized as being somewhat vague and indefinite in its terms. The company, according to the prospectus, secures under the contract the benefit of all arrangements now in course of negotiation, or which may hereafter be entered into, by Mr. Clement Albert Foster in regard to mining properties in this mining field or district, including the right to take the same over at cost price and free of any commission or profit to Mr. Foster. The purchase consideration for the above was $125,000. If the capital of the company is increased the vendor will get $5 in cash for every two shares issued up to 50,000.

The company has to date obtained control of the Tough-Oakes Gold Mines and the Sylvanite Mining Company, and has an option on the control of the Teck-Hughes Gold Mines. The Messrs. Tough and Oakes' interests in the the company were bought out and treasury stock was bought to an amount which gives a clear control. Treasury stock in the Sylvanite Mining Company, which holds the Wright Robbins claims, has been purchased. The company is now working both the Tough-Oakes and the Teck-Hughes mines. The shaft at the Tough-Oakes is to be sunk to the 300-ft. level at once, while drifts on the vein at both the 100 and 200-ft. levels are being prosecuted with vigor. Another compressor has been installed which will give considerable additional power. A shaft has been started on the No. 3 vein in the porphyry.

There is no difference in gold content per ton where the main vein is in the porphyry at the 200-ft. level.

There is no intention of erecting a larger mill until ore reserves have been developed to a much greater extent than at present. The last shipment of ore from the Tough-Oakes mine at Kirkland lake brings the total in gold contents to $60,084, and in silver $3,288. This does not include the gold bars produced in the mill. The fifth shipment, though made months ago, has not been settled for until quite recently. It ran slightly higher in gold than any of its predecessors and averaged much better to the ton in silver.

The total shipments totalled 131.55 tons and cover a period of a little over a year. All the ore shipped consists of vein matter alone.

Dome.—The December statement of the Dome mine shows that for the nine last months of 1913 the company milled 104,330 tons and produced $936,106 in bullion. The production for the month of December was 13,475 tons. Gold produced amounted to $106,904. The mill ran 90 per cent. of the possible time. For the nine months of April to December, inclusive, the Dome ore averaged $8.97 per ton recovered from the mill. May showed the highest grade per ton with $13.68, and August was lowest with $6.31. The ore averaged $7.93 in December.

Teck-Hughes.—The test shipment from the Teck-Hughes, Kirkland lake, ran 102 oz. in silver and $38.15 in gold. These samples were taken from the 100-ft. level of the No. 1 vein, where it was about 20 in. wide. The value of the silver content of the car surprised the company. The shipment consisted of four tons.

The drift on No. 3 vein has now been pushed for 125 ft. to the north at the 75-ft. level. The vein varies from 5 ft. to 18 in. in width. At the present time it is narrow in the face. The controlling interest in the Teck-Hughes is under option to the Kirkland Lake Proprietary Ltd.

Huronia.—An order has been placed for an additional five stamps for the five-stamp mill at the Huronia mine in Gauthier township. The addition includes also a tube mill, three slimers, and other equipment. Development work has been suspended until the work on the power installation on Victoria creek has been completed.

COBALT, GOWGANDA, AND SOUTH LORRAIN

A three-year course in mining at the Haileybury High School, supported by the Provincial Government, is believed to be of considerable importance to the mining profession. It was adopted after a personal inspection of the situation by Dr. Merchant for the Board of Education and a guarantee that the Government would be favorable to finding support for the enterprise. It will, of course, have small beginnings. With the future of Northern Ontario as a mining country assured, there appears every probability of developing a school of mines. At the end of the three-year course a diploma will be given. At the present time the instructor whose salary will be paid by the Government, will devote his whole time to the classes formed. That there is a demand for this instruction is shown by the fact that already 18 pupils have been enrolled. The movement started with the voting of $5,000 for instruction in mining by the Board of Education. It was intended then that these classes should be conducted to enable men already working at the mines to attain some theoretical knowledge; but the classes formed were not a pronounced success as only a very few took advantage of the opportunities offered.

Nipissing.—Three new veins were encountered in the crosscut at the Nipissing's shaft No. 73, at the fourth level. This is believed to be a most important development as the finds were made in virgin territory. This crosscut is exploring the eastern portion of R. L. 400 opposite veins 64 and 73. Near the end of November this crosscut encountered three veins in a distance of 15 ft. All of them are about the same width, being from one to one and a half in. wide and assaying from 1,000 to 2,500 oz. They have a strike north and south and at the present time do not appear to be the extension of anything met with heretofore.

At shaft 64 the new vein encountered during November continues to show up well as drifting proceeds to the northeast. The vein assays 2,500 oz. over a width of 2 in. The main shaft now has a depth of 762 ft. There has been no change in the formation of the rock. The crosscut will be driven to the vein when the shaft has reached a depth of 900 ft.

Favorable developments were met with at shaft 86. The main vein is faulted between the first and second levels and the extension has just been found at the second level. The vein is 2 in. wide and assays 500 oz. The width and assay are satisfactory as development is in the Keewatin, and value should improve in the conglomerate.

The diamond drill is working in the diabase near the southern boundary of R. L. 408. No veins were encountered by the holes driven.

The high grade mill treated 161 tons and shipped 674,984 oz. of bullion. The low grade mill treated 6,268 tons.

During the month the company mined ore of an estimated net value of $164,794, and bullion was shipped from Nipissing and custom ore of an estimated net value of $399,999.

Ore Shipped to Hamburg.—The movement in direction of wider markets for the ore from Cobalt is apparently having some effect. In the past week both the Dominion Reduction Company and the Crown Reserve shipped to Hamburg. This is the first shipment of ore to Germany apart from the regular contract of the Crown Reserve with the kingdom of Saxony.

Cobalt in Demand.—There is now a good demand for ores with a low silver content and a high cobalt content in Europe owing to the success of various experiments with the metal. It has been found that cobalt will toughen steel, whereas nickel hardens it, and, for certain purposes, this is greatly to be desired. While Dr. Kalmus, who is making the experiments with cobalt at Kingston for the Department of Mines, denies that there is any definite economic result from his work, it has gone far enough to make him very optimistic of success. The price of cobalt has advanced considerably. As yet this rise in price has not affected many of the mines. Mines that ship raw ore or concentrate to the smelter still receive no returns for the by-product.

Downey Ships Ore.—The first important shipment of high grade ore from Elk Lake since 1910 came from the Downey property situated between Elk lake and Silver lake. The ore was taken out of an open cut by hand and is expected to run about 2,000 oz. to the ton. There is about ten tons of ore to be shipped. The property from which the silver was taken was in litigation for two or three years, and at the end of that time, as it was hard to raise capital, the owner decided to take out all the ore in sight in order to provide money for development.

The Miller Lake O'Brien Company at Gowganda is developing the property with great energy. The production now averages about 60,000 oz. a month.

The Mann mine on Gowganda lake is now sinking to the 200-ft. level with a view of raising up and blocking out ore. Another carload of high grade ore, which has been bagaged for the past six months, will be shipped soon. It is again stated that there is every probability that the property will be taken over by a British syndicate within a few months.

The Beaver Auxiliary, at Elk Lake has again been obliged to stop underground work owing to lack of water for the boilers. It is feared that the Cobalt Frontenac will also have to close down for a like reason. The Mapes Johnston has commenced the transfer of the machinery from the Montreal James, a defunct Hawthorne company, to the property.

The Mapes Johnston has been previously worked by hand. More rapid development will now be possible.

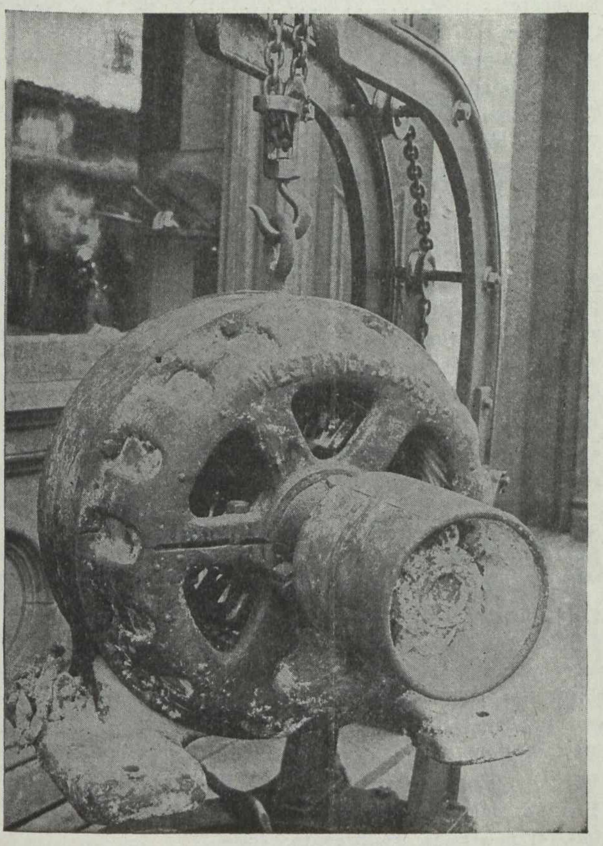
The Cobalt Aladdin, which is working the old Silver Queen on a lease, is now stoping a short shoot of ore which the former company overlooked. The shoot only has a length of 30 ft., with about the same depth in the conglomerate above the second level. It shows six inches of 1,400 oz. ore. Work in the Keewatin below the first and second levels has been abandoned. There are on hand now about 50 tons of high grade ore and concentrate. It is officially announced that 36,737 oz. of silver has been produced since the Aladdin Cobalt took over the lease.

Gould.—The shoot of ore found on the Gould proved to be quite short. It was found near the Seneca Superior line and was but 30 ft. long. The ore in sight is being stoped out and the winze sunk to the 250 ft. level.

Chambers-Ferland.—No. 4 shaft, which is being sunk on the northern portion of the Chambers-Ferland between the La Rose and Nipissing, has reached a depth of 300 ft. Within the next month a station will be cut at 350 ft. and crosscutting operations commenced. While this work is progressing the shaft will be taken down to the Keewatin contact.

WESTINGHOUSE MOTOR GIVES GOOD SERVICE IN A SALT MINE.

The accompanying view shows a Westinghouse motor which was recently removed from the mine of the Myles Salt Company on Avery Island, La., and sent in to be rewound, the motor having burned out after six years' continuous service.



Note the incrustation of salt in the pulley, around the terminals, and especially on the rotor fans and stator windings. Needless to say, a good deal of this salt was shaken off during transit, but in view of the water absorbing quantities of rock salt, there is sufficient remaining to indicate the conditions under which the motor was used.

SHUSHANNA MINING & TRADING CO.

The Financial Times, Montreal, says of this company: An advertisement, worded in extremely objectionable style, has been appearing in the columns of the Montreal Standard and other Canadian papers, urging the public to subscribe for the shares of the Shushanna Mining & Trading Co., Ltd., at 14 cents per share. Applications are to be addressed to the Aetna Investment & Trust Co., of Vancouver. In this connection it is important to remember that the laws of Vancouver permit the free use of the word "trust company" by organizations which are practically free from all inspection and control, and many of which are engaged in speculative business in realty and all kinds of get-rich-quick securities.

The advertisement, which closes with an application blank, includes a half-column of historical matter entitled "Millions in Mines," a quantity of "dope" about Shushanna which is certainly not in agreement with the reports of the Government experts, a few paragraphs on "the broad policy and the men behind the company" (naming, as usual, the reputable law firm which has no more responsibility than that of preparing the charter), and an article entitled, "An Assurance of Legitimate, Big Profits." It does not, however, contain one single word designating any properties, prospects, real estate, buildings, vehicles, trading business, or other assets owned or contracted for by the company, save and except the following:

"There is the already acquired mining property at Shushanna right in the heart of the discovery. Development work will commence on this as soon as weather conditions permit. The following contract has been entered into: Contract dated 8th October, 1913, between the Shushanna Mining & Trading Co., Ltd., and Michael L. McAllister, of Dawson. The company has transferred its properties at Shushanna to the Shushanna Gold Mines No. 1, in consideration for $500,000 to be paid in stock of the latter company." There is no statement of the terms on which these properties were acquired by the parent company, the valuation of the "prospect," or the total capitalization of "Gold Mines No. 1."

The officers of the company are "E. N. Winslow, capitalist; C. J. Stacey, director; J. W. Kager, field manager; H. C. Crumplin, secretary"; to which list is appended the names of the registrar, barristers and broker. There does not appear to be any good reason why these gentlemen should solicit the money of the Eastern Canadian public for so extremely hazardous and vaguely-described an adventure. We are assured that they are "well and favorably known in Vancouver," but there is no mention of any other and more serious financial or commercial organizations which have seen fit to employ their services. Our advice to all intending investors would be to go to Vancouver first, and, if possible, to Shushanna, and find out for themselves. They should remember that the Standard has already assisted in luring its readers into purchasing stock, of Standard Coal, Ruthven's railway-collision patents, and numerous other flotations which never had the smallest chance of yielding any returns—except to the promoters and the newspapers which divided the spoils.

COBALT ORE SHIPMENTS.

The shipments for the week ending January 23, were:

	High.	Low.	Total.
McKinley-Darragh	63,650	....	63,650
Townsite	84,700	....	84,700
Timiskaming	86,450	....	86,450
Casey Cobalt	59,085	....	59,085
	293,885	....	293,885

The bullion shipments for the week ending January 23 were:

	Bars.	Ounces.	Value.
Nipissing	76	92,439.10	$53,152.48
Dominion Reduction	35	39,585.00	22,771.00
Kerr Lake	18	8,462.75	4,167.22
Foster Lease Co.	3	2,187.25	1,141.44
Penn Can.	5	3,416.50	1,771.52
	129	146,090.60	$83,003.66

—Cobalt Nugget.

MARKETS

STOCK QUOTATIONS.

(Courtesy of J. P. Bickell & Co., Standard Bank Building, Toronto, Ont.) January 23.

New York Curb.	Bid.	Ask.
Alaska Gold	23.12	21.37
British Copper	2.25	2.37
Braden Copper	7.75	7.87
California Oil	284.00	285.00
Chino Copper	41.50	42.00
Giroux Copper	1.00	1.50
Green Can.	40.00	43.00
Granby	81.00	86.00
Miami Copper	23.75	24.00
Nevada Copper	16.37	16.50
Ohio Oil	155.00	157.00
Ray Cons. Copper	19.12	19 25
Standard Oil of N. Y.	185.00	186.00
Standard Oil of N. J.	418.00	440.00
Standard Oil (old)	940.00	....
Tonopah Mining	7.12	7.25
Tonopah Belmont	7.75	8.00
Tonopah Merger	.63	.68
Inspiration Copper	16.50	16.75
Goldfield Cons.	1.62	1.87
Yukon Gold	2.37	2.50

Porcupine Stocks.	Bid.	Ask.
Apex	.01¼	.01½
Dome Extension	.07	.07¼
Dome Lake	.25	.25½
Dome Mines	17.75	18.00
Eldorado	....	.01
Foley O'Brien	.18	.19½
Hollinger	16.90	17.10
Jupiter	.08¼	.08½
McIntyre	1.00	1.25
Moneta	.02	.04
North Dome	....	.40
Northern Exploration	3.05	3.25
Pearl Lake	.09½	.09¾
Plenaurum	....	.50
Porcupine Gold	.11½	.12
Imperial	.01¾	.02¼
Porcupine Reserve	....	.06
Preston East Dome	.01¼	.01½
Rea	.20	.25
Standard	....	.01
Swastika	.04	.04½
United	....	.01
West Dome	.05	.11
Porcupine Crown	1.24	1.26
Teck Hughes	.20	.25

Cobalt Stocks.	Bid.	Ask.
Bailey	.06	.06¼
Beaver	.28	.29½
Buffalo	1.95	2.05
Chambers Ferland	.15	.15½
City of Cobalt	.30	.35
Cobalt Lake	.70	.75
Coniagas	7.30	7.50
Crown Reserve	1.94	1.96
Foster	.06	.08
Gifford	.03	.04
Gould	.02	.02¼
Great Northern	.10¼	.10
Hargraves	.02	.02½
Hudson Bay	70.00	75.00
Kerr Lake	4.95	5.00
La Rose	1.94	1.97
McKinley	1.20	1.25
Nipissing	7.90	8.00
Peterson Lake	.26½	.27
Right of Way	.04¾	.05¼
Leaf	.01	.02
Rochester	.02	.02¼
Cochrane	....	.40
Silver Queen	.05	.07
Timiskaming	.12½	.13
Trethewey	.24	.25
Wettlaufer	.06	.08
Seneca Superior	2.90	3.00

TORONTO MARKETS.

Jan. 26.—(Quotations from Canada Metal Co., Toronto):

Spelter, 5¼ cents per pound.
Lead, 5½ cents per pound.
Tin, 40 cents per pound.
Antimony, 8½ cents per pound.
Copper, casting, 15½ cents per pound.
Electrolytic, 15 cents per pound.
Ingot brass, 10 to 15 cents per pound.

Jan. 26.—Pig iron—(Quotations from Drummond, McCall & Co., Toronto):

Summerlee No. 1, $26.00 (f.o.b. Toronto).
Summerlee No. 2, $25.00 (f.o.b. Toronto).

Jan. 26.—Coal—(Quotations from Elias Rogers Co., Toronto):

Coal, bituminous, lump, $5.25 per ton.
Coal, anthracite, $8.25 per ton.

GENERAL MARKETS.

Coke.

Jan. 23.—Connellsville coke (f.o.b. ovens):

Furnace coke, prompt, $1.90 per ton.
Foundry coke, prompt, $2.40 to $2.65 per ton.

Jan. 23.—Tin, straits, 38.30 cents.

Copper, Prime Lake, 14.75 to 15.00 cents.
Electrolytic copper, 14.50 to 14.60 cents.
Copper wire, 15.75 cents.
Lead, 4.10 cents.
Spelter, 5.30 to 5.35 cents.
Sheet zinc (f.o.b. smelter), 7.25 cents.
Antimony, Cookson's, 7.25 cents.
Aluminum, 18.75 to 19.00 cents.
Nickel, 40.00 to 45.00 cents.
Platinum, soft, $43.00 to $44.00 per ounce.
Platinum, hard, 10 per cent., $46.00 to $47.50 per ounce.
Platinum, hard, 20 per cent., $49.00 to $51.50 per ounce.
Bismuth, $1.95 to $2.15 per pound.
Quicksilver, $38.00 per 75-lb. flask.

SILVER PRICES.

	New York. cents.	London. pence.
Jan. 13	57¾	26⅝
" 14	57¾	26⅝
" 15	57½	26½
" 16	57½	26½
" 17	57¾	26⅝
" 19	57⅝	26 9/16
" 20	57½	26½
" 21	57½	26½
" 22	57⅜	26 7/16
" 24	57⅝	26 9/16
" 26	57⅝	26 9/16
" 27	58	26¾
" 28	57⅝	26⅝