Brief Account of the History
Central Patricia Gold Mines

by

Alan Cockeram

PROSPECTING in the Crow River-Pickle Lake district began in 1926.

In 1927, on information that gray copper had been noted on the north bank of the Crow River a few miles east of Pickle Lake, Louis Cohen of Haileybury formed a prospecting group with an original membership consisting of himself, Fred McLeod, Lorne Howey and Jack Morrison. That early winter they sent Alex and Murdoch Mosher in to stake the chalcocite. These men walked in from Savant Lake, an airline distance of 90 miles.

In December they staked 18 claims (Pa. 71-88). While staking, some trenches were noted on the south side of the river (in Claim Pa. 78) showing well oxidized material. History now has it that in the previous year George Simmons and Bill Seiger had made these trenches on an iron formation band. Their best sample only assayed $3.00 in gold, so the ground was not staked.

The following spring (1928), Alex Mosher gave Lorne Howey a sketch of the location of the trenches, and Howey examined the same on his way home from a trip into the area. He brought out samples of the small chalcocite vein, of the iron formation and also picked up a large specimen of quartz carrying chalcopyrite, from a property on Savant Lake. These were sent to Haileybury but in Howey's absence the origin of the two copper samples became confused.

*Secretary, Central Patricia Gold Mines.

D. B. ANGUS
Manager, Central Patricia Gold Mines.

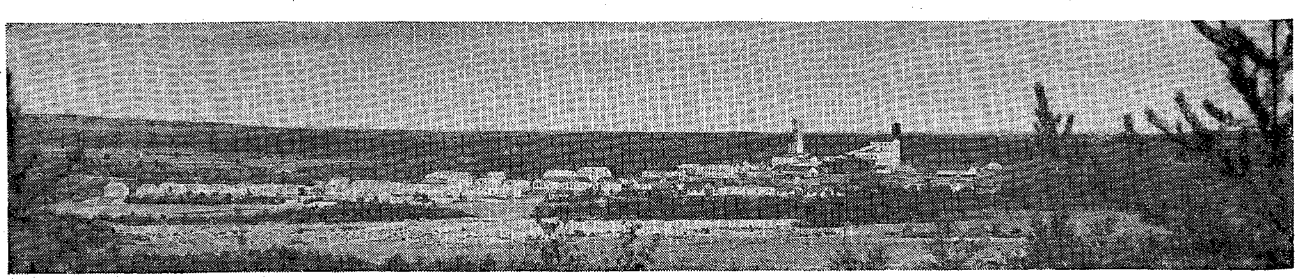
On August 28 these samples were examined by F. M. Connell, and on the strength of the copper sample a thirty day examining option was taken from the syndicate and Arthur Cockshutt sent in to make the examination.

The chalcocite vein proved to be of no importance and the iron showing did not resemble the quartz-copper sample Cockshutt had seen in Haileybury. The iron formation panned poorly. Cockshutt was very disappointed in the showing, but he put the crew to work doing further trenching of the iron formation. One of these trenches cut a heavily mineralized zone which panned well. Being located only 600 feet from the west boundary of the property, he staked an additional six claims for protection.

Cockshutt's samples assayed well, so the option was exercised, and he returned to the property immediately and began directing further trenching and sampling. Construction of camps was begun that October.

Central Patricia Mines Limited, was incorporated on February 19, 1929 with Mr. F. M. Connell as president, which position he continues to hold.

Diamond drilling commenced in February 1929 under the direction of J. C. Kirkland and was completed in September. Based on these results, a decision was made to go underground and A. J. Keast was appointed manager in September.



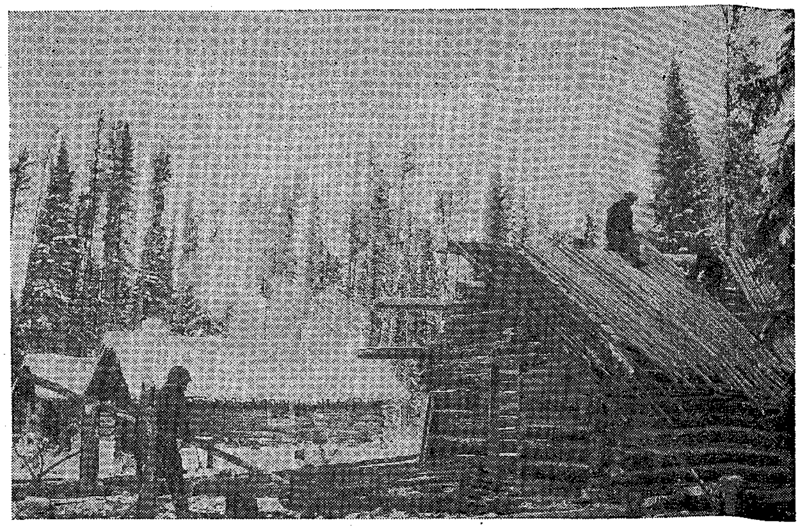
General View of Central Patricia Gold Mines.

A steam powered mining plant was purchased in November and moved to the mine by tractor trains from Savant Lake during the early part of 1930. 400 tons were transported over the frozen lakes and muskegs in 40 days. The plant was erected in February and sinking started on March 1. J. P. Millenbach was resident superintendent and directly in charge of the sinking and development of the 125, 250, 375 and 500 ft. levels.

The high cost of operating 130 miles from the railroad and the limitation of funds forced the closure of the property in September 1930.

The company was reorganized in April 1931 and the assets taken over by Central Patricia Gold Mines Limited, a 2,5000,000 share company. Old shares were exchanged on the basis of one share of new stock for every four old.

Operations were resumed in January 1932 under the direction of A. J. Anderson, who flew to the mine with five men. Camps were rehabilitated and a wood cutting crew employed. The mine was dewatered and underground development work resumed, with such favourable results that a 50 ton mill was designed in February 1933 and delivered to Savant Lake station later in the winter. Due to the lateness of the season and poor ice conditions, most of the equipment was lost when the tractor and sleighs broke through the ice. Fortunately the machinery was recovered the following summer, and was finally delivered to the mine early in 1934. The mill building and foundations, having already been completed, installation was soon effected and production commenced early in May.



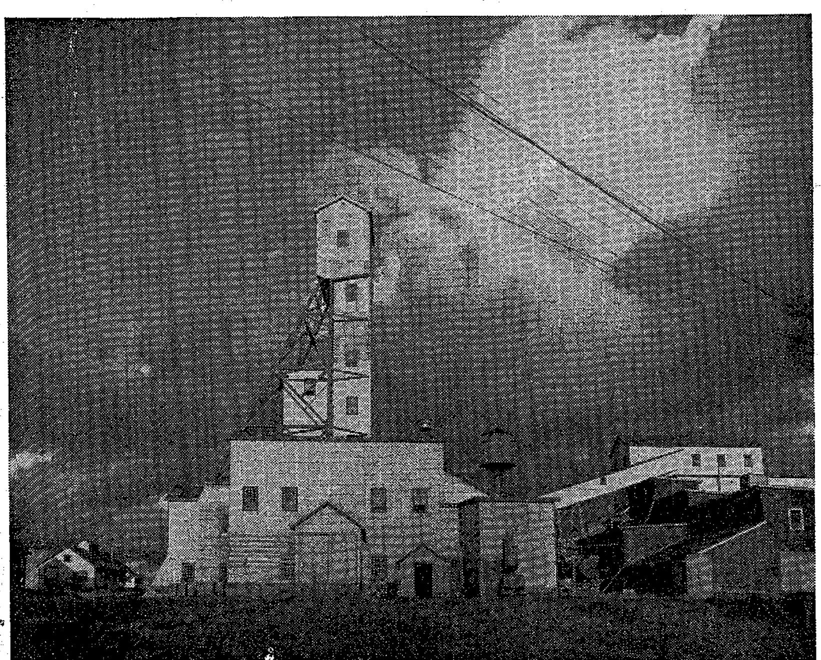
Central Patricia Mines, Winter 1928.

By August it was decided to double the mill capacity and the equipment was delivered the following winter. The high cost and limitation imposed by a steam plant that was burning over 30 cords of wood per day was alleviated by the advent of Hydro-electric power from a development at Rat Rapids at the head of the Albany River. Contract price was $65.00 per h.p. year. Power was first delivered in April 1935, and the following year the tonnage rate increased to 165 tons per day.

In 1936 construction of a road from the east end of Lake St. Joseph was started to complete a system of water and truck road transportation from Hudson, Ontario. A series of marine portages and a 3 mile railroad now connects Lac Seul with Lake St. Joseph.

A. J. Anderson resigned in August 1936 and was followed by R. E. Barrett, who in turn was succeeded in December 1947 by D. B. Angus, the present manager.

Ore reserves and production continued to expand and resulted in an addition to the mill in 1940, bringing the capacity to 400 tons per day.



Central Patricia Shaft and Mill.

Geology of Central Patricia Mine

by

T. T. Tigert *

THE GEOLOGY of the Central Patricia mine is unique in many ways as the ore does not occur in quartz, as in the case with the majority of Canadian Precambrian gold deposits, but in mineralized cross-fractures in a band of sedimentary iron formation. This and other peculiarities touched on below conceivably might cause a similar deposit to be overlooked and at least some knowledge of the deposit will be of interest to engineers, prospectors, and others interested in the development of new mines. It is interesting to relate that although assays of as high as twenty ounces per ton are recorded, visible gold is found only in rare instances.

The mine is developed to the 2050-foot level through a vertical, three-compartment shaft and thence by a four compartment winze to the 3400-foot level. Preparation for sinking another winze to the 3850-foot level is now underway. The mine is made up of a number of different ore bodies in the iron formation and to date twelve ore bodies have been mined, but sometimes an ore body on one level will merge with another or split up at greater depth—therefore the differentiation of ore shoots becomes difficult.

The following notes on geology of the deposit are to be understood as being of a generalized nature and some technicalities have been avoided. A great deal more space would be required for a detailed and documented article.

General Geology

The formations in the vicinity of the ore deposits are Precambrian in age and more or less typical of similar greenstone belts of the Canadian Shield. They present the usual series of Keewatin basic lava flows, volcanic fragmentals and sediments with Keewatin intrusives of diabase, but it is not our intention to perplex the reader with a tabulation of uncertain names and dates.

Of main interest is the iron formation, more fully described below, which is enclosed by beds of volcanic rock with occasional recognizable pillow flows. Within the mine workings are sill-like bodies of rock, which conform in strike and dip with the lava flows, which at first was thought to be an intrusive porphyry but now proven to represent a sheared quartzite. It bears remark that only one instrusive body occurs within the immediate vicinity of the ore body and that is a steeply dipping diabase dike averaging 270 feet in width, striking nearly due north and south, located 2800 feet east of the shaft opening. This dike has been encountered underground on the 1300 level drift and by diamond drill on the 3400 level.

The main structural feature in the area is an anticlinal fold that strikes northeastward and pitches in that direction at an angle of about 60 degrees. The ore-bearing iron formation in the Central Patricia mine occurs on the north limb of the anticline with the beds dipping to the north at an angle of about 70 degrees. The strike of the iron formation is nearly due east-west.

A general outline of the geology in the immediate vicinity of the mine is shown in figure 1.

Iron Formation

The iron formation is a sedimentary rock occurring in bands, which lie between the lava flows. These bodies range in width from a mere trace to 250 feet. Some prominent bands have been traced more or less continuously for miles along the strike, others are quite lenticular and may extend only a few feet. The more persistent

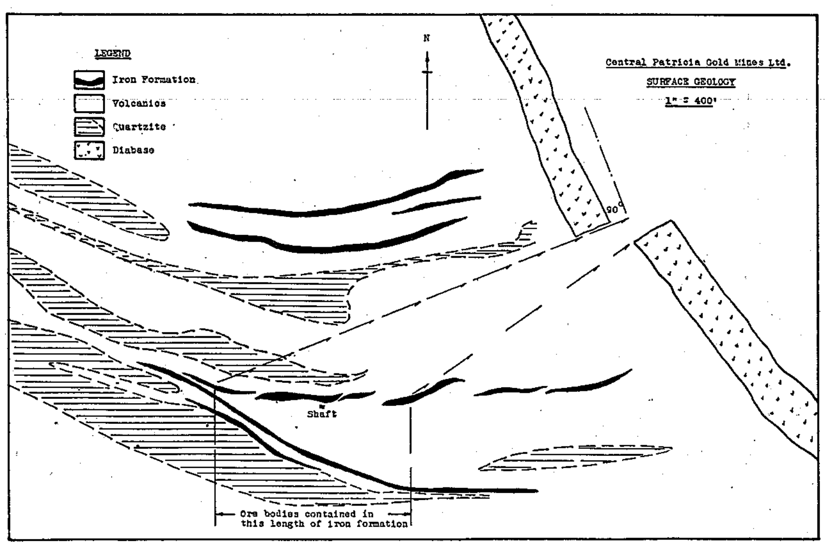


Figure 1.—Geological Outline Map of Surface Areas. General Surface Geology in the Vicinity of the Central Patricia Mine.

*Assistant Manager.

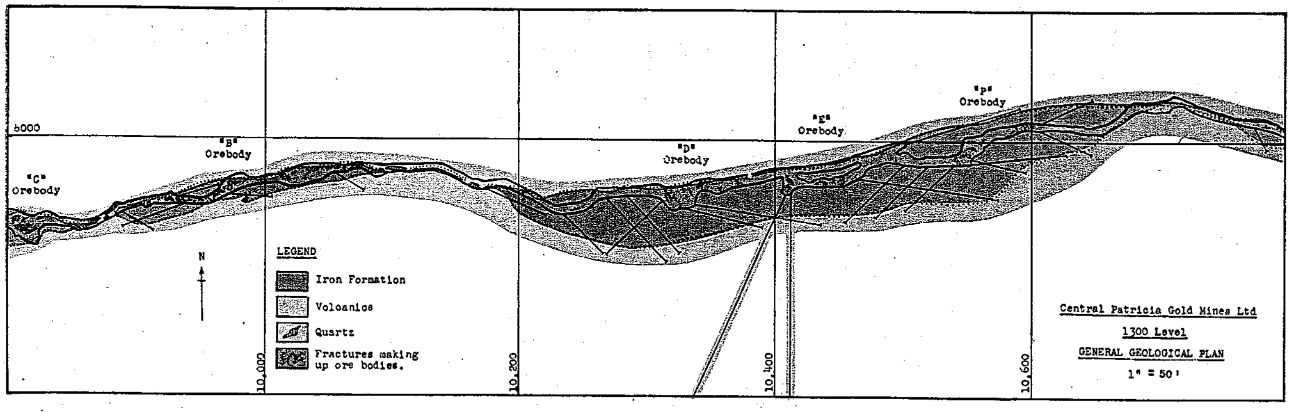


Figure 2.—General Geological Plan of 1300-foot Level.

horizons may pinch or swell, or even die out, at various places along the strike but are sufficiently continuous to provide horizon markers in the greenstone complex. The iron formation presents a wide range of appearance and composition. It is a harder and more competent rock than the greenstones surrounding it and in the general deformation of the country the greenstone has flowed under pressure and become schistose, whereas the brittle iron formation yielded by fracturing, brecciation, or folding.

There are numerous bands in the vicinity of the mine, half a dozen being apparent in the accompanying map (figure 1), and others which do not outcrop have been discovered and explored in underground workings. Almost all of these bands have disclosed some gold values but at the present time only one is of economic importance. Its description may be taken as typical of the iron formation in the vicinity of the mine.

The iron formation is distinctly banded, although at times this may be obscured by shearing or brecciation. The banding shows as alternate strips of light and dark layers, the colours ranging from white to black, and the individual bands range in thickness from a fraction of an inch to an inch or more. Minerals present are chiefly silica, iron carbonate and magnetite, and their relative proportions determine the colour and hardness of the bands. The iron formation occurs as lenses in plan view, up to 800 feet in length and varying in width from a mere thread to 75 feet. The lenses are irregular in shape but in general pinch towards the extremities as may be seen by an inspection of figure 2, showing a portion of the geology of the 1300 level. The lenses are sometimes separated by gaps, locally termed "fault gaps". Vertically, the lenses have been traced the entire depth of the mine, 3400 feet, without a break.

The iron formation conforms in general to the regional eastwest strike but the individual lenses have an *en echelon* arrangement striking 10 degrees off the east-west direction. The greenstone along the hanging wall contact is well sheared which is evidence of considerable movement, while the footwall is relatively "tight."

Of interest to the prospector is the fact that detection of the iron formation by magnetic means is not always possible. There is a high variation in the mineralogical composition of the iron formation and it may, at times, be almost barren of iron minerals, being then made up largely of silica. Under these circumstances, the magnetic reaction will be low, and the other rocks in the vicinity may have as great or greater influence on the needle. When the magnetic content is high, magnetic tracing of the formation is easy and satisfactory.

Ore Occurrences

The gold occurs entirely in the iron formation, associated with sulphides and chlorite, and confined to cross-fractures. The nature of these cross-fractures may best be understood by reference to figure 3 where a large scale plan of the geology of an ore body is shown. It will be noted that the fractures strike roughly across the iron formation, and vary considerably in width, length and number within a given area.

The outline of the different ore shoots is quite irregular and boundaries are determined by assay values rather than by any well-defined change in geological conditions. The openings in the rock have been subsequently filled with vein quartz and heavy sulphide mineralization. The iron formation adjacent to the fractures has been replaced by sulphides and carries low values. The heavy sulphide material in the fractured zones makes high grade ore, and this compensates for the low values in the replaced iron formation. Accordingly the main ore bodies consists of groups of mineralized fractures, which are close enough together to bring the grade of the whole mass up to ore standards. Some of the iron formation between the ore bodies is not entirely barren and may contain a few mineralized fractures with gold values, but these are so disseminated that a minable section would not be of commercial grade.

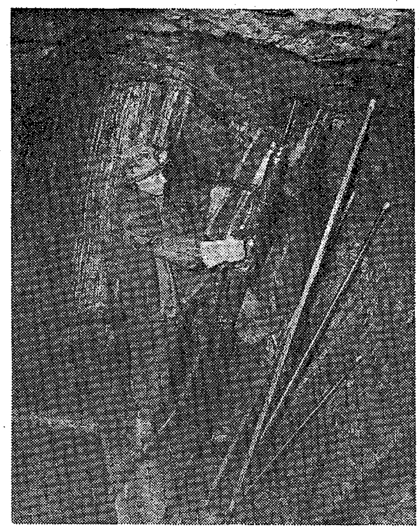
Mining at Central Patricia, therefore, may be said to be bulk-mining of a large number of short, closely spaced, veins. Fortunately the fractures occur in groups and between ore shoots there is only an unimportant number of individual gold-bearing fractures. At times, such fractures will have sufficient length, width, and grade to permit individual mining but the tonnage is minor.

Occasionally, the ore will occur as a more or less solid mass of sulphides replacing iron formation, and at other times a fairly well defined vein of sulphides will follow closely along a contact. However, these cases are exceptional and probably represent special cases of fractures which are too closely spaced or too short to be recognizable as such. In general the cross-fracture pattern as portrayed in figure 3 is most typical and most common.

The fractures dip to the east at a variety of angles, averaging about 56 degrees, and due to this characteristic the ore bodies are endowed with a rake of 56 degrees to the east. The varying length of the fractures are a cause of great irregularity in plan of the ore shoots.

Pyrrhotite and arsenopyrite are the sulphides with which most of the gold is associated; pyrite and chalcopyrite also occur in small amounts and carry gold. Chlorite is invariably associated with gold values and is an excellent indicator of grade which varies roughly with the chlorite content of the fractures.

Visible gold is most rare, only four or five occurrences having been reported during the history of the mine. Microscopic study showed that 38 per cent was free in quartz-chlorite gangue, 18.9 per cent associated with arsenopyrite, 28.3 per cent associated with pyrrhotite and the remainder with pyrite and chalcopyrite. This confirms with the general experience at the mine that high-grade ore is found where heavy arsenopyrite-pyrrhotite mineralization is associated with vein quartz.



Drilling 'uppers' in narrow vein. Banding of iron-formation is clearly visible.

Structural Control

It is apparent that in this deposit the competent iron formation has played a role more commonly assigned to quartz. The ores are entirely related to tensional fractures in hard, brittle iron formation and these openings do not extend into the softer greenstones, which yielded to the deforming stresses by shearing. The fractures provided a path for the ascending mineralized solutions and it is probable that the fractures were open at the time of mineralization as vugs occur lined with gold-bearing sulphides. The chlorite apparently is pre-ore, probably altered greenstone, and its relationship to the ore bodies may be the result of its providing favourable chemical environment for gold deposition.

Many theories on structural control have been expressed by various geologists to account for the fracturing, localization of the fractures and their rake to the east. The writer's opinion is that it is more probable that faulting of the diabase dike to the east accounted for the movement which produced the fracturing in the iron formation (see figure 1). The following facts lead the writer to his opinion:

1. Evidence indicates that the iron formation in which the ore bodies occur was originally one long continuous iron band and due to the faulting of the diabase, which exerted a tensional pull to the northeast, it was drawn out into lenses. This tensional force also oriented these iron formation lenses so formed in a *en echelon* arrangement with their east extremities tending to point towards the fault (figure 3). The evidence of this tensional pull can be seen in figure 3 which indicates the drawing out of the iron formation between lenses, iron forma-

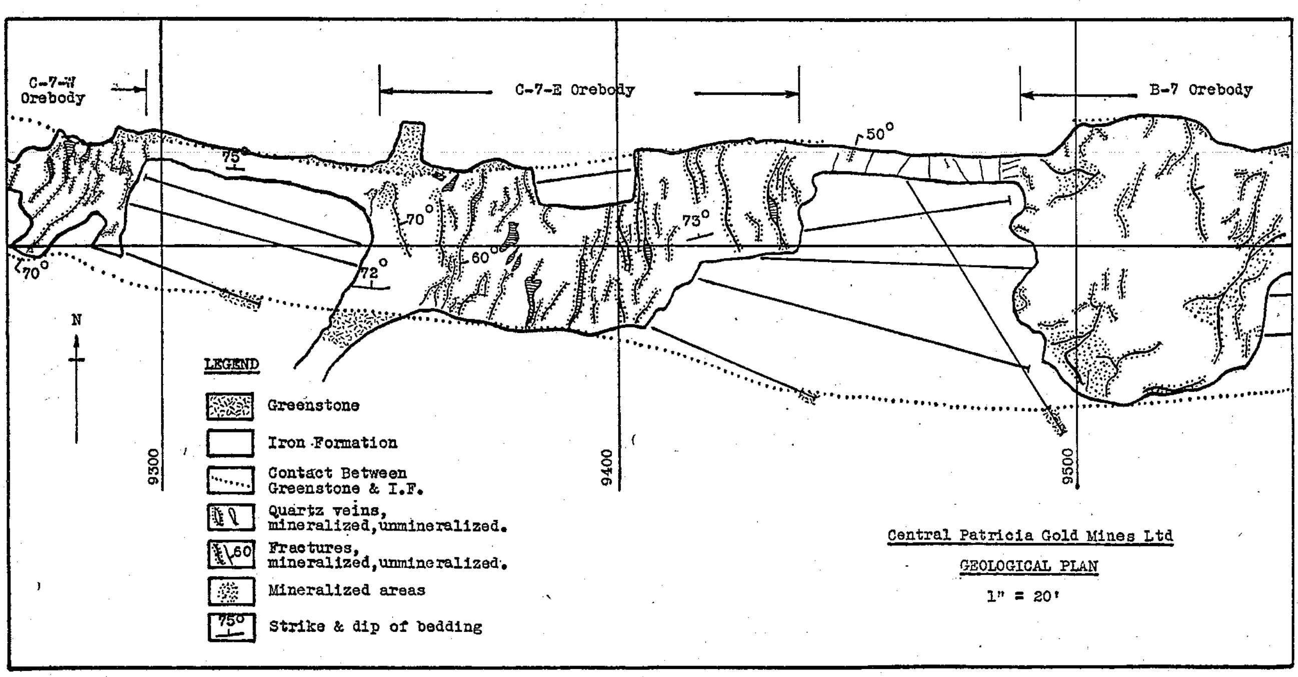


Figure 3.—Portion of geological plan of 750-foot level, showing fracture pattern which makes up the ore bodies at the Central Patricia Mine.

on fragmentals where it was ulled apart, and vugs, now filled ith quartz, between the lenses.

2. The intense shearing along ne hanging wall contact of the on formation in contrast to the elatively "tight" footwall indiates a concentrated movement on he north side of the iron band, n which side the diabase dike aulted. The iron formation bands orth of the fault were not afected when the dike faulted as hey are north of the fault movenent. The iron bands south of the nine were protected from the hearing action as the movement was deflected along the hanging wall contact of the iron formation in which the ore occurs.

3. The localization of the fractures along the length of iron formation in which the ore bodies occur in the Central Patricia mine and the reason why these ore bodies rake to the east can be explained as follows:

Localization. — When the diabase dike faulted maximum tensional force on the iron band was along a line at right angles from that end of the dike which did not move, and this tensional force diminished in strength in an anticlockwise direction from this right angle line as shown in figure 1. Experience has proven that a higher grade and greater tonnage of ore has been mined from the western half of the mine where this maximum tensional force resulted on the iron formation and produced a greater number of fractures.

Rake. — The dike fault being nearly vertical and the iron formation band dipping 73 degrees to the north the resultant tensional force produced when the dike faulted would act on this iron band at an angle of approximately 56 degrees to the horizontal, at which angle the ore bodies rake.

Ore Bodies

Due to the peculiar nature of their occurrence, ore bodies vary greatly in shape and size. Where only one or two fractures are being mined, the true length of the ore body must be considered as lying across the iron formation, but in most cases the total length of the ore body is relatively great compared to the length of the individual fractures and thus, in general, the length of an ore body is considered the dimension in the direction of strike of the iron formation, while the width of the ore body is the length of the individual fractures.

As noted before, the ore bodies rake to the east at 56 degrees, and down the rake at least one ore body has been traced for 2700 feet. It must be understood that the individual fractures do not attain these dimensions but succeed one another in parallel and *en echelon* arrangements; however, some of the strong fractures have been traced for an interval of 300 vertical feet.

Individual fractures carry high gold values with character samples usually showing assays of more than an ounce per ton. A glance at figure 3, however, will show that there is a comparatively large amount of waste rock between fractures, hence values are greatly diluted in actual mining practice. Individual, high-grade fractures are being mined, but the tonnage is minor.

Development

Development of the deposit must be carried out on somewhat unusual lines due to the nature of the ore bodies. In general, drifting is carried out at or near the hanging wall of the iron formation as experience has shown that the majority of the bodies originate here. However, all of the iron formation must be explored with

the diamond drill, and in carrying out this work it is necessary to exercise care that the holes are drilled about at right angles to the locally prevailing fracture pattern (figure 3). Normal drilling, at right angles to the drift, obviously might produce very false results.

Sampling must be carried out with due care and the results carefully analyzed. An advancing drift face may break exactly at a fracture, or between fractures and in either case it is necessary to take the structure in consideration when assessing the results. Much samples have proved the most reliable guide to probable mining grade but in the stopes chip samples, properly analyzed, are used as a guide to breaking.

In seeking for ore on new levels, the known tendency to vertical continuity is of the utmost importance and it is usually possible to predict the position of an ore shoot within a few feet.

In stoping, it is necessary to exercise care in order not to overlook possible "blind" fractures at either end of the workings. Diamond drilling must be carried out at regular intervals to test the ends of the stopes and, in case of wide iron formation, to test the walls for fractures which may not extend to the open workings.

Exploration

The iron bands in the vicinity of the mine have all been diamond drilled from surface but to date no occurrences of value have been found other than that being mined.

Besides having the usual machines and crews for underground exploration the Central Patricia Gold Mines Limited maintains a separate department for surface exploration of the main property and the examination of outside prospects and options. Underground exploration is under the direction of the engineering department while all surface work is supervised by an exploration engineer. The surface exploration department is fully equipped with large and small diamond drills, canoes and outboard motors, necessary camping equipment, etc., and organized so that a party can leave to examine a prospect within a few hours notice.

Mining at Central Patricia

by

E. Marchbank,[1] J.B.C. Lang[2] and T.T. Tigert[3]

THE CENTRAL Patricia mine has been developed by two vertical shafts, with level intervals at 125 feet to the 1000 level and at 150 feet from the 1000 level to the 3400 level.

No. 1 shaft services the mine from surface to the 2050 level. No. 2, an internal shaft, services from the 2050 level to the 3400 level, the lowest level of the mine. The two shafts are connected on the 2050 level by a haulage drift to No. 1 shaft ore and waste pockets.

Levels are developed by drifts and cross-cuts through the known orebodies. Subsidiary fracture zones are not developed until full information regarding them has been obtained by exploration diamond drilling.

The ore pass for No. 1 shaft starts at the 625 level and bottoms at the loading pocket below the 2050 level. The No. 2 ore pass is completed between the 2500 and the 2950 levels. The waste pass system consists of a series of open stopes and raises with gravity flow down to the 2650 level, and each level is equipped with a chute and dump hole, so that waste or gravel fill may be loaded or dumped at each level. A transfer door, hinged on the bottom, is installed opposite the chute for by-passing fill from one level to the next, and in a closed position protects the opening at the top of the pass. Waste rock below the 2650 level in the No. 2 shaft is hoisted and trammed either to a loading pocket at the No. 1 shaft, thence hoisted and dumped into the waste pass system on surface, or trammed to the waste pass on the 2050 level, depending on fill requirements.

Raises are driven in the orebodies after backstoping and stope timbering is completed. They usually go up the west rake of the orebodies at an inclination of 55 degrees, more or less, depending on the dip of the fracture zone.

Diamond drilling is an established part of the mine development program. At least one drill is operated two shifts per day for a total footage of approximately 1,100 feet per month. Most of the holes are short, their purpose being chiefly to test all the iron formation in the mine workings from the drifts, and the walls of current stopes, for gold bearing fractures. Occasionally, longer holes are drilled to test parallel iron formation bands for values and to secure geological data.

Shafts

No. 1 shaft handles the hoisting and servicing from surface to the 2050 level in a single stage. The shaft is 15 feet 3 inches by 6 feet 3 inches outside dimensions and consists of three compartments. No. 1 and No. 2 compartments are skipways and measure 5 feet 0 inches by 4 feet 6 inches while No. 3 compartment is 5 feet 0 inches by 4 feet 0 inches and serves as a manway and service compartment. Local 8-inch by 8-inch was used from surface to the 1000 level and 8-inch by 8-inch. B.C. fir has been installed at deeper levels. Guides are 4-inch by 6-inch dressed B. C. fir throughout the entire shaft, and bearing sets are installed below each level.

Hoisting of ore and servicing of levels is done by a skip-over-cage combination. The skip has a maximum capacity of 3.5 tons of ore, which is hoisted in counterbalance at the rate of 1135 feet per minute. Air-operated measuring and loading pockets have been installed below the 2050 level for both ore and waste. The cage has a maximum loading of nine men or 6,200 pounds. The headframe is a steel structure covered with shiplap, building paper, asbestoside and galvanized sheet iron. The 6-foot sheave wheels are 102 feet above the deck level.

The hoist is a C.I.R. 72-inch by 60-inch type PE-1 double-drum electric, machine-grooved for 1-1/8-inch rope and having a maximum rope pull of 21,000 lb. It is equipped with a 500-h.p., 720-r.p.m., 550-volt induction motor.

No. 2 shaft handles the hoisting and servicing of levels between the 2050 and 3400 levels. The shaft is 22 feet 5 inches by 6 feet 7 inches outside dimensions and consists of four compartments. No's 1 and 2 compartments are cageways, No. 3 is used for shaft sinking and spare hoisting compartment and No. 4 serves as a manway and service compartment. Cage compartments are 5 feet 0 inches by 4 feet 6 inches and manway is 5 feet 0 inches by 5 feet 0 inches. The shaft is timbered throughout by 9-1/2-in. by 9-1/2 in. B.C. fir and guides are 4-in. by 6-in. dressed B.C. fir. Bearing sets are installed below each level. Ore and waste is hoisted by single-deck cages in one-ton mine cars to loading pockets above the 2050 level.

The headframe is of timber construction and the 4-foot sheave wheels are 106 feet above the shaft collar. The hoist is a C.I.R. 42-

[1] Mine Captain.
[2] Shift Boss.
[3] Assistant Manager.

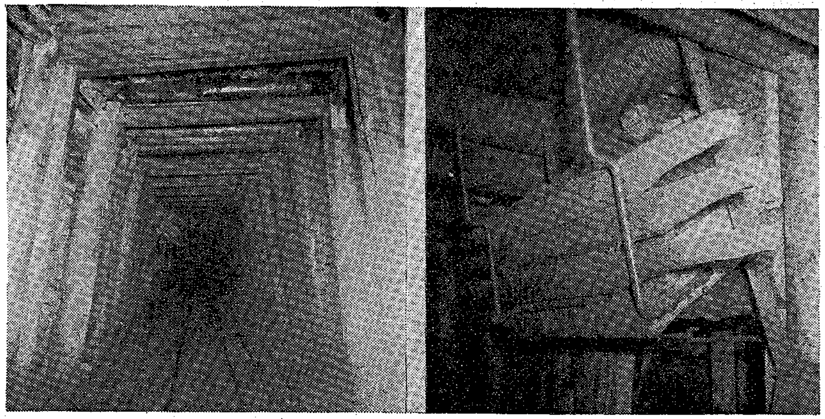
inch by 30 inch double-drum electric, machine grooved for 7/8-inch rope, having a rope pull of 8,000 lb. It is equipped with a 125-h.p., 875-r.p.m., 550-volt induction motor.

All shafts are closed to the levels by doors located beyond the shaft stations, to assist in controlling ventilation.

Drifts and Cross-cuts

Drifts and cross-cuts are driven on levels spaced at 125-foot intervals in the upper 1000 feet of the mine and at 150-foot intervals in the lower part. Drifts are generally driven in the iron formation, but along the hanging wall contact with the greenstone, as experience has proven that the majority of the orebodies originate here. Lateral headings are driven 6 feet wide by 7 feet high on a grade of 0.25 per cent in favour of the load. In later years, 16-lb. rail is generally used due to the advent of mechanical mucking and haulage, having a gauge of 18 inches, with ties at 24-inch intervals. Two-inch air lines and one-inch water lines are carried on the drift floor on the side opposite the ditch while an eleven-inch vent pipe is suspended on the upper north corner of the heading.

The burn cut is used exclusively in the mine. Two drillers, with two 3-1/2-inch automatic drifters mounted on one column bar which is equipped with a Purdie jack, drill off and blast a normal round of twenty-eight holes in one shift. As the iron formation varies in hardness and breaking qualities, depending on the magnetite content, trouble is experienced in breaking a full length of round so the cut and square are blasted separately. Mucking is done with Gardner-Denver loaders, into one-ton rocker side-dump mine cars, and Mancha motors are used for hauling.



Left: Filled drift timbering in wide ore-bodies. Right: Stope chute, showing method of protecting head-block against wear.

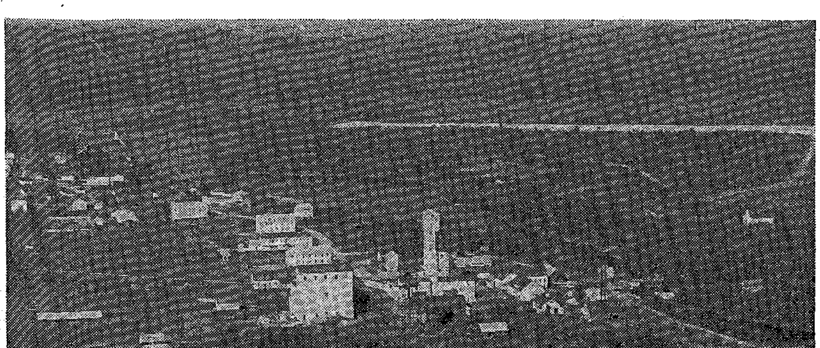
Powder used in lateral headings being 40 per cent Polar Forcite in the square and 55 per cent Stopeite in the cut. Black tape fuse, cut to lengths of ten feet, is dipped in red fuse paint on the ends as a safety measure when cutting for timing, and hot-wire lighters are generally employed.

Average advance is slightly better than 7 feet per round and powder consumption averages 14.3 pounds per foot.

Raises

Raises have been standardized as to cross-section and are 7 feet wide and 4 feet high. Two raise men, using two stopers, drill and blast a 21-hole burn-cut round in one shift, the average advance being 6.3 feet per round and powder consumption 13.5 pounds per foot. Raises, to complete the development of an orebody, are usually driven after the backstope and stope timbering is completed and the muck produced by raising drawn through the chutes provided for stoping operations. Whereever this is not possible, and the raise is to start from the drift back, the first 30 feet is taken 7 feet by 10 feet and divided by timber into a manway and a rock compartment, the rock compartment being equipped with a chute. Access to the face is maintained by the use of ladders hung over pins which are inserted into holes drilled in the bottom of the raise. Raise ladders are constructed of 4-inch by 4-inch timber sides with rungs made from discarded drill steel and they are 10 feet long.

Raises over 50 degrees are timbered but are driven in the conventional manner and size. A manway, carried up the footwall side, consists of stulls at 5-foot centres and covered either with 3-inch by 5-inch planks or lagging and contains the ladders, air and water pipes. A bulkhead is built over the top of the manway and an opening is left in the planks or lagging just below the bulkhead to permit entrance to the face. After each advance of approximately 25 feet, the bulkhead is removed and timber completed to within eight feet of the face.



General View, Central Patricia Mine. Pickle Lake Landing Strip is shown in upper right of photo.

Mining Methods

Introduction

In the early years of mining at the Central Patricia mine, the shrinkage stope method was exclusively employed but it was soon found that this method was not feasible for all orebodies, due to the structure of the fracture system in which the gold occurs. The orebodies consist of groups of mineralized fractures, which are close enough together to bring the grade of the whole mass up to ore standards, and the outline, in plan, of some of these orebodies is extremely irregular due to the discontinuous nature of the fractures and their varied length and dip. The hanging wall was found to be weak, and in many cases excessive dilution and dangerous conditions occurred as the stopes were pulled. No satisfactory method of removing sill-floor pillars was possible under these conditions without sacrificing the levels.

Cut-and-fill methods, more suitable to conditions in wide orebodies of irregular shape, were eventually adopted. Standard cut-and-fill methods such as are used with success at many other mines were introduced at relatively shallow horizons. It was found, however, that, on account of the structural conditions existing, a considerable amount of modification and experimentation was necessary in order to evolve a system which would apply at this mine. The cut-and-fill method employed embraced the horizontal slice, and no timber is required, other than a minor amount of temporary support of back and walls, until the stope back reaches a height of within 35 feet of the level above, and there, due to pressure acting on this sill-floor pillar, stope sets are brought into use for ground support to afford more protection to the men and for complete extraction of the ore. Close control and selective mining of long individual fractures occurring within the fracture group, which often give the stope a local width of 20 to 30 feet, is possible by this method. The average rake, or pitch, of the orebodies is 55 degrees to the east and dip is 72 degrees to the north.

Generally, orebodies averaging less than 10 feet in width and fairly uniform in shape are operated as shrinkage stopes. In those orebodies with an average width greater than 10 feet and having irregular shapes the cut-and-fill method is used.

During the life of the mine the production tonnage from the two types of stoping methods employed has averaged about the same. Due to average small size of the stopes at least twenty-five stopes must be kept in production to maintain an average milling rate of 400 tons per day.

Stope Preparation

Preliminary stope preparation is similar for both shrinkage and cut-and-fill methods in use, apart from such variations as are compelled by ground conditions. Before backstoping, the ore is slashed out to its full width on the drift for the entire length of the section to be mined, following the information given by drift plans available in the Engineering Office. The backstopes are then taken up on this section to a height of 19 feet above the rail in the drift and this operation usually commences at that end of the orebody furthest removed from its entrance. Starting thus reduces the possibility of choking the entrance to the workings, and of mucking before breaking is completed. The pipe lines and ventilation ducts can be removed as the crews work back towards the entrance. Carrying two breasts, the lower about 25 feet ahead of the upper, increases the efficiency and time element of the breaking operation.

In abnormally wide sections where ground conditions are bad, backstoping is started at the entrance to the orebody and a complete cycle of breaking, mucking, and timbering is followed for every three breasts, and the backs and walls supported from the completed timber.

The mucking of backstopes is accomplished with the use of a Gardner-Denver, type D-9, loader, one-ton side-dump mine cars, the hauling being done with Mancha battery locomotives. Before breaking down the backstope, track is layed to encompass the whole area of the new stope to provide efficient mucking conditions.

The timber sets, installed at 5

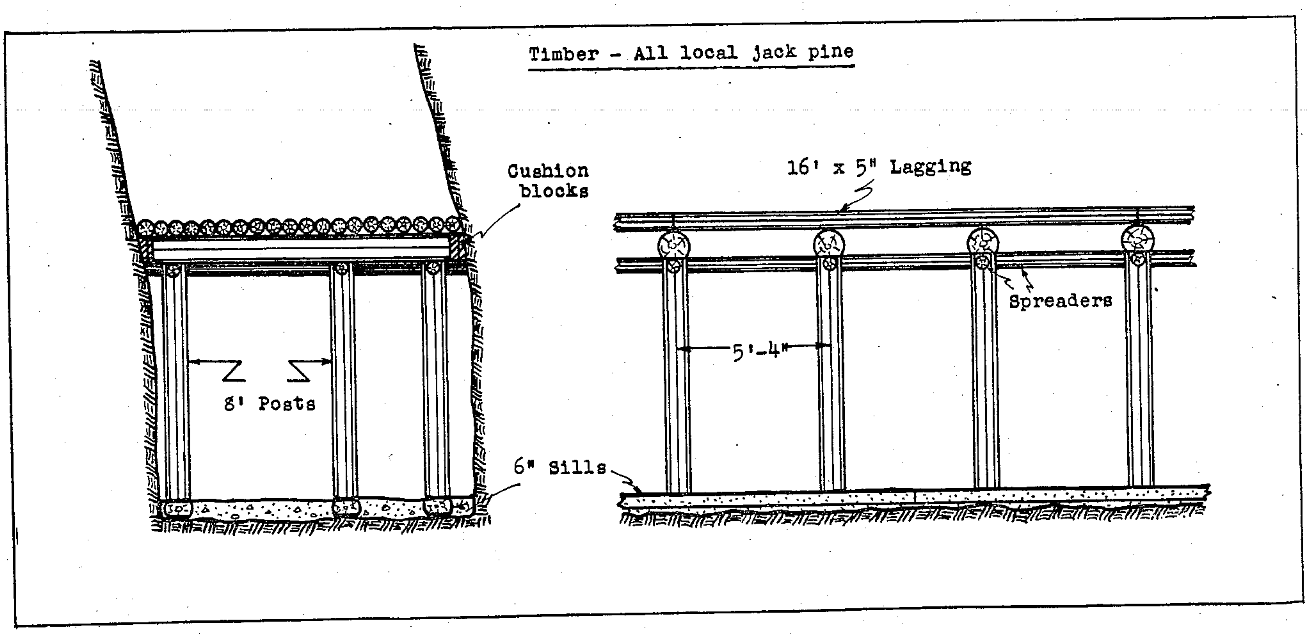


Figure 1.—Stope Drift Timbering when floor pillar has not been extracted.

feet 4 inch centres, are covered with lagging extending across four caps. Cushion blocks of 6-inch flat timber are placed on both ends of the caps to relieve compression due to closing movement of the walls. For narrow stopes of drift width, timbering of backstopes consists simply of stulls placed across the drift. Stopes that are greater than drift width and where the sill-floor pillar beneath the stope has not been extracted are timbered as shown in figure 1. Where the sill-floor pillar has been extracted the stope timber is installed as illustrated in figure 2. Chutes are located on the footwall side of the orebodies, and are constructed of 6-inch square timbers with the wearing surfaces armoured against wear and blasting with 3/8-inch plate. The timber headblock is wound with discarded hoisting rope to prevent breakage and wear (see photos on page 77).

In the early life of the mine the chutes in cut-and-fill stopes were spaced close together to facilitate hand-mucking, but with the advent of slusher hoists and scrapers, two chutes were found to be adequate to handle the draw from stopes up to 200 feet long. The average distance between chutes in shrinkage stopes is 35 feet starting with the first chute being right at the west rake of the ore-shoot.

To complete the development of an ore-shoot between levels and to prepare it for mining, a raise is driven up the west rake from the backstope to the level above and subsequently partitioned off into a manway and a fill compartment with stulls and 3 in. by 5 in. planks, in the case of a cut-and-fill stope, or ladders and landings for a shrinkage stope. The 3-inch by 5-inch planks installed for partitioning a raise in a cut-and-fill stope are eventually used as flooring on the fill. To facilitate drawing the broken rock produced by the raise in cut-and-fill stopes a temporary chute is constructed at the west rake.

Horizontal Cut-and-Fill Stoping

The orebodies mined using this method range from a maximum length of 320 feet down to a minimum of 55 feet, and vary in average width from 16 to 17 feet. Widths in any one stope may vary from 3 to 30 feet and average about 14 feet. Locally, the structure of the ore-shoots is very irregular, with at least one end tapering off into a long, narrow finger. This irregularity of the ore necessitates the complete flexibility that is possible in horizontal cut-and-fill stoping.

The orebodies predominately occur along the hanging wall contact between the iron formation and greenstone, and the greenstone hanging wall exposed in mining is schisted and fractured, so it is desirable to break cleanly to this contact, not only for reasons of dilution, but to prevent loose and dangerous conditions. The iron formation, in which the ore occurs, contains numerous slips cutting across the formation at right-angles and dip at 5 to 10 degrees to the west, and unless breasting is carried out from west to east, and allowing a slight arch across the width of the breast, loose backs occur.

The first slice in a stope is broken down with the use of stoper machines as vertical holes greatly lessen the danger of damage to stope timber. The ore is mined from wall to wall in horizontal, flat-back slices, commencing at the raise nearest the west rake and progressing along the length of the stope with a breast 7 feet high. Normally, two slices are taken down for the full length of the stope before mucking out to fill, but occasionally only part of one slice can be taken, due to bad ground. On approaching within 35 feet of the level above it is found necessary to take only two or three breasts at a time; then muck and fill, as the wall pressure begins to crush the sill-floor pillar causing loose backs, which must be timbered as the breaking advances.

Breasts are drilled with horizontal holes 9 feet long and spaced at 2-to 4-foot centres, depending on the width of the breast at any particular section. The ore is blasted on a 3 inch by 5 inch plank floor placed on top of the filling. When breaking is completed the

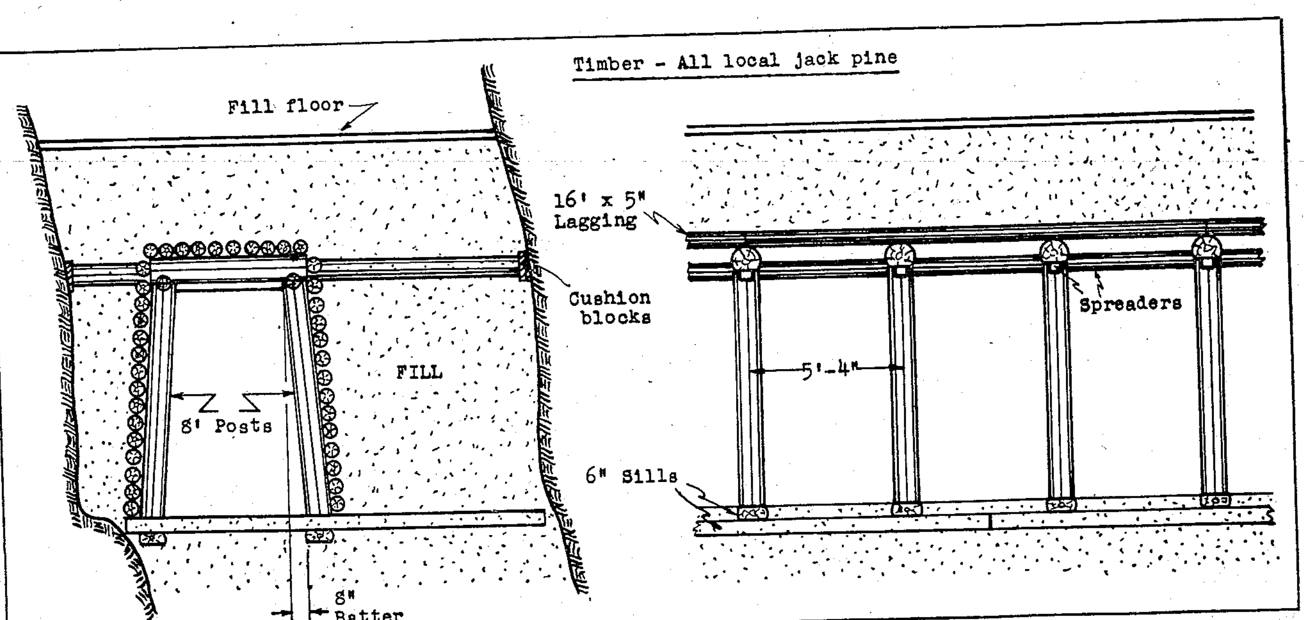


Figure 2.—Stope Drift Timbering after floor pillar has been extracted.

broken ore is mucked out employing Gardner-Denver 15-horsepower double-drum slusher air-hoists. Scrapers are of the hoe type, 36 inches wide, constructed of cast manganese steel and reinforced along the wearing edges with discarded drill steel. As the ore is mucked out the floor is lifted, millholes raised to within 8 feet of the stope back, and fill rock or gravel trammed to the fill raises as required.

To supply the fill requirements for cut-and-fill stoping it has been necessary to truck gravel from a pit located a half-mile distant from No. 1 shaft. Utilizing a C.I.R. 35 h.p. electric double-drum slusher hoist, a box-type scraper and a measuring and loading hopper at the pit, a four cubic yard truck averages 200 cubic yards of gravel delivered to the waste pass system in 12 hours, at a cost of twenty-five cents per yard.

Combination cribbed millholes of pre-dapped 6-inch lagging, which form an 8-foot by 5-1/2-foot millhole and a 3-1/2 foot by 5-1/2-ft. manway, usually slope to the west at the same degree as the rake of the ore-shoot, and as the wear on the bottom lining is far greater than on the sides it is necessary to place the 3-in. by 5-in. lining planks on edge for two-thirds of the length of the millhole so that the bottom will withstand the wear for the production life of the stope. The manway compartment is chinked on the outside with slabs hauled from the company sawmill. Millholes are carried up evenly spaced from the ends of the stope until they reach a point about 40 feet below the next level, then they are steepened to bring the east millhole out to the extreme east end of the stope to provide a draw point for mucking until the last breast is taken out of the sill-floor pillar.

The same miners complete the whole cycle of operations: breaking, timbering, mucking and filling. The crew consist of one runner and one helper on each shift.

When a stope has reached the position where the sill-floor pillar is about 14 feet thick, and where there is stope timber on the level above, the mining method must be altered to prevent the stope above from collapsing, and to provide more protection to the men and for complete extraction of the ore.

First, it is necessary to timber up the stope backs from the fill floor with sets at 5-foot centres before any breaking is undertaken. The sill-floor pillar is then extracted, taking one or two breasts at a time depending on the width of the ore. After blasting down a breast, or breasts, the next breast is drilled off but not blasted then all the muck slushed out. A vertical barricade is then constructed, placing stulls horizontally at 8-foot centres faced with upright gob-lagging, and fill run in. The stulls are tied back at each end and in the centre with discarded hoisting rope for additional safety and these barricades are installed about 3 feet from the breast to be blasted. The drift timbers of the stope above are supported by 10-inch by 12-inch by 24-foot long B.C. fir booms spanning from the top of the fill at the level to the top of the solid sill-floor just past the breast blasted

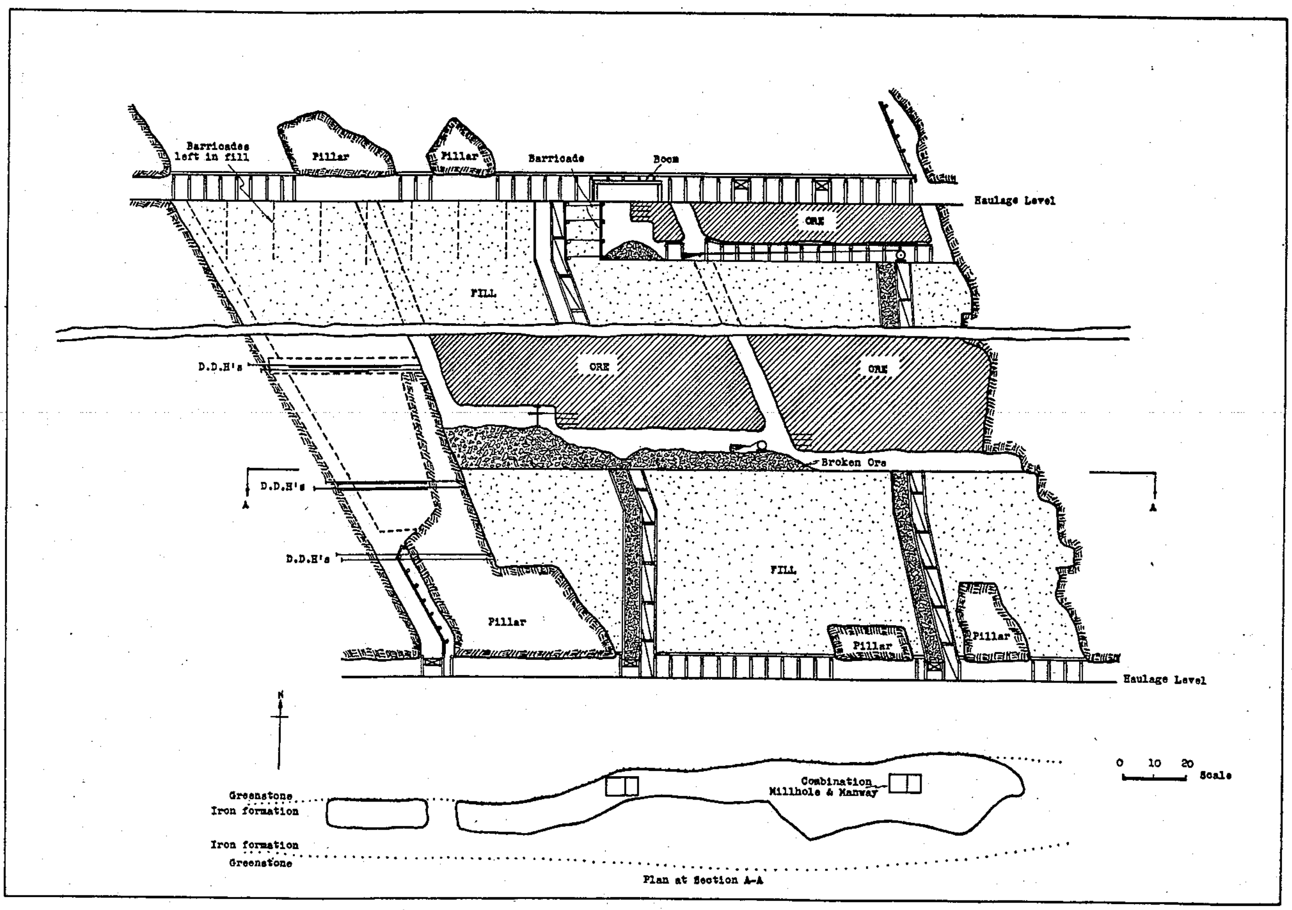


Figure 3.—Horizontal Cut-and-Fill Stoping. Lower portion shows normal stoping operations. Upper portion shows method of extracting floor pillar.

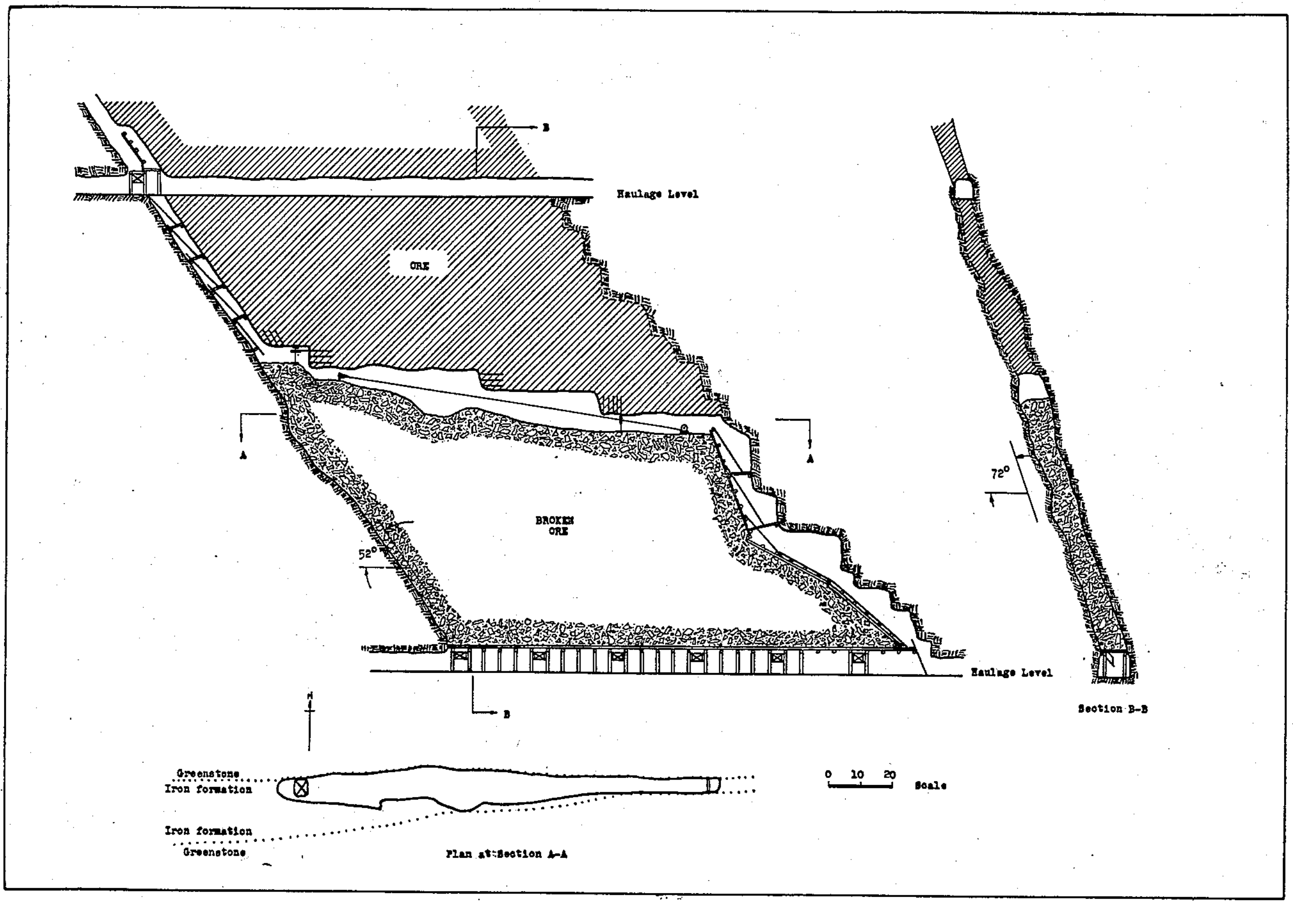


Figure 4.—Shrinkage Stoping at the Central Patricia Mine.

out, the number of booms depending on the twidth to be supported. The timber set in the stope directly under the next breast to be extracted is blasted out and the timber snaked out, using the slusher cable.

An orebody being mined by the cut-and-fill method is illustrated in figure 3.

Shrinkage Stoping

The orebodies mined by this method have ranged from a maximum length of 180 feet down to a minimum of 25 feet, and widths vary from 10 to 3 feet. Maximum width at any one point must not exceed 15 feet, and in plan, must be fairly regular in shape for an orebody to be mined by the shrinkage stope method. Both leyners and stoper machines are used in breaking, depending on the width, the horizontal holes being 9 feet long and the vertical holes 7 feet in length. The ore is mined from wall to wall in horizontal slices seven feet high commencing at the west raise. Carrying two to four breasts at any one time improves efficiency and the time element in breaking operations. A typical shrinkage stope is shown in figure 4.

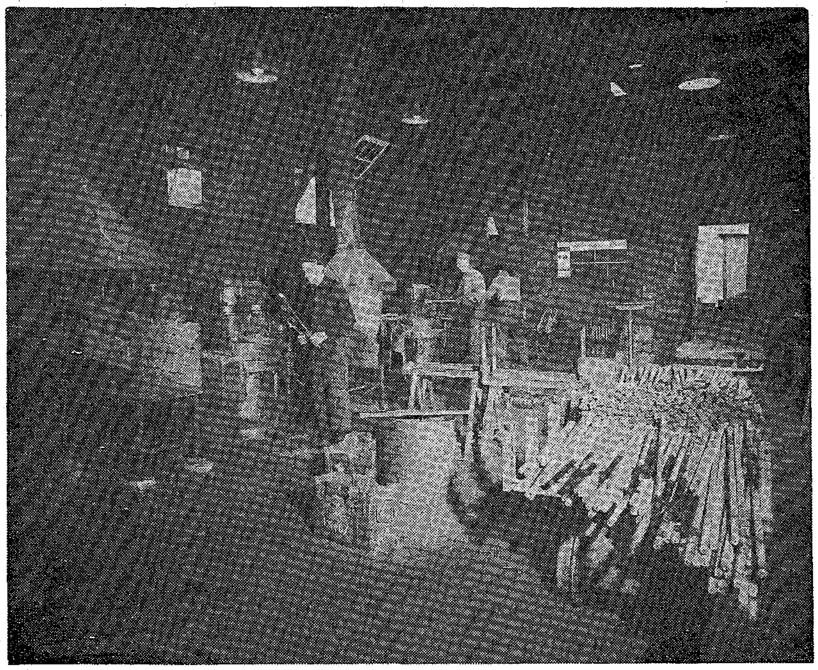
Generally, when stoping has advanced upward to a height of about 90 feet above the level slushers-hoists and scrapers must be put to use to keep a working height at the west end of the stope as the broken ore here will not draw to the chute on the west rake of the orebody due to the flat rake. A manway, fabricated from heavy stulls and 4-inch lagging, is maintained up the east rake of the stope. As shrinkage stopes are narrow the sill-pillar can be extracted from the top of the ore in the stope with stiff-leg trusses supporting the drift timbers of the stope above.

Because of the flat rake of the stopes to the east, and a dip to the north, slushers and scrapers play a very vital part when drawing the muck from a shrinkage stope. Liberal use of stulls, in conjunction with cushion blocks, overcomes a major difficulty in supporting the hanging wall as the much is removed. (photo p. 82). The cushion blocks not only serve as a relief to compression but are long enough to cover a large area, thus holding small pieces of loose from falling. The stulls are installed at right angles to the sloping walls to prevent them from snapping out due to closure of the walls.

Methane Gas

Methane gas has been frequently encountered in the underground workings of the Central Patricia mine, in both sedimentary and volcanic rocks, so safety education, to develop a spirit of safety conciousness and alertness in recognizing the hazards of this gas, and the provision of specific instructions as to safe practice, is a constant and essential job.

Methane has been encountered for the whole length and breadth of the underground workings below the 1300 level, but does not follow any conceived pattern or zone. Most frequent emanations are from diamond drill holes as these are cutting across geological faults, fractures and contacts. Methane is rarely encountered in stopes so it is assumed that lateral developments and raises tend to



Bit re-conditioning and shank-forming shop. The shop is situated at Central Patricia and is operated jointly by Central Patricia Gold Mines Limited and Pickle Crow Gold Mines Limited.

bleed the stoping areas of gas. It is also possible that during blasting operations in stopes the shattering effect releases any nearby gas and that this gas is carried away by the mine ventilation air currents.

No open crevices have ever been encountered in the rock, but it has been more or less proven that the pockets of methane are associated with faults, fractures and contacts under great pressure. When these pockets are encountered by any type of drill hole liberation is usually rapid. A few holes have produced 'bleeders' of methane; these liberate gas slowly and must be watched closely, by making repeated tests in nearby high-spots, especially backstopes and dead-end raises. Only one occurrence has produced any great volume of methane where complete liberation through one drill hole lasted two weeks.

To provide a means whereby an atmosphere may be quickly and conveniently tested for methane all shift-bosses are equipped with M.S.A. type E-2 methane testers. The flame-safety lamp, commonly used for this purpose in coal mines, is not considered sufficiently accurate at low concentrations and is definitely not a safe instrument except in experienced hands. The methane encountered is practically always accompanied by small concentrations of hydrogen sulphide. The presence of the former gas can usually be detected by the odour of the latter.

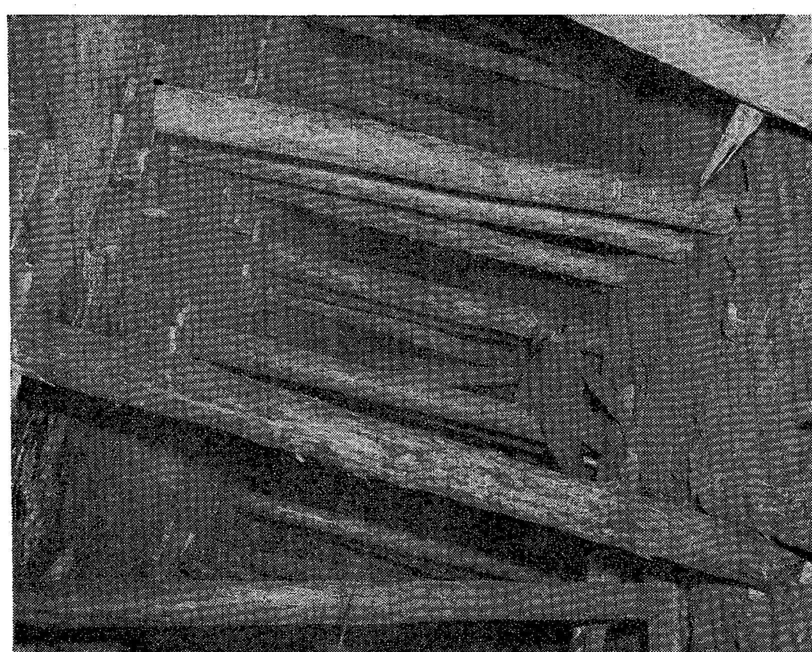
The chief danger from methane lies in the range of explosive mixtures which it forms with air. Air that contains 5 to 14 per cent of methane will explode if brought in contact with an igniting medium, such as an open flame or an electric arc. The best and surest precaution against accidents from ignitions of explosive mixtures of methane and air is to prevent the accumulation of such concentrations by dependable and adequate ventilation that will remove the gas.

Ventilaiton

Controlled ventilation, besides removing methane and carbon monoxide gas from workings and furnishing comfortable air conditions so that efficient work can be done, may also be important as the means to save the lives of men underground in case fire of any origin ignites mine timbers. In this connection, printed instructions, entitled 'Organization and Procedure in the Event of an Underground Fire', are issued to all employees when hired.

Mechanical ventilation is provided at the Central Patricia mine by a 54-inch two-stage axial flow fan located on surface. This unit downcasts air through the ore zone, via raises, stopes, drifts and manways, down to the 3100 level. Return air is then upcasted to surface through the two shafts in series interconnected by the 2050 cross-cut. Normal capacity of the fan is 40,000 cubic feet per minute; but this is cut to 25,000 cubic feet per minute during the winter months to prevent excessive icing in the upper levels.

To control the distribution of



Timbering in a Shrinkage Stope at Central Patricia.

the air currents, doors have been installed on all levels. Experience has taught us that these stoppings must be strongly constructed and that employees must be educated to keep them closed. The doors are of three-ply wood construction, totally covered with galvanized sheet iron, and are fitted with the McIntyre type of latch, to keep them closed under all pressure condtions. Door frames are either of timber or concrete construction, depending on the amount of traffic. These ventilation doors are finished off with a good paint job, hi-visibility yellow on the doors with bright red frames. "Keep closed" signs fabricated from fluorescent "Scotchlite" sheeting, are fitted to both sides of the doors.

An auxiliary fan located in the fresh air circuit on the 2950 level delivers at least 6,000 cubic feet per minute of air through 22-inch galvanized iron vent pipe to development work below this horizon. Auxiliary blowers are installed, with metal or flexible tubing, to ventilate dead-end drifts and cross-cuts and, where the length of the line is too great for a single blower of the type available at the mine, fans are used in series.

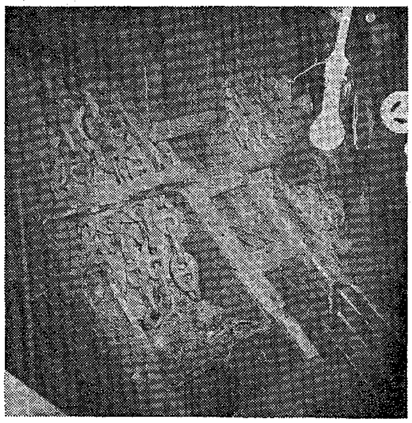
Below the 2050 level it is standard practice to drive raises along diamond drill holes where methane gas is liable to be encountered. Air is blown in the collars of these diamond drill holes by means of piping connected to the compressed air lines.

In shaft sinking experience has proven that fans should be operated as blowers, not as exhausters. For temporary use over short periods of time auxiliary ventilation by blowing compressed air is the quickest form available. This is often used, to dilute and remove methane gas.

Miscellaneous

Air. — Compressed air is delivered to No. 1 shaft at 100-lb. pressure from four compressors. Together, they are capable of supplying 3,000 cu. ft. of air per minute through three mains carried down the shaft to the 1300 level. Through a 6-inch main the air is carried from here to the 3400 level.

At each level station, 3-inch air



Loading Station, 2100-foot level, No. 1 Shaft.

lines take off from the lines in the shaft and are carried as far as the heads of the cross-cuts. Two-inch air lines supply air to drifts, raises, stopes, etc.

Water. — Water used underground is obtained either from the water tank on surface or from dams which trap water on the upper three levels of the mine. Pressure relief boxes are placed in the shaft every two levels, and float valves keep the boxes normally full. Main supply lines are of 2-inch pipe and the practice is to carry 1-inch lines along the drift and cross-cuts, with 3/4-inch lines off to stopes and raises.

Power. — Electric power for the mine is obtained from the Ontario Hydro plants at Ear Falls and Rat Rapids. It is delivered to the substation at the mine at 22,000 volts where it is stepped down to 2,200, 550 and 110 volts as required.

Drainage. — As the underground workings parallel the Crow River and dip under it, the first four levels can be classed as wet. Considerable seepage is encountered from faults, fractures and diamond drill holes. The water is trapped in suitable places by dams, and either piped or laundered by gravity flow to the shaft. Two-inch drainage lines in the shafts collect the water from each level and deliver it to the sumps and pumping stations on the 500, 1000, 1450, 2050, 2650 and 3400 levels. Sumps are provided with settling tanks and are of various sizes and types depending on requirements.

Engineering

This department, consisting of two engineers, maintain complete up-to-date records which are kept available for instant correlation. Besides their usual duties of supervision of sampling, diamond drilling, geology and surveying, this department compiles a great deal of useful operating data. This is tabulated monthly and is used for a unit cost system, which supplies a yard-stick for the whole operation.

Latitude co-ordinates of the mine plans are oriented due east and west, which is the general strike of the iron lenses. Departure co-ordinates thus afford a true cross-section of the mine.

Sampling

Sampling routine at the Central Patricia mine is much the same as elsewhere. The sampler inspects each working place that has been blasted, and sectionalizes and samples normal to the dip of the fractures. Chip samples only are taken. The extreme irergularity of the ore necessitates frequent and careful sampling of stope backs and walls. New orebodies are frequently bulk-sampled by taking down a complete slice over the whole area to secure a true grade picture.

The ore boundaries are limited by assay walls, except in places where mining is carried to the greenstone contact. Visible gold is extremely rare. Muck samples are taken from all development faces and also of ore drawn from stope chutes.

Milling and Metallurgy at Central Patricia Gold Mines

by

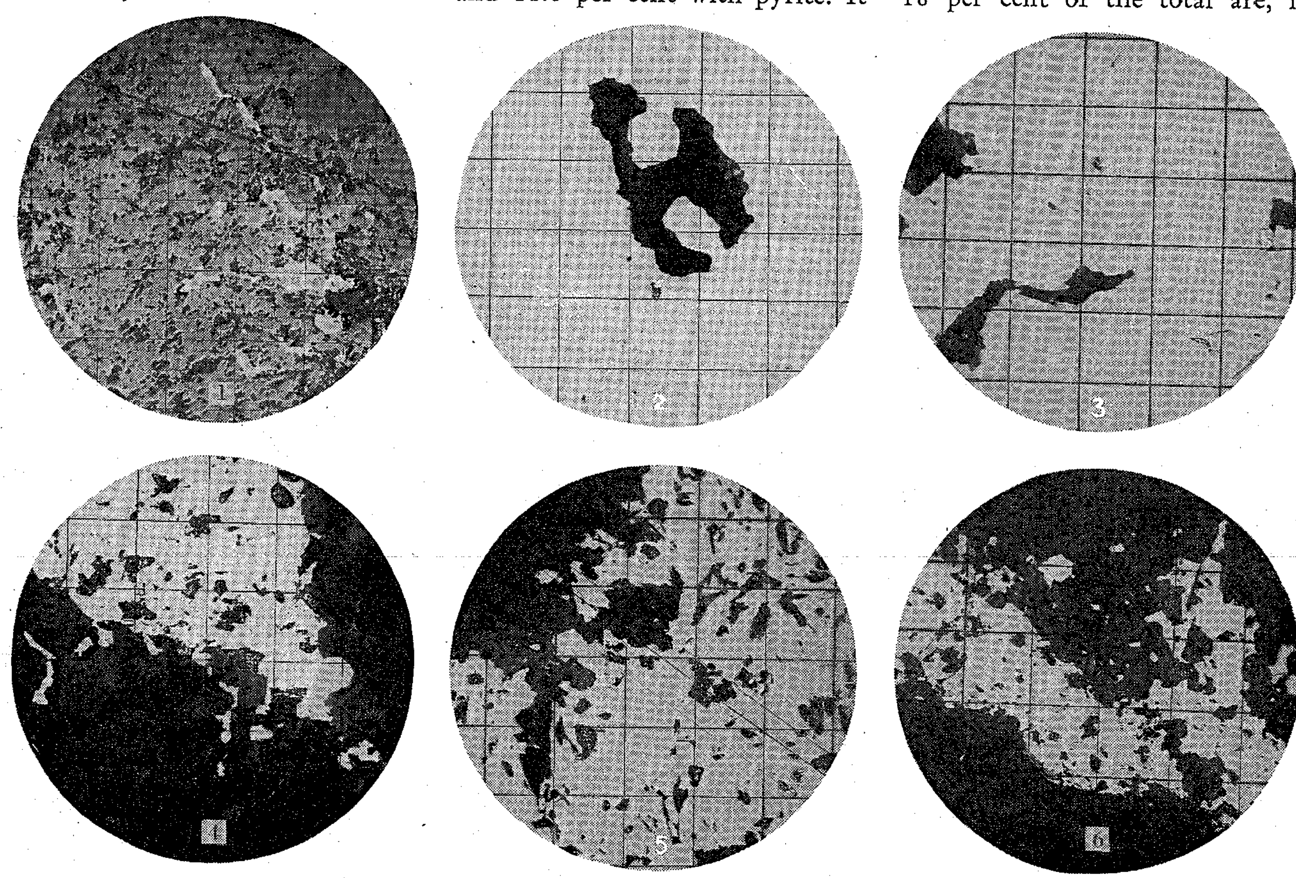
D. L. McCann*

THE extraction of 96 per cent of the gold inthis Central Patricia high sulphide ore is accomplished by a straight cyanide process at a direct milling cost of $1.08 per ton. (1948)

As noted previously, the ore bodies consist of sulphide concentrations in cross fractures in an iron formation with selective samples running well over one ounce grade. The gold, which is native and very fine, is primarily associated with the sulphides being found in the gangue matrix, along the borders of and within the sulphides. This is shown in the photomocrographs, figures 1, 2, 3, 4, 5, and 6. As may be seen from Table 1, the combined grain analysis of five samples showed 38 per cent of the gold to be free in gangue, 28.3 per cent to be associated with pyrrhotite, 18.9 per cent with arsenopyrite and 11.6 per cent with pyrite. It is also noted that only 4.9 per cent was larger than 150 mesh while 11 per cent was under 1600. A microscopic examination of the mill tailings, (Federal Bureau of Mines microscopic report dated June 20, 1939) showed that some of the gold was also in submicroscopic form.

The gangue minerals consist of carbonates, (ferrous) silicates quartz and magnetite while the sulphides which comprise 11 to 18 per cent of the total are, in

*Mill Superintendent.



Photographs. Bureau of Mines, Ottawa.

Mineralogical Characteristics of Central Patricia Ore.

Figure 1, native gold in gangue, native gold — yellow, gangue — gray, pits — black, 200-mesh grid; Figure 2, native gold and gangue in arsenopyrite, native gold — yellow, arsenopyrite — white, 200-mesh grid; Figure 3, extremely fine native gold in dense arsenopyrite, native gold — yellow, arsenopyrite — white, gangue — black, 200-mesh grid; Figure 4, native gold in gangue and associated with pyrrhotite, gold — yellow, pyrrhotite — light gray, gangue — black, 200-mesh grid; Figure 5, native gold along border of and within pyrrhotite, native gold — yellow pyrrhotite — light gray, gangue — black, 200-mesh grid; Figure 6, very fine native gold associated with pyrrhotite, native gold — yellow, pyrrhotite — light gray, gangue — black, 200-mesh grid. All magnifications x 120.

order of their abundance, pyrrhotite, arsenopyrite, pyrite and a little chalcopyrite. With the exception of pyrrhotite which is very unstable being of a low order with formulae Fe_5S_6 and which continues to oxidise very rapidly in the finely ground state, the sulphides in this ore are quite stable and present no particular problems.

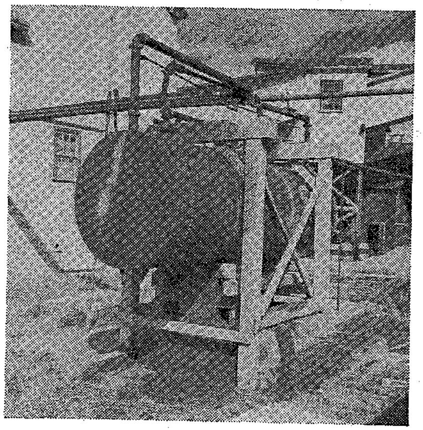
The chemical analysis of several samples showed the following to be present:—

Iron, Ferrous	20%
Total Iron	22%
Arsenic	0.5 to 1.8%
Sulphur	5 to 7%
Lead and Zinc	Trace
Magnesia	2.6%
CaO	5.9%
Insoluble	45 to 46%

Crushing and Conveying

The reduction of the ore to ball mill feed size is accomplished in two stages. The primary breaker is a 12 inch x 24 inch Allis Chalmers Superior (Blake Type) jaw crusher fitted with a non-choke swing jaw plate and driven by a 35-h.p., 550-volt, slip-ring motor, while the secondary is an Allis Chalmers No. 7, Style B, Newhouse gyratory crusher set to 3/4 inch and driven by a 60-h.p. 550-volt, 450-rpm motor and operating in closed circuit with a 3 foot x 6 foot Niagara screen.

The capacity of this crushing



Barren Solution Cooling Arrangement.

plant was doubled to 55 tons per hour in 1942 by the simple expedient of changing the 1/4-inch throw eccentric on the secondary crusher to a 3/8-inch, altering the screen openings to conform with the new setting, and speeding up the necessary conveyors by changing the sprocket sizes.

Details of the Crushing Plant

The ore which has passed through a 12-inch grizzley is hoisted in three ton skips into a 75-ton skip bin, from which it is drawn by an Allis Chalmers 30-inch x 7-foot Utah Type magnetic feeder and discharged on the sorting belt. Here the major portion of tramp timber and steel is removed and any over size pieces of muck broken to jaw crusher feed size. From the sorting belt it is conveyed to a 180-ton steel coarse ore bin which discharges onto the jaw crushed feed conveyor.

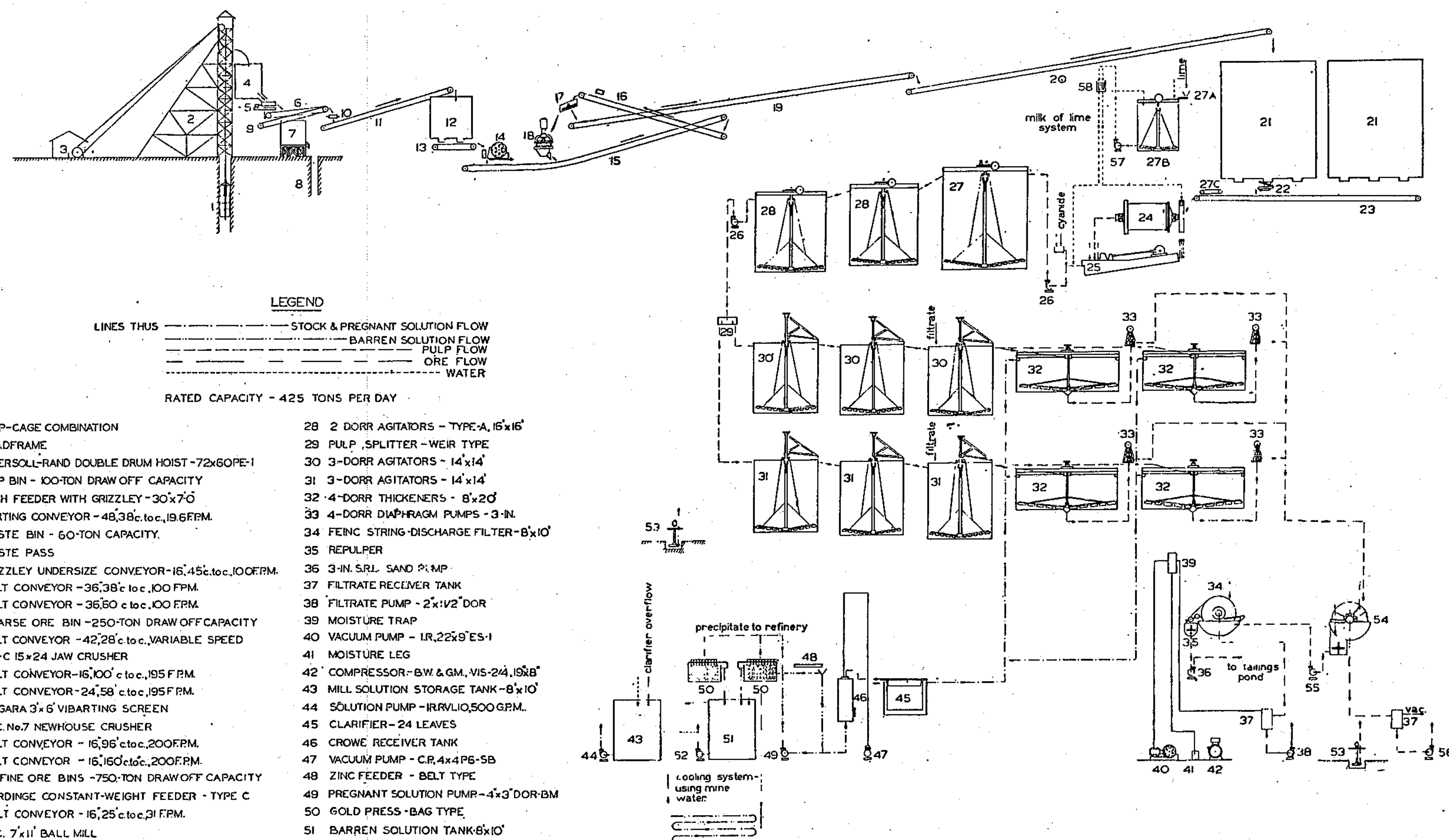
The product from the jaw crusher is conveyed to the screen past a magnetic pulley and a stationary magnet. These remove the rest of the tramp steel and a portion of the barren magnetite which is detrimental to the cyanide circuit. The screen oversize falls directly to the secondary crusher whose discharge joins the product from the primary breaker and is returned to the screen. The undersize is conveyed 300 feet to the 1000-ton (750 ton draw-off) fine ore bin. This bin, a 26-foot diameter x 30-foot wood stave tank, has three draw-off points located along the ball mill feed conveyor.

The wearing parts of both crushers, jaw plates, concaves and mantles are of manganese steel while the cheek plates are "Ni-hard." The screens which have an average life of 70 days are 3/8-inch double crimped high carbon steel with 3/4-inch x 1-1/2-inch openings.

The Newhouse crusher drive shafts which were found to fail rather quickly once the 3/8-inch throw eccentric was installed are now made from "Atlas SPS 245" alloy steel with the splines flame

TABLE 1.—GRAIN ANALYSIS OF NATIVE GOLD IN THE FIVE SAMPLES COMBINED
(Extract from the Canadian Department of Mines Minerographic ReportM-395)

Mesh		Free in Gangue %	Associated with Sulphides							Totals %
			With Arsenopyrite %			With Pyrite %				
			In dense Arseno pyrite	In Cracks	Against Arseno-pyrite	In Dense Pyrite	Against Pyrite	With Pyrrho-tite	With Chalco-pyrite	
Minus	Plus 150	2.2	2.7							4.9
150	200	1.9	1.8		2.0		3.8		2.2	11.7
200	280	1.6	2.8					3.8		8.2
280	400	3.1		1.1		0.7		7.1		12.0
400	560	6.4	2.1			1.3	0.8	4.1	0.7	15.4
560	800	7.6	2.1			1.1	0.6	3.8		15.2
800	1100	4.7	1.6		0.3	1.3		3.3	0.3	11.5
1100	1600	3.9	1.2			1.2	0.2	3.6		10.1
1600	2300	3.6	0.4		0.3	0.2		1.4		5.9
2300		3.0	0.5			0.4		1.2		5.1
Totals		38.0	15.2	1.1	2.6	6.2	5.4	28.3	3.2	100.0
			18.9			11.6				



LEGEND

LINES THUS
—·—·—·— STOCK & PREGNANT SOLUTION FLOW
———————— BARREN SOLUTION FLOW
— — — — PULP FLOW
—— —— —— ORE FLOW
- - - - - - - WATER

RATED CAPACITY - 425 TONS PER DAY

1 SKIP-CAGE COMBINATION
2 HEADFRAME
3 INGERSOLL-RAND DOUBLE DRUM HOIST - 72x60PE-1
4 SKIP BIN - 100-TON DRAW OFF CAPACITY
5 UTAH FEEDER WITH GRIZZLEY - 30"x7'0"
6 SORTING CONVEYOR - 48,38'c.to c.,19.6 F.P.M.
7 WASTE BIN - 60-TON CAPACITY.
8 WASTE PASS
9 GRIZZLEY UNDERSIZE CONVEYOR - 16,45'c.to c.,100 F.P.M.
10 BELT CONVEYOR - 36,38'c to c.,100 F.P.M.
11 BELT CONVEYOR - 36,60'c to c.,100 F.P.M.
12 COARSE ORE BIN - 250-TON DRAW OFF CAPACITY
13 BELT CONVEYOR - 42,28'c.to c.,VARIABLE SPEED
14 C.A-C 15x24 JAW CRUSHER
15 BELT CONVEYOR - 16,100'c to c.,195 F.P.M.
16 BELT CONVEYOR - 24,58'c.to c.,195 F.P.M.
17 NIAGARA 3'x6' VIBARTING SCREEN
18 A-C. No.7 NEWHOUSE CRUSHER
19 BELT CONVEYOR - 16,96'c.to c.,200 F.P.M.
20 BELT CONVEYOR - 16,160'c.to c.,200 F.P.M.
21 2 - FINE ORE BINS - 750-TON DRAW OFF CAPACITY
22 HARDINGE CONSTANT-WEIGHT FEEDER - TYPE C
23 BELT CONVEYOR - 16,25'c.to c.,31 F.P.M.
24 A-C. 7'x11' BALL MILL
25 D.S.F. DORR CLASSIFIER - 8'0"x25'0"
26 5-IN. S.R.L. SAND PUMP
27 DORR AGITATOR - TYPE-AX 22'x26'
27A VIBRATING FEEDER
27B AGITATOR - 5'x6'
27C LITHARGE FEEDER
28 2 DORR AGITATORS - TYPE-A, 16'x16'
29 PULP SPLITTER - WEIR TYPE
30 3-DORR AGITATORS - 14'x14'
31 3-DORR AGITATORS - 14'x14'
32 4-DORR THICKENERS - 8'x20'
33 4-DORR DIAPHRAGM PUMPS - 3-IN.
34 FEINC STRING-DISCHARGE FILTER - 8'x10'
35 REPULPER
36 3-IN. S.R.L. SAND PUMP
37 FILTRATE RECEIVER TANK
38 FILTRATE PUMP - 2"x1 1/2" DOR
39 MOISTURE TRAP
40 VACUUM PUMP - I.R.,22x9"ES-1
41 MOISTURE LEG
42 COMPRESSOR - B.W. & G.M., VIS-24,19x8"
43 MILL SOLUTION STORAGE TANK - 8'x10'
44 SOLUTION PUMP - I.R.RV,L10,500 G.P.M.
45 CLARIFIER - 24 LEAVES
46 CROWE RECEIVER TANK
47 VACUUM PUMP - C.P.,4x4 P6-SB
48 ZINC FEEDER - BELT TYPE
49 PREGNANT SOLUTION PUMP - 4"x3" DOR-BM
50 GOLD PRESS - BAG TYPE
51 BARREN SOLUTION TANK - 8'x10'
52 BARREN SOLUTION PUMP - I.R.,RV5 1/2,360 G.P.M.
53 SUMP PUMPS - 2" S.R.L. & 2" B.W.-G.M.
54 DISC FILTER - 6 FT. DIAM.-5 DISCS
55 2-IN. WILFLEY PUMP
56 9-IN. KROGH PUMP
57 1 1/2"x2" PUMP - DOR
58 PROPORTIONER

CENTRAL PATRICIA GOLD MINES LTD

DIAGRAMMATIC FLOW SHEET

CRUSHING AND CYANIDE PLANT

DRAWN BY - T.T. TIGERT

MAY 1943

hardened. These outlast the standard steel shafts, which had a life of approximately six months, by 30 per cent.

Treatment General

As has been noted from the ore characteristics, the fine native gold, once liberated, would be readily recovered by cyanidation providing solution fouling due to the cyanicides (sulphides and ferrous iron) could be controlled and good dissolution conditions maintained. Thus fine grinding and an all slime straight cyanide process has been evolved with fouling and dissolution conditions being controlled by:

(1) Short agitation at high dilution,

(2) The use of litharge in conjunction with low alkalinity,

(3) Barren bleeding,

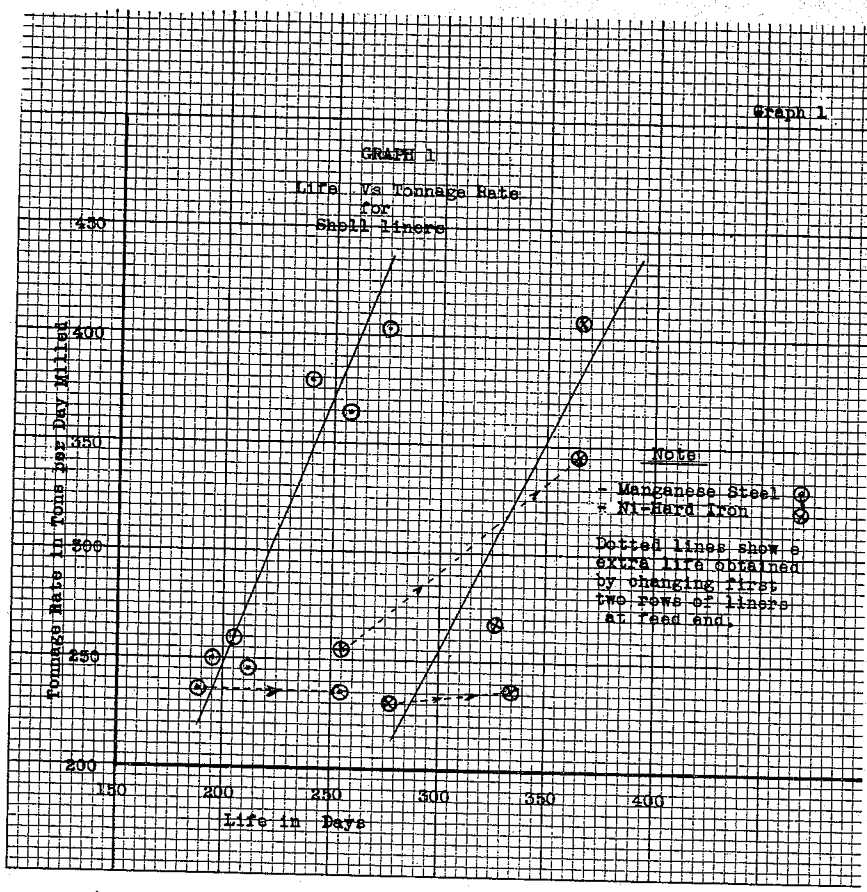
(4) The use of low lime and cyanide strengths,

(5) Temperature regulation,

(6) Maintaining a precipitation rate of over 3 to 1.

Present practice is as follows:

(a) Grinding in low alkaline cyanide solutions (0.1 lb. CaO and 0.45 lb. KCN per ton) agitating at classifier overflow dilution (15 per cent solids) for eight hours with a solution strength of 0.12 lb. CaO and 0.8 KCN per ton; (b) Single stage thickening followed by double stage filtration using barren wash and precipitating by the Merrill Crowe process.



Graph 1.

Grinding and Classification

The grinding circuit which is a single stage closed circuit unit has a capacity of 425 tons when reducing minus 3/4-inch feed to 70 per cent minus 325 mesh. It consists of a 7-foot diameter x 11-foot Allis Chalmers Overflow

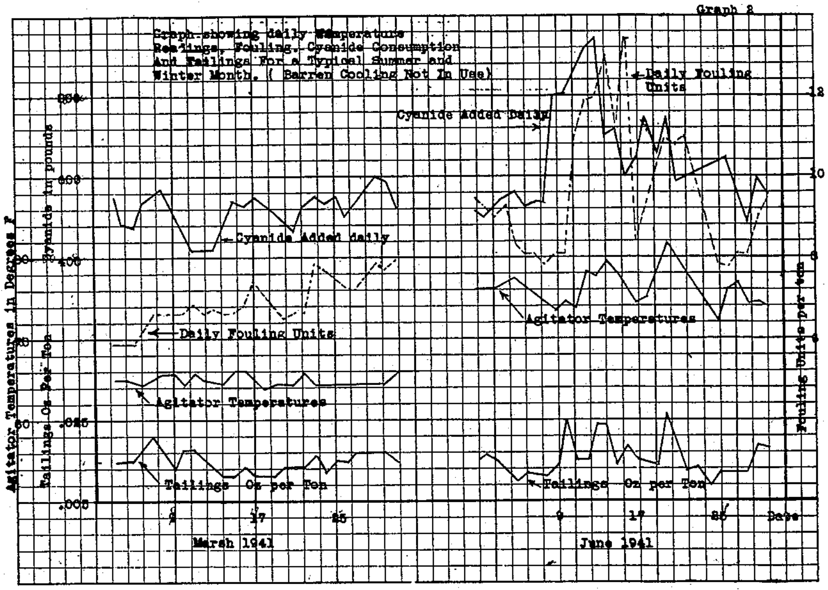
Table 2.—GRINDING CIRCUIT SCREEN ANALYSIS OCTOBER 2ND, 1949

MESH	FEED		B.M. DISCH		CLASSIFIER SANDS		C.O.	
	% ON	% CUM	% ON	% CUM	% ON	% CUM	% ON	% CUM
1.05	.1	.1						
.742	10.8	10.9						
.525	22.3	33.2	.1	.1				
.371	13.5	46.7	.48	.58	.54	.54		
3	10.5	57.2	.65	1.23	.51	1.05		
4	8.4	65.6	.90	2.13	.62	1.67		
6	5.4	71.0	1.17	3.30	1.33	3.00		
8	4.9	75.9	1.4	4.70	1.65	4.65		
10	3.2	79.1	1.4	6.10	1.76	6.41		
14	2.7	81.8	2.17	8.27	2.68	9.09		
20	2.0	83.8	2.9	11.17	3.51	12.60		
28	1.6	85.4	4.45	15.62	6.44	19.04		
35	1.3	86.7	5.65	21.27	8.58	27.62		
48	1.0	87.7	6.6	27.87	9.00	36.62		
65	.83	88.53	7.9	35.77	10.7	47.3		
100	.79	89.32	12.5	48.27	16.4	63.7	.2	.2
150	.68	90.00	12.3	60.57	14.1	77.8	.9	1.1
200	.74	90.74	9.5	71.07	8.8	86.6	4.9	6.0
325	1.5	92.24	7.8	78.87	5.1	91.7	9.2	15.2
							14.4	29.6
—325		7.76		21.13		8.3		70.4*
—200		9.26		28.93		1.34		84.8*
	96% Solids		73.4 %Solids		84% Solids		%Solids 16.2	

New Feed Tons/24 Hours—444
Tons of Return Sands/24 Hours—1573
Amps 296

Kilowats 167 H.P.—224.2
Class Rake Speed 14.75 Strokes/Min.
Efficiency Calculated 87% (100 Mesh)

HP Hours/Ton—12.12
Ratio: Sands to Feed 3.54
* Obtained By Difference



Graph 2.

Type Ball Mill driven by a 300-h.p., 550-volt slip ring motor; an 8-foot x 25-foot Dorr DSF type classifier set at a slope of 2-1/2-inches to 1 foot with a rakespeed of 14.75 spm and driven by a 7-1/2-h.p. gear motor; and a Hardinge Type C constant weight feeder.

The ball mill is operated at a discharge density of 65 to 70 per cent solids. The discharged pulp flows by gravity to the classifier which is maintained at 15 per cent solids using mill solution for dilution. The overflow is pumped by a 5 x 5 SRL pump to the first agitator, while the sands fall by gravity to the scoop box where they are returned to the mill by the 60-inch rubber lined combination scoop and drum feeder fitted with a Ni-hard scoop tip.

The ball charge is held at trunion level, 45 per cent of mill volume. It was found that the desired grind could not be maintained in spite of a constant density on the classifier overflow when the proportion of fine steel in the mill became too low and the capacity of the mill for 3/4-inch feed was seriously reduced when it became too high. Since it was also found that the addition of one-size balls could not provide this proper equilibrium charge, ball rationing was adopted. Present steel consumption is 2.24 lb. per ton of forged steel balls with daily ration being made up as follows — 25 per cent 5-inches, 30 per cent 3-inches and 45 per cent 1-13/16-inches. This ration which varies slightly with changes in feed character, liner wear, etc., is controlled through the screen analysis of the grinding circuit.

TABLE 3.—REAGENT CONSUMPTION

Reagent	Consumption in Lb. per Ton
NaCN	0.620
Lime	1.93
Litharge	0.662
Zinc Dust	0.0951
Filtercel	0.100

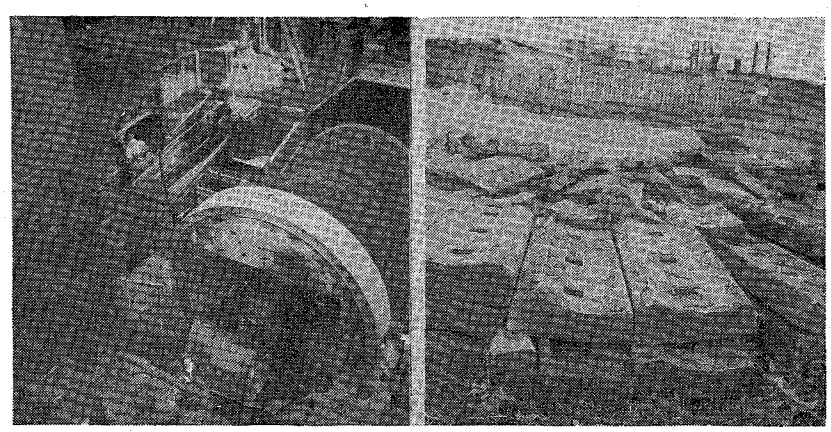
TABLE 4.—CONSUMPTION OF FORGED STEEL BALLS

Ball Diameter In Inches	Consumption in Lb. Per Ton
Five	0.597
Three	0.667
One and Thirteen Sixtenths	0.974
Total	2.238

Tonnage is calculated from the revolutions and the average weights obtained by taking hourly ore cuts on the ball mill feed conveyor. The length of the cut is equal to one-quarter the circumference of the rear pulley. The feed conveyor motor and a warning horn are interlocked with the mercuroid switch on the Hardinge feeder so that if the feeder runs empty the horn sounds and the conveyor stops. Thus revolutions are only recorded when ore is actually being fed to the mill. Another labor saving item is that the power for both the Hardinge feeder and the feed conveyor is taken from the ball mill breaker. When the ball mill is shut down or kicks out both motors stop and cannot be restarted thus preventing any serious spills.

"Ni-hard" liners which give about 30 per cent more life than manganese steel (Graph 1) are used throughout the mill. The shells are the offset wave type while the ends and throats are ribbed. The offset wave type liners allow full tonnage to be reached more quickly after a reline, and due to the bolt holes being in the thickest part of the liner when wear is completed the cheaper square head standard machine bolt can be used.

Due to the shell liners wearing more rapidly at the feed end it is becoming the practice, when worn, to replace the first two rows at the feed end with partially worn ones thus obtaining up to two months extra life from a set. Another economy practiced is the hard surfacing of the classifier rake blades along the wearing edge with Colmony No. 1 hard surfac-



Left: Grinding Section, Central Patricia Mill. Right: Off-set wave liners which are used in standard practice at Central Patricia.

ing rod which doubles their life.

Metallurgy Control and Reagents

Litharge PbO — It has been found that litharge at .5 to .75 lb. per ton ore when fed continuously to the ball mill, and used in conjunction with a low lime alkalinity reduces reagent consumption, and fouling, and accelerates the dissolution rate of gold. The litharge has not proved fully beneficial when batch fed, added after the grinding circuit, or used in conjunction with a high lime circuit. No marked benefits were found if used in excess of .8 lb. per ton.

Lime and Alkalinity. — Lime is used in the Mill to provide alkalinity, and all alkalinity referred to is that as determined with sulphuric acid using phenolphthaline as the indicator, and reported in lb. per ton CaO.

While this ore protective alkalinity must be maintained, and it must be low. If the alkalinity drops too low reagent consumption is prohibitive, and the tailings rise alarmingly while, if the alkalinity is too high maximum extraction is not obtained due to high lime. In his paper "Cyaniding a Pyrrhotitic Ore at Central Patricia," *Canadian Mining Journal* for August. 1942, N. Gritzuk dealt extensively on the effects of litharge and alkalinity as applied to this ore. Plant operation has shown that the ideal alkalinity is 0.1 lb. per ton CaO with the safe range lying between 0.05 and 0.25, and, if these limits are exceeded immediate reaction in increased tailings, reagent consumption and fouling is noticed.

Since dry lime could not be fed due to the close control necessary, a milk of lime system was installed consisting of a small ball mill, cone classifier and agitator with the lime being distributed from movable cutters.

Just sufficient lime is added to the ball mill to keep the alkalinity of the discharge between .05 and 0.1 lb. the balance necessary to bring the first agitator up to .12 lb. per ton being added to the classifier overflow. Usually no further additions of lime are required to keep the alkalinity above .05 by the end of agitation, but when needed dry lime in five or ten pound lots is added to the other agitators with titrations being made fifteen minutes after each addition.

Cyanide. — All cyanide (Cassels brand sodium cyanide) is added to the classifier overflow in five or ten pound lots bringing the solution up to strength for agitation. This practice was adopted to avoid high cyanide strength in the ball mill which only leads to increased consumption and fouling.

Agitation. — Due to the extreme rate of oxidation of the pyrrhotite a very short agitation time has been adopted to prevent prohibative reagent consumption and solution fouling. All agitation is carried out at classifier overflow dilution, thus omitting an extra thickening stage where the pulp would have a further chance to oxidize, as well as permitting low lime and cyanide strengths to be carried and yet provide an ample reserve of both to meet any sudden requirement.

Due to low lime, the fineness of the grind, and the agitator speed (7-1/2 RPM for the 26 ft. one and 6 RPM for the others) no difficulty is encountered with the agitators settling out as might be expected at this dilution.

Temperature. — As may be seen from Graph 2, showing temperature reagent consumption and tailings for a typical summer and winter month, when the pulp reaches a temperature around 75 deg. F., which seems to be critical for this ore, the circuit becomes very touchy, giving erratic reagent consumption and fouling and tailings. To offset these conditions the following means of temperature control are used:

1. Maintaining a high dilution in the ball mill, and taking in of all make up water (Mine water is used in the summer) into the ball mill where very high impact temperatures are met.

2. The cooling of the barren solution before re-use. The details of this barren cooling system which is capable of reducing the temperature of the barren solution by six deg. is as follows:

Barren solution from the presses flows to the barren tank from which part of the barren is pumped to the cooling unit which is a fire tube boiler rigged up so that the barren solution may be circulated through one side of the coils while mine water flows through the other. The cooled barren returns to the barren tank where it is distributed to the mill solution tank, filters etc.

Following temperature readings are typical of this system and are shown in degrees F: —

Barren from presses	74
Barren to cooling tank	72
Barren return	68
Barren tank	72
Mine water to cooler	48
Mine water return	54

Barren Bleeding. — In spite of the successful application of metallurgical control approximately 0.3 tons of barren solution per ton of ore must be continuously removed from the circuit to prevent the accumulation of fouling compounds. This is accomplished by the use of a barren wash on the last filter and supplemented by bleeding a measured amount through a rectangular weir. Plant results have shown that whenever reducing power of the circuit rises above 800 cc 0.1 N $KMnO_4$/litre the rate of barren bleeding must be increased until the circuit returns to normal.

Precipitation. — The rejuvenation of solution through precipitation is another important factor. It has been found that when the precipitate ratio falls below three to one extraction drops off and plant practice is to keep slightly above this limit.

It is to be noted that, although the individual function of each reagent and control is to reduce reagent consumption and improve extraction, it is only through their close control and combined effects that maximum economical extraction is obtained. The neglect of any one will finally destroy the beneficial effects of all the others.

Mill Laboratory. — Practically all test work resulting in the present plant practice was carried out in the mill laboratory. This laboratory which is fairly well equipped has an Abbe mill, a Ro-Tap screen shaker, a 6-bottle agitator, a 1000 gm. unit flotation cell, and a Beckman industrial model Ph meter as well as all apparatus and chemicals for carrying out wet analysis.

Thickening and Filtration. — No difficulty is encountered in thickening, but due to the low lime and their size, the four 20-foot thickeners are used in parallel. These are driven at 0.24 rpm by a 1-h.p. gear motor.

The thickener overflow which flows direct to the clarifier tanks is fairly clear in spite of the low lime, although it becomes slightly more turbid during winter necessitating the washing of extra clarifier leaves to maintain the required precipitation ratio. The thickener underflow which is removed by Dorrco 3-inch simplex diaphragm pumps flows direct to the primary filter hoppers.

The primary filter is a five disc 6-foot diameter American-type filter while the secondary is an 8-foot diameter x 10-inch FEINC type filter as made by Northern Foundry and Machine Company. Barren wash is used on both filters, the filtrate from the former going to the clarifier tank, and from the latter, to the mill solution tank. The solids from the primary filter are repulped with barren and pumped to the secondary whose discharge is repulped with water to 40 per cent solids, sampled with an automatic sampler, and sent to tailings disposal.

Both filters have a filtering area of 250 square feet and operate under a vacuum of 23 inches provided by an Ingersol Rand 22-inch x 9-inch Type ES-1 vacuum pump. The filters handle 1.8 tons of solids per sq. ft. per 24 hours with average soluble loss running 0.001 oz./ton milled.

No. 10 twill cotton sector bags which have an average life of 5 months are used on the American filter. Each sector is washed and acid treated every ten days. Scythes No. 400 cotton sheeting which gives a life of 120 days is used on the Feinc filter. This cover is scrubbed daily and acid treated every five days.

Precipitation. — Precipitation is accomplished in a Merrill Crowe system using bag type presses. There are four sixty-bag presses, two of which are in service while the others are being readied for the next change. The bags consist of an inner cotton sock of No. 86 sheeting which retain the precipitate and an outer burlap one made from 10-1/2 oz. extra quality jute. The former are used only once while the latter, which have a life of two years, are reused until they become damaged or contaminated with precipitate.

Press cleanup is carried out twice a month with the new presses being primed with 30 lb. filtercel and 8 lb. of zinc with a slow drip of lead acetate being added to the cone for the first eight hours. The precipitate in the full presses is washed with water (it was found that the washing helped prevent a rash which the operators were developing from handling the precipitate as well as improving refining) and blown with low pressure air for 24 hours. The presses are then dismantled; the precipitate separated from the socks by emptying into tubs and, sent to the refinery together with the inner socks which are burnt and whose ashes are added to the precipitate.

Zinc dust is fed at .03 lb. per ton solution, and, as long as the fouling stays down and clarification is complete, trace barrens and a fairly high grade precipitate are obtained.

Clarification is carried out using 24, 6-inch x 4-inch leaves fitted with two cotton covers of No. 86 sheeting. It was found that the double covers, which scarcely affected the capacity of the leaves, gave better clarification and prevented slimes from entering the press with its usual disastrous results should the outer cover become holed.

Six to 8 leaves are washed daily with each being acid treated once per month and all precoated with filtercel before they are returned to the tank. The average life of a cover is six months.

Refining. — The precipitate is trucked to the refinery, a galvanised steel building with cement floor where it is fluxed and melted.

The melting is done in an oil fired Rockwell type furnace, the oil being supplied from a steady head tank located outside and the air from a receiver which is held at 40 PSI pressure. The furnace is lined with "carbofrax." 1000 lb. is used for each lining giving an average life of 120 fusion hours.

A high nitre flux is used with fusion time running 1-1/2 hours per 100 lb. of precipitate. The precipitate, which contains 20 to 30 per cent moisture with a dry analysis averaging 31 to 40 per cent gold, 6 to 9 per cent sulphur, 2 to 5 per cent lead and 22 to 30 per cent zinc is fluxed as follows:

Ppte.	100 lb.
Niter	30
Borax	33
Silica	9
Soda Ash	12
Fluorspar	2

Usually 200 to 300 lb. of precipitate is fluxed at one time with 125 to 150 lb. of the mix being added to the furnace and fused until the charge has fumed strongly for 5 to 10 minutes and no beads of metal are seen on the back of the furnace as it is rotated. The furnace is then slagged off and another charge added. When sufficient metal has accumulated in the furnace it is slagged off and the gold poured direct into the hot bar mold care being taken not to run it over. This eliminated the button stage with subsequent fluxing, remelting and pouring.

The bars are then cleaned, sampled by drilling, weighed, packed, and shipped to the Mint. Average bullion fineness is 825 gold and 67 silver.

Mill Costs. — The average direct milling costs for the first six months of 1949 in terms of dollars per ton ore milled are:—

Supervision	.03
Crushing and Conveying	.10
Grinding and Classification	.32
*Agitation and Thickening	.37
Filtration	.05
Clarification & Precipitation	.07
Tailings Disposal	—
Motor Maintainence	.01
Oils and Greases	.01
Compressors	.02
Lighting	—
Assaying	.03
Refining	.02
Fire Insurance	.03
Miscellaneous	.03
Cartage	.01
Total	1.10

*Including Reagents NaCN, PbO and CaO

Labour. — Mill is operated by a 12 man crew consisting of 3 solution men, 3 ball mill men, 2 crushermen, a filterman, a helper, a mill mechanic and a superintendent.

Central Patricia Assay Office

by

Ernest Warren

THE ASSAY office is a frame building 24 feet by 52 feet with galvanized sheet roofing and asbestos siding. It is lined with gyproc sheeting and all the flooring is concrete. It contains crusher room, furnace room, wet room, flux and store room, balance room and office. (see plans). The building is heated by steam, being connected with the main heating system of the mine.

The ventilation system is a very efficient unit in itself. The fan is in the crusher room installed eight feet above the floor. It is a Sheldon mill exhauster, type XA, size 13, and driven by a 3-hp motor at 1,150 rpm. The intake capacity is 2,450 c.f.m. The intake pipe runs along the ceiling of the crusher room, furnace and flux room. Feeder pipes with damper control system are fed into the main intake from the Number 1 crusher, bucket elevator, Number 2 crusher, pulverizer, pulp table, furnace and flux room. Konimeter samples are taken at various operations in the crusher room and the average count was 70 p.p.c.c., same indicating that the air is relatively clean.

Description of Crusher Room

The crusher room is 11-1/2-feet x 24-feet containing sample dryer, two crushers, bucket elevator, one automatic riffle, one hand riffle, pulverizer and a filing system for pulp rejects. The dryer is made of asbestos sheet flextos, being 78 inches wide, 72 inches high and 26 inches in depth. There are five shelves, each shelf is made up of 1-1/2-inch pipe and 1-1/2-inches apart directly connected to the steam line. The dryer has three doors, (figure 1), and having the capacity of drying 90 samples at one time.

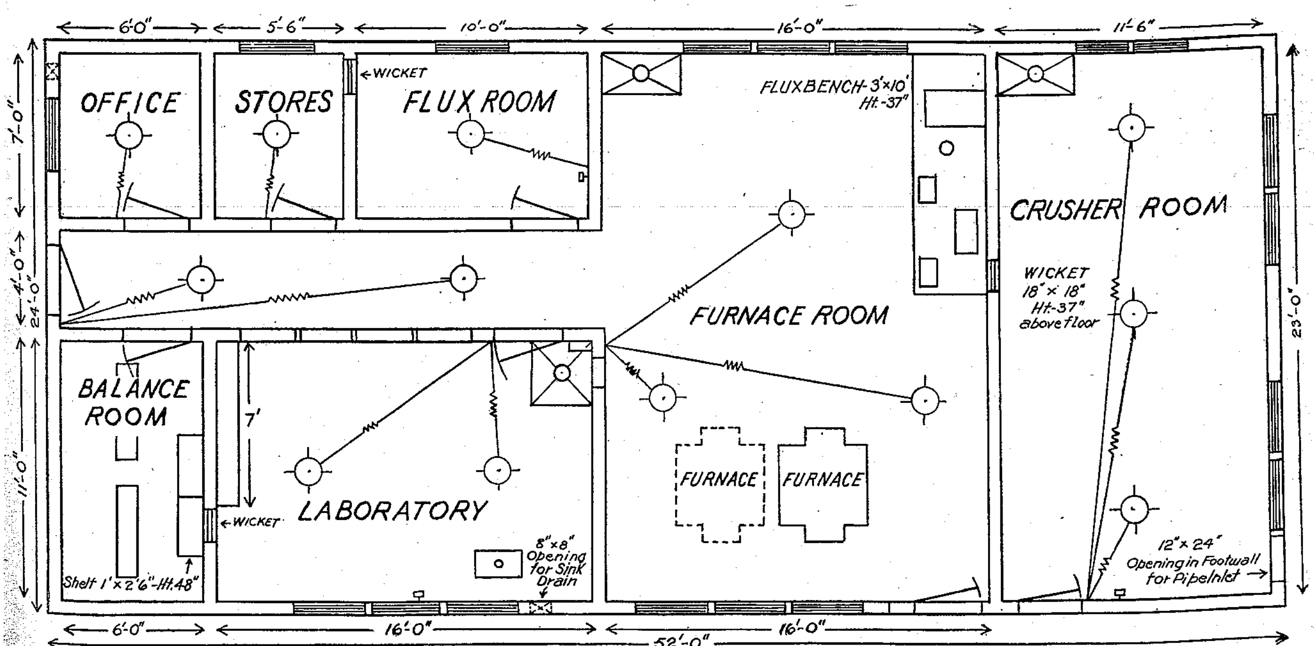
The number one crushing unit is a Mitchell No. 1 crusher with 5-1/2-inch x 8-inch jaw opening. This crusher is totally covered with enclosures of sheet metal as shown on the left hand side of figure 3. This enclosed unit contains a riffle below the crusher with a chute directly connected to the bucket elevator; thus all crusher rejects are mechanically handled. A pipe with a damper leads from this unit into the main pipe of the ventilation exhauster.

The number two crushing unit is a Mitchell No. 0 crusher with 4-inch x 5-1/2-inch jaw opening. This unit is also totally covered with enclosures of sheet metal as shown on the right hand side of figure three. A pipe with damper leads to the main ventilation pipe. This unit is connected to the high pressure air line and worked by a valve for cleaning out the crusher after each sample.

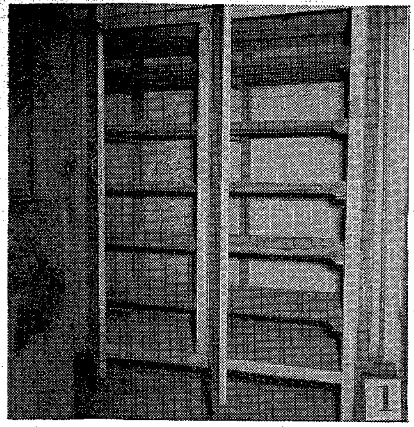
The hand riffle which is located between the two crushers is used specifically for diamond drill samples. A chute leads from this riffle to the bucket elevator.

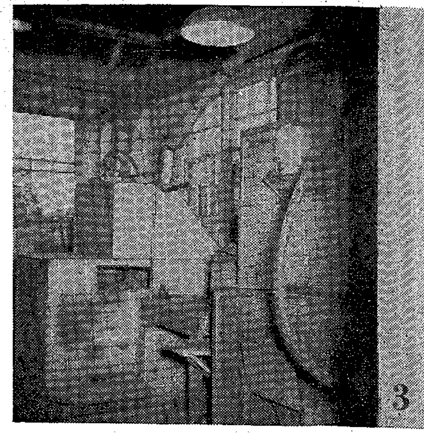
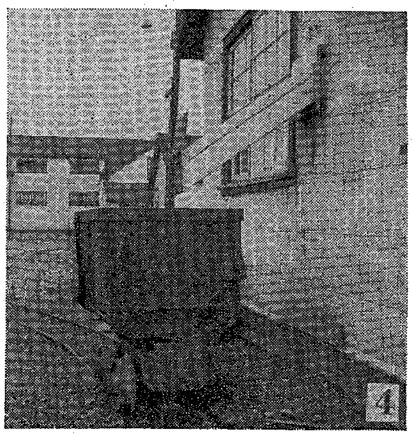
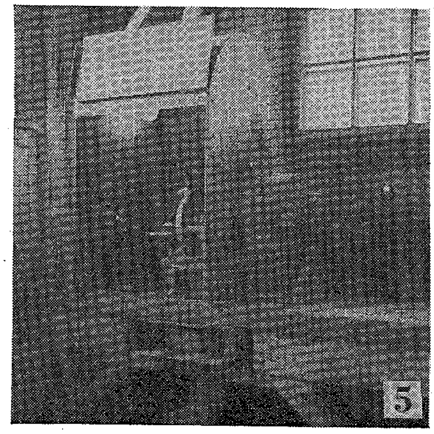
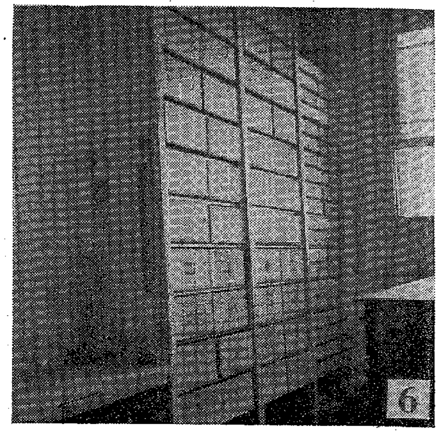
The bucket elevator saves con-

*Chief Assayer.



Plan of Assay Office.



Central Patricia Assay Office. Figures 1 to 6.

siderable time and also does away with having the crusher room cluttered with crusher rejects. The elevator takes the rejects from the Number 1 crusher unit and the hand riffle. The elevator is 8-1/2 feet high, containing 24, 3-inch by 4-inch buckets, which are 6-1/2 inches apart. At the top of the elevator, a chute leads outside into a muck car as shown in figure 4.

The two crushers and the bucket elevator are connected to one main shaft line driven by a 7-1/2 h.p. motor at 1,160 rpm. The various units have different sizes of pulleys according to the ratio of speed required.

The pulverizer is a Braun type UA enclosed as shown in figure 5. There are two ventilation pipes with dampers, leading from this unit to the main ventilation pipe. Two high pressure air lines are connected to this unit, one at the opening and one into the stationary plate for cleaning purposes after each sample is pulverized. This unit is driven by a 3-hp motor at 1,150 rpm.

The pulp reject filing system as shown in figure 6 contains 31 boxes numbered from 1 to 31, representing each day of the month. The rejects are held for one month only, then discarded.

Furnace Room

The furnace room is 16-feet x 24-feet in plan containing furnace, pulp table, tray table, cupel shelves, anvil and the furnace table.

The furnace is a Wayne assay furnace having a capacity of 32, 20-gram pots, (figure 7). It is heated by four Globar Carbofrax elements. Life of same is from two to eight months depending on the number of hours the furnace is used per day. It is automatically controlled by a pyrometer. A ventilation pipe with damper control leads from the furnace to the main ventilation pipe.

The pulp table as shown in figure 8, has a glass canopy with a sloping glass front coming within 14 inches of the top of the table. In the middle of this table against the wall, there is a ventilation duct with 3-1/2-inch x 72-inch opening, this being connected to the main pipe, to draw out any dust entailed in the process of rolling the pulp samples. A Voland and Sons pulp balance is used. This is placed three inches above the table so as not to obstruct the ventilation duct.

The tray able with the pots in each tray is a time saving factor in the furnace room. The top of the table is covered with asbestos flextos sheeting. Each tray consists of 32 pots and also specifically used for certain samples, namely "mill", "underground," "underground diamond drill" and "surface diamond drill". (fig. 2).

Cupel shelves as shown in figure 7 consist of 46 trays, each holding 210 cupels or a total of 9,660.

Wet Room

This room is completely equipped to run solution assays and all base metal assays. It also contains an annealing furnace, hot plate and distilling apparatus, as shown in figure 9.

The aluminum plate method is used for solution assays. This will be described in the latter part of this article.

The hot plate as shown on the left hand side of figure 9 is 18-

inches x 30-inches using two top elements of a Moffat range rated at 3,000 watts connected to a 220-volt line, having a switch with a low, medium and high selector. It is all enclosed with asbestos flextos sheeting having two sliding doors with glass window, one facing the furnace room and one the wet room. A venturi connected to the high pressure air line is used for ventilating the hot plate enclosure.

The annealing furnace as shown in figure 10 was made in our electrical shops. The outside dimensions are 8-inches x 14-inches and 18 inches in depth enclosed with 1/8-inch aluminum plate. Inside dimension are 11 inches x 4 inches x 15 inches in depth enclosed with 1/2-inch asbestos flextos. It is packed with asbestos fibre between the inside and outside. It is heated by a suspended Acme oven grill with a capacity of 1,500 watts connected to a 220-volt line.

The still is a Barnstead still with automatic low water shut off unit. Capacity is 1 gallon per hour.

Flux Room

Flux is made in a 15-gallon barrel dairy churn. This barrel being in a partially enclosed box like affair and is connected to the main ventilation pipe.

Flux consists of the following reagents:—

Litharge	50 gms.
Soda Ash	26 gms.
Silica	6.5 gms.
Borax	7 gms.

made up in batches of 89-1/2 lb. at a time and stored in cans.

A Denver fire clay cupel machine is used, making 1-1/4-inch cupels. The cupel mixture consists of 50-50 cement and bone ash by volume.

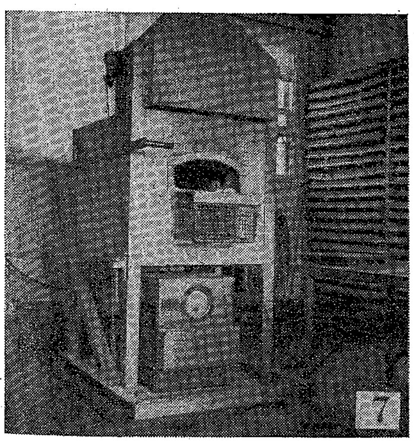
Balance Room

There are three balances as shown in figure 11, Heusser 1/300 used for bullion only, a Roller Smith used for daily assays and a Central Scientific Analytical balance used for base metal assaying. The three balances rest on a concrete table 24-inches x 130-inches and 36-inches above the floor. The Roller Smith balance is a time saving factor in this assay office. Up to one hundred samples can be weighed in one hour, with extreme accuracy.

Crusher Room Procedure

The usual daily run is between 105 to 175 samples, consisting of muck, channel, mill and diamond drill samples. The mill samples consist of six mill heads, seven pulp samples and ten solution samples. The pulp samples are not pulverized, but put through a 40-mesh screen to break up all lumps.

The mill heads, channels and muck samples are all sorted to numerical order and crushed in the Number 1 crusher. After each sample is crushed, the crusher is thoroughly blown out with a high pressure air hose. As this crusher has a riffle in the bottom of it, the samples are put through till the amount that is required is left in the pan. From there it is put through the number 2 crusher and then put in the dryer if the sample requires to be dried. The number 2 crusher is thoroughly cleaned out with air after each sample. All diamond drill samples are crushed in the number 2 crush-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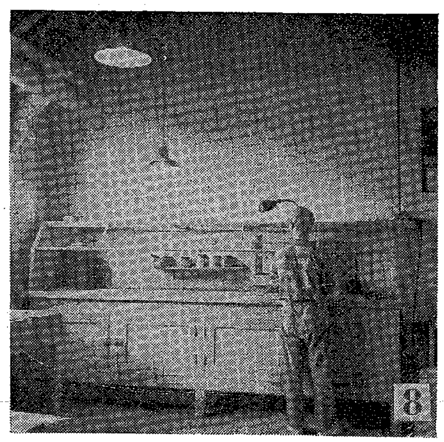
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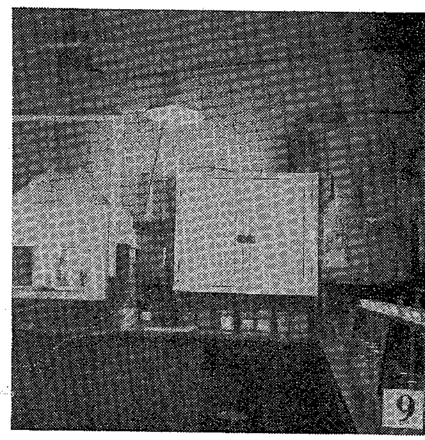
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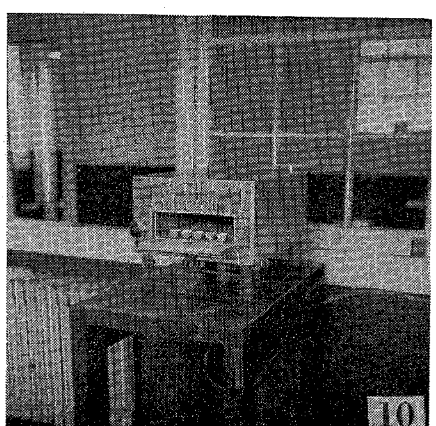
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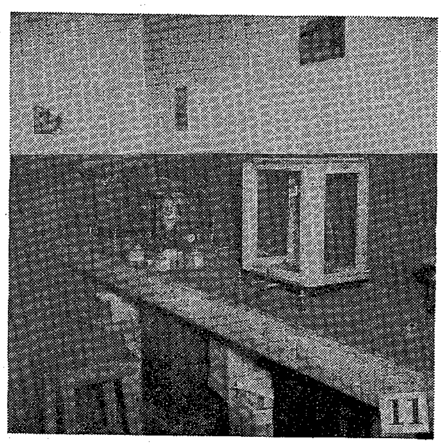
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Central Patricia Assay Office. Figures 7 to 11.

er only, and riffled to size by the hand riffle.

All samples are pulverized in numerical order. After each sample is pulverized, the pulverizer is thoroughly cleaned and blown out with air. All samples are pulverized between 93 per cent and 95 per cent minus 200. As will be noted in the plan, there is no inside door between the crusher room and the furnace room. All samples are handed through a sliding window 18-inches x 18-inches directly to the pulp table.

Furnace Room Procedure

The mill tray is the first group of samples to be weighted out. All mill samples are weighed out as follows: 6 mill heads at one A. T. each; classifier o'flow No. 1 agitator, No. 3 agitator, one A.T. each; No. 9 agitator at two A.T. in a 30 gram pot, and the three tails are weighed out using four 30 gram pots to each tail sample, each pot having one A. T. All other samples, viz. muck, channels and diamond drill, are weighed out 1/2 A.T. each. The samples are rolled forty-two times before weighing. Since the mill samples are high in sulphides, no reducing reagent is added. Flour or nitre, according to the amount of sulphides present, is added to the other mine samples. Each crucible which contains the flux and sample are then rolled 18 times. All mill samples are given a light cover of PbO due to the sulphides in the ore. Silver is added to all the samples.

The tray of 32 crucibles is then put on a swivel table on the left hand side of the furnace. Using a multiple crucible tong, four cricibles at a time, it only takes 30 seconds to put 32 crucibles into the furnace. The furnace heat for fusion purposes is at 1950 deg. F. and after the charge is in, the temperature goes down to 1700 deg. F. When the temperature is back to 1950 deg. F., taking about 35 to 40 minutes, a distinct click is heard when the furnace cuts off, thus the fusion is finished ready to pour into molds. After they are poured, another charge of 32 crucibles is put into the furnace, as it was found that it was most economical to cupel 64 samples at a time. The buttons are squared, being an average of 20 grams each, and put on a button tray, which is eight to a row and 9 rows deep. The cupels are then put on a tray made of heavy sheet metal 10-inches x 20-inches with 1-inch sides on the longitudinal part, the cupels being eight to a row and eight deep. This tray has a handle 24 inches long also having a heavy metal push type affair that fits within the walls of the tray for pushing off the cupels in the furnace. After the second fusion is finished and poured, the cupels are put into the furnace which takes about 5 seconds.

By this time the buttons of the second fusion are squared, the furnace is at cupellation temperature, which is started at 1950 deg. F.

After the cupels are in the furnace for 5 minutes, the furnace door is opened wide and the ventilation pipe damper is opened, and this is connected to the main ventilation pipe. It takes about 20 to 25 minutes for cupellation to finish. During this process, the furnace temperature is stationary at about 1750 deg. F. The cupels are then taken out, the beads cleaned, flattened and put into Coors No. 000 parting cups.

All pulp rejects are put into 1-lb. paper bags, each bag being numbered corresponding with the sample number and stored for one month.

The slag and used cupels are taken to the mill.

Wet Room Procedure

There are nine mill solution samples consisting of three barrens, three tails, one mill, one classifier o'flow, and one pregnant. The barrens are run individually 20 A.T. in a 800 ml. beaker. The tailing solutions, two samples of 30 A.T., are measured into 1000 ml. beakers. Mill and classifier o'flow are 10 A.T. and the pregnant solution is run in duplicate at 10 A.T. each. The aluminum plate method is used in this process. 25 cc of lead acetate solution is added to each beaker. The lead acetate is made up of 750 grams of lead acetate dissolved in 1000 ml of water plus 10 ml of acetic acid. A few drops of silver nitrate solution is added to each beaker. An aluminum plate 1-3/4-inches x 3-inches x 1/8-inch is inserted into each beaker and the solution samples added. The beakers are then put on a hot plate, adding 20 ml of HCl to each barren, 25 ml to each tail and 15 ml to the mill, classifier o'flow and pregnants. They are boiled vigorously until the solution is clear, which takes about 20 to 25 minutes, then taken off the hot plate, filled with cold water and the aluminum plates taken out by tongs, making sure that no sponge is adhering to the plates. The solution is then decanted into a 15-20 cm. evaporating dish, decanted, then filled with clean water transferring the lead sponge into it. The sponge is squeezed into a ball, dried on a hot plate and cupelled.

The parting cups are handled in an aluminum tray 5-1/2-inches x 8-1/4-inches x 3/4-inch in depth, holding exactly 24 cups. The parting solution is 5:1 nitric acid. The samples are parted on the hot plate, acid decanted, washed twice, dried and annealed in the annealing furnace, taking 5 minutes to anneal. They are then cooled and ready to weigh.

As the mining company pursues an aggressive exploration program, examining many outside properties, the assay office is prepared and equipped to assay samples for gold, silver, copper, lead and nickel.

Balance Room Procedure

The parting trays are all numbered and beads are weighed in numerical order on the Roller Smith Balance. After all the samples are weighed, the various reports are made out and distributed to the proper departments in the mine. As a rule, 100 to 175 samples are assayed daily and the reports are all finished by 2:30 p.m.

Bullion Assaying

Bullion is assayed twice a month, on the 3rd and 18th. There are two bars for each period. Between 200.00 and 200.15 mg. of bullion are weighed on the 1/300 Heusser balance and wrapped in a lead foil 3-1/2 inches x 6 inches with enough silver to make the ratio of gold to silver 1 to 3. It is then cupelled in Morganite cupels at 1950 deg. F. Experiments have proven that cupelling at a high temperature gives the assaying a greater degree of accuracy. After cupelling, the beads are cleaned and rolled in an Oliver Jeweller's Rolls, then parted in a 50 ml. bullion parting flasks using 1 to 5 nitric acid. To each parting flask is added a small piece of battery carbon to avoid any spitting or boiling over. They are boiled vigorously for 20 minutes, decanted and 1-1 nitric acid is added and boiled again for another 20 minutes. They are then washed three time, decanted into a No. 000 Coors parting cups, dried and annealed. After cooling they are weighed on the Heusser balance and calculations made for gold and silver fineness. There are two samples for gold fineness and two for silver fineness assayed from each gold bar.

Costs

The location of this mine, and the high cost of transportaion affect the costs of assaying, here given for the past three years:

Year	Cost Per Assay
1946	$0.3293
1947	$0.3167
1948	$0.338

Reference

Assay Office Dust Control at Wright-Hargreaves Mines Limited. — C.I.M.M. Transactions. Vol. XLIV, 1941, pp. 423-434.

Safety at Central Patricia

During the current year a plant wide safety program has been in force, and although a perfect safety record has not been attained the improvement over previous years has been very gratifying and proves beyond all doubt that if enough effort is extended to educate the supervisors and workmen in safety, the average person will become safety concious to a point where dividends are paid off in terms of a good safety record, increased efficiency, good labour relations and less lost production due to accidents.

Being members of the National Safety Council, a selection of safety posters and literature is received periodically. The posters are displayed on special notice boards which are moved around to the plant departments on a regular route. Posters are on display no longer than four days in each department.

Each department has on display, in a prominent location, an Accident Record board as shown in the photo, and it is the duty of each department head to keep his board up to date on a daily basis. Beside the above accident indicator the mining department has on display a large 'novel' accident record board, which is 60 feet long by 12 feet wide, as shown.

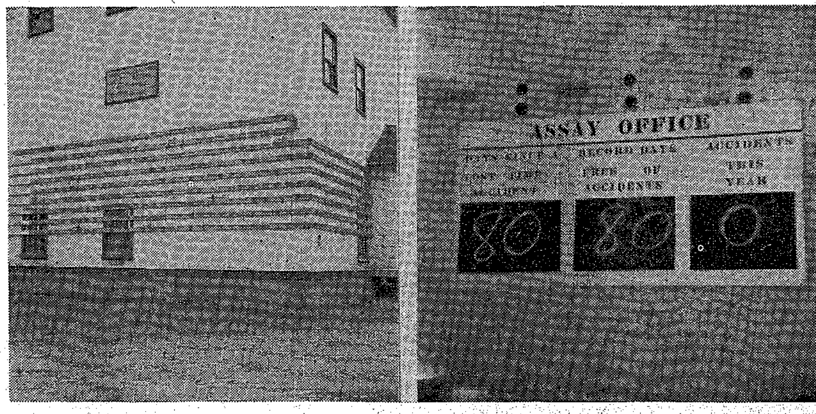
Other methods of promoting safety by visual means is by articles published in the employee's newspaper, safety rallies held in the Community Hall at which films on safety are shown, and a Green Cross safety flag flown in plain view of the plant and townsite. Liberal use is made of fluorescent "Scotchlite" tape to indicate accident hazards underground.

A safety committee, appointed by the management, meets once a month to discuss means for the prevention of accidents, and for the improvement of working conditions. A committee of three makes regular inspections of all departments and submits their findings to the safety committee.

The ethyl mercaptan installation is tested periodically, and emergency tool boxes and first aid supplies are kept at strategic places throughout the mine. A procedure in case of an underground fire has been drawn up and issued to employees in book form.

In common with all the mines in Ontario, a number of men are being constantly trained at the Department of Mines' Rescue Station located on the mine property.

All injuries, however slight, must be reported to the shiftboss before going off shift. He makes a record of the injury, giving all details. A hospital with the customary equipment and supplies is maintained on the property for the treatment of men who may have been hurt or have taken ill on or off the job. The services of a doctor and nurse are available at all times.



General and Departmental Safety Score Boards.

Medical Services
Pickle Lake Area

by

E. S. Connor*

MEDICAL services in the district are provided in two hospitals, one located at the property of each of the producers. Each has a full-time nurse living in, and the doctor's residence is close by. When necessary, qualified married nurses in the communities provide temporary assistance. The most harmonious relations exist between the personnel of the two institutions, which makes possible close co-operation and mutual assistance in all branches of medical practice, particularly in major surgery.

Each employee has deducted from his wages the sum of one dollar per month which entitles him to regular medical care and hospitalization. Hospital accommodation is available to employees' families at reasonable rates.

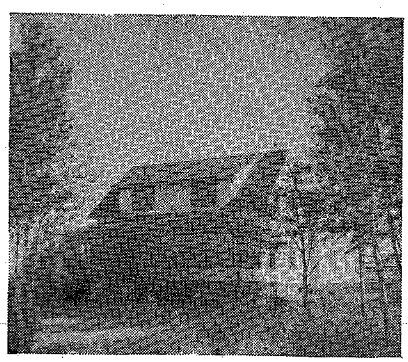
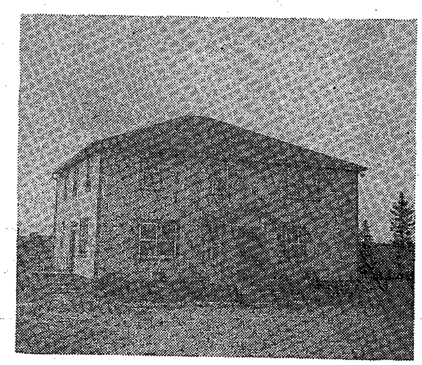
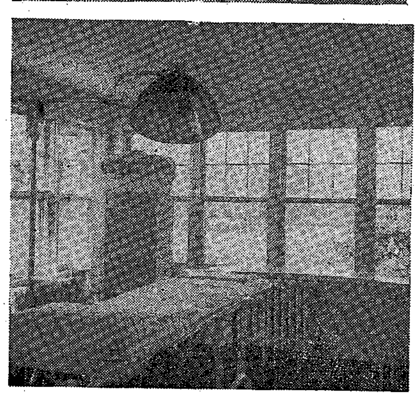
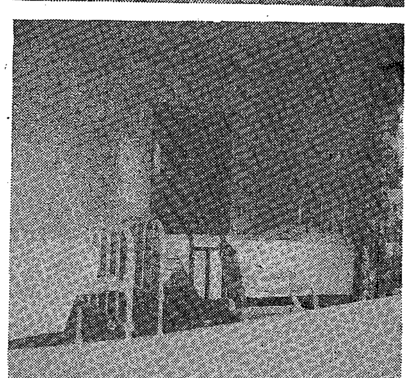
In addition to their industrial work, both physicians do a limited private practice in the mining communities and in the smaller settlements at Pickle Lake and Lake St. Joseph. Medical care is also provided the numerous Indians native to the district.

No "Blue Cross" or similar organization exists, due probably to the fact that the married employees are in the minority. The effect of this is that only a small proportion can be classified as permanent residents, and as in most young mining camps, a high proportion of labor is transient.

The earliest facilities consisted of, at Central Patricia, a first-aid kit on top of the office safe, and at Pickle Crow, of little more, located in a tent containing a stove which fought a losing battle. In 1936 a small hospital was built at Central Pat, to which a large addition was made in 1944. In 1934 the Pickle Crow dispensary was graduated to a back room in the office, in 1935 to a two-room log building, and in 1938 to the present well-equipped hospital.

In the first days of the camp, both mines were serviced by Dr. J. R. Robertson, from 1933 to 1934, then for a few weeks by Dr. C. F. Galway, now of Mackenzie Island, and from 1934 by Dr. E. S. Connor. In that year there was no road connecting the two properties and transportation between them took the form of two miles afoot and six by canoe during open water, and fives miles by winter trail. There is no record of the travels of the first two physicians, but the writer can attest to a total of over 3,000 miles covered in this manner, the record for one day being 28 miles on foot between Pickle Lake and the old Albany River property, back and forth.

To the delight of the local representative of the medical profession, a road was finished in 1936, enabling the use of a car — at which point the directors of the two companies decided that the growth of the population justified the services of a full-time physician at each property! The writer thenceforh confined his attentions to Pickle Crow, and was followed at Central Patricia by Dr. J. V. Connell who retired in 1941, due to ill-health, to be succeeded by his son, Dr. H. M. until he moved to other pastures in 1946.

The two photos on left show the Central Patricia Hospital and operating room. On the right are shown the Pickle Crow Hospital and one of its wards.

*Medical Office, Pickle Crow Gold Mines.

Since that time Dr. F. J. Squires has filled the post in a most becoming and capable manner.

At the risk of sounding pedantic, but in all sincerity, we conclude this brief account with the statement that in medicine as in mining, there are many frontiers left in the literal as well as in the allegorical sense. While medical practice in a small mining community is certainly more strenuous, and the remuneration possibly less in some respects than that enjoyed by physicians in the older and well-established centres, there are many compensations to mining industrial practice. Variety is the key-note, and self-reliance is thrust upon the isolated physician to-day just as surely as it was upon his horse-and-buggy predecessor. There are no famous specialists around the corner, general practice means just that, and preparedness for emergency is the watch-word.

Educational Facilities

by

C. H. Hunt* and F. Forster**

ONE OF the amenities that received earliest consideration as the two mines became established was that of the education of the children of the two communities, which are situated some six miles apart. In order that each community have some control over the primary education in its own area, Connell Township was divided into two school sections, S. S. No. 1 embracing the Pickle Crow townsite, and S. S. No. 2 serving Central Patricia, the latter having been established in 1939 with an enrolment of seven pupils.

The buildings of the grade schools were constructed by the mining companies immediately concerned, and are rented to the respective school boards at nominal rentals of one dollar per year. The continuation school, situated adjacent to the S. S. No. 1 School at Pickle Crow, was constructed as a joint effort of the two mining companies, and this also is rented to the Continuation School Board at a nominal figure.

*Chief Electrician, Central Patricia Gold Mines Limited.

**Chief Electrician, Pickle Crow Gold Mines Limited.

S. S. No. 1 (Pickle Crow)

The school building is of frame construction, 30 feet by 40 feet in plan area, and Grades I to VIII are taught in a single room on the main floor. The building is heated by a hot water system installed in its elevated basement, which also provides accommodation for the science laboratory of the Connell Township Continuation School situated nearby.

Twenty-eight pupils are in regular attendance at this school at the present time.

S. S. No. 2 (Central Patricia)

Serving a larger number of children of grade-school age, the school building consists of two rooms on a main floor, arranged on an "L" plan in order to take the fullest advantage of natural lighting. Its full-size basement contains the warm-air heating apparatus, woodfuel storage, and store-rooms.

The whole building is soundly constructed, pleasingly and usefully decorated for all of its purposes, and its general appointments are comparable with those of schools found in large centres of population. A large factor in the adequacy of the school is the close collaboration between the school board and the teaching staff on all construction and equipment plans.

In the two years since 1947 the enrolment at this school has grown from twenty-five to the present figure of sixty-five.

At the present time plans are being formulated for the establishment of a kindergarten for children of pre-school age. This however will not be administered by the school board, but will be a private venture, using facilities made available by the Central Patricia recreation hall.

Left: Connell Township School No. 1 and Continuation School, situated at Pickle Crow. Right: Connell Township School No. 2, at Central Patricia.

Continuation School

As referred to above, the Continuation School is situated adjacent to the S. S. No. 1 School at the Pickle Crow townsite, having been founded in 1946, and it serves the older children of both communities. Some twenty-five pupils are enrolled in the four grades IX to XII, and of these fourteen are transported from and to the Central Patricia townsite by a daily bus service.

Financing and Maintenance

As in all educational ventures, the raising of funds for construction and maintenance is a matter of continual concern. The mining companies have been most generous in providing buildings, and in helping in all other respects.

Until 1947 the revenues of the various school boards were obtained from voluntary monthly pay-roll deductions amounting to fifty cents per employee. This, together with a grant from the Ontario Department of Education, was adequate, if administered with care and caution, to maintain a reasonable standard of education. In 1947 however it became necessary to expand the educational facilities, and this expansion, together with generally rising costs, made it necessary to obtain additional revenues for operation.

The legislative grants made by the Ontario government, designed as they were to meet needs in more established industrial and agricultural communities, had already been drawn upon to the limit. The lack of any municipal corporation, and therefore the lack of any taxing authority in the community, posed a considerable problem.

However, in 1947 and immediately operative in that year permission was obtained from the Department of Municipal Affairs for the school boards to institute a scheme whereunder property holders in the school areas could be taxed, and a considerable portion of the revenue is now raised by this means. It is hardly beside the point to note that the principal property holders are the mining companies concerned, though the arrangement does to some extent widen the participation in the raising of necessary funds.

Conclusion

In all matters pertaining to the children's upbringing the schools present centres of activity, together with the facilities offered by the recreation clubs. Chapters of boys' and girls' national organizations, such as the Boy Scouts, Girl Guides, Cubs, and Brownies, are supported enthusiastically by the communities, and participated in vigorously by the children.

The school boards of both sections of the area take pride in the standards of education being offered, as evidenced by the periodic reports of visiting inspectors, and are in constant endeavour to maintain those standards.

General Amenities at Central Patricia and Pickle Crow

by

J. T. Ward*

TAKING into consideration the isolated location of the two mines and the many problems which have had to be met in connection with transportation and the high cost of such transportation, the amenities offered to both married and single employees can be considered excellent. Both mines commenced operation within a few months of each other, and are located only some five miles apart, therefore they have had to meet similar conditions at about the same time in the life of the mines. It is natural, therefore, that the two campsites should be operated along the same general lines, and that living conditions should be somewhat similar. A brief breakdown of the various facilities available to employees follows:

Townsites

Both townsites have between 90 and 100 residences of frame construction. Some are privately owned, but the majority are company owned. Naturally there is some variation in the style of the houses, but the average consists of four rooms and a complete bathroom. All are wired for electricity and many are heated by means of furnaces. A few houses in each camp are steam heated, the steam being piped from the mine heating plant. The rents run from $15.00 to $25.00 a month, depending on the style of house.

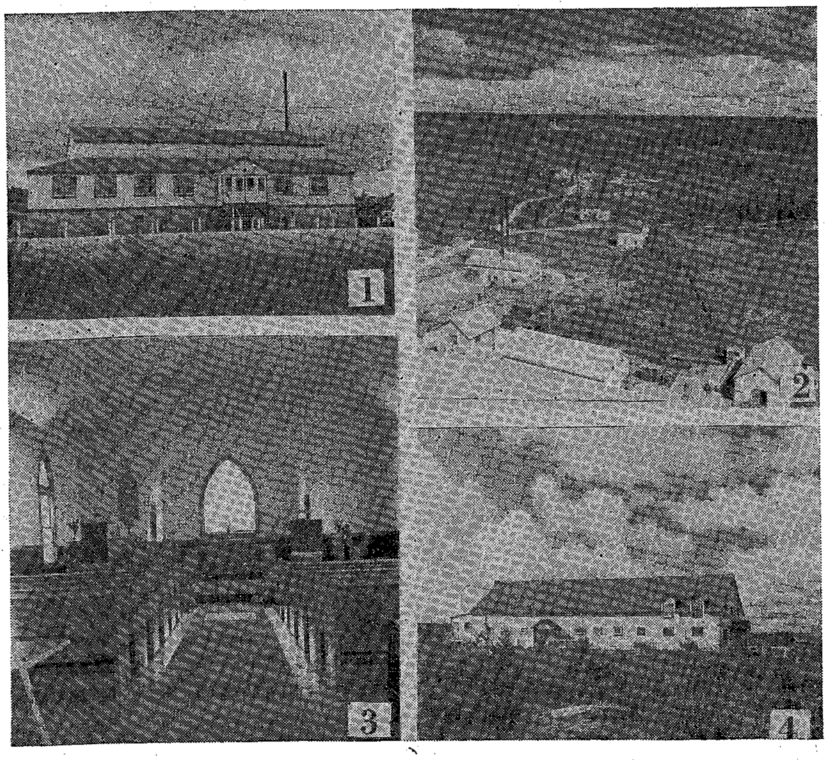
Retail Stores

Each mine operates a general store in which there is carried a full line of foodstuffs. Also stocked are work clothes and a selection of drygoods, together with a small assortment of hardware and household goods. The major portion of the stock is purchased during the summer months when the cheaper water transportation is available, but throughout the year fresh fruit and vegetables are brought in weekly by aircraft. Both stores are operated on the basis of returning to the two companies the operating cost and a reasonable rate of interest on the heavy investment, the stock on hand averaging around $65,000.00.

Bunkhouses

The bunkhouses in both camps are constructed on the same gen-

*Office Manager, Central Patricia Gold Mines.



1. Central Patricia Recreation Hall. 2. Portion of Pickle Crow Townsite. No 3. Shaft in distance. 3. Interior, All People's Church. 4. Pickle Crow Recreation Hall.

eral principal of two-men rooms. These rooms are equipped with standard single beds. Hot shower baths are installed in all bunkhouses. An attempt has been made to brighten these buildings by having the rooms painted in light shades, and different color schemes are used in each room. Every room is also equipped with two clothes-cupboards, bedside tables and chairs. The door has an individual spring lock and keys are supplied to the occupants. In one of the bunkhouses a large room is set aside as a lounge room.

Unmarried staff members reside in modern staffhouses which, however, do not contain dining room facilities.

Cookery

The two cookeries are run on entirely different methods. At Central Patricia the cafeteria system is followed and the dining room is equipped with hot and cold serving tables, and 4 to 6-men tables. The kitchen equipment consists of electric stoves, steam ovens, kettles, and dishwashing machine. At Pickle Crow the camp style method is followed, food being served by waiters at large tables. The kitchen is equipped with wood-burning ranges. Both cookeries are equipped with electric bake ovens, and bread is supplied from there to the retail stores. The rate for room and board is the same in both camps, namely $2.00 per day. The food supplied is of the best quality and a wide variety is served. No limit is placed on the amount each man may eat.

Recreation

The focal point of recreation in both camps is the community hall. These halls were constructed by the respective companies but are operated by the employees' recreation club, to which each member contributes 50c per month. Both halls are equipped with 35 millimeter sound movies, and two shows per week are run at each camp. Dances are held about twice a month. Both halls are equipped with a large and reasonably modern library, and there are facilities for such games as badminton, volleyball, checkers, chess, cards, ping pong, etc. Central Patricia have in addition two bowling alleys and three pool tables. A coffee shop and soda fountain is run in conjunction with both clubhouses, and light meals can be obtained in addition to the usual soft drinks and ice cream. The main auditorium of each hall is equipped with a stage suitable for amateur theatricals or concerts. The employees' recreation clubs sponsor outdoor sports of all kinds, and both camps have tennis courts and outdoor hockey rinks. Soccer is played extensively by the new Canadians. Skating, skiing, hardball and softball are engaged in actively. Central Patricia is fortunate in its location in so much that a river runs through the camp and this permits water sports in the summer months. In addition both camps have enclosed curling rinks, each with two sheets of ice. These clubs are extremely active, and while not sponsored by the employees' recreation club, the majority of members are employees of the two mines and financial assistance has been given by the companies. Inter-mine rivalry in practically all sports is very keen, and cups have been donated for inter-mine competition. Good fishing and hunting are available in the district.

Other Facilities

The Anglican St. Andrew's Church for All People, and the Roman Catholic Church of the Holy Rosary, are situated at Central Patricia, and services are held there and also at Pickle Crow Hall each Sunday. There are also active Sunday schools in both communities.

A hotel is situated in each camp, both having beverage room facilities. The Pickle Crow Hotel also has a cocktail bar.

Transportation to rail is by Canadian Pacific Air Lines daily service to Sioux Lookout, together with a once-weekly flight to Winnipeg via Red Lake and Kenora.

Telegraph facilities of both Canadian National and Canadian Pacific systems are available. In addition, long distance telephone communication can be made through a hookup of the local Lake St. Joseph Telephone Company, Canadian National Telegraph, Sioux Lookout Telephone Company, and the Bell Telephone System. The above mentioned Lake St. Joseph Telephone Company is jointly operated by the two mines and supplies the local telephone system. The two mines are the main subscribers, but some telephones are installed in private residences.

Lake St. Joseph Transportation Company

by

C. M. Low*

THE LAKE St. Joseph Transportation Co. is jointly owned by Central Patricia and Pickle Crow Gold Mines. The company owns and operates the portages on the Root River, which form part of the waterway from Hudson, on the Canadian National Railway to the mines in the Pickle Lake Area. It also maintains the road from the East end of Lake St. Joseph to the Mines. This road is the last stage on the route.

Previous to 1934 all supplies for the Pickle Lake Mining Area were taken in by float plane or by tractor train in the winter. In that year an agreement was concluded between the Mines and the Northern Development Department of the Ontario Provincial Government for the improvement of an old Hudson's Bay Company route from Lac Seul to Lake St. Joseph so that freight could be handled by scow from Hudson on the Canadian National Railway to Doghole Bay at the eastern end of Lake St. Joseph. Central Patricia is only some twenty miles north of this point and Doghole Bay is about 170 miles from Hudson by water.

The Root River flows easterly from Lake Otatakan, which is about 20 miles west of Lake St. Joseph, to a point where it is only five miles south of that lake. At this point the course of the river veers sharply from easterly to southerly. Here also a small stream, the Root Creek, joins the main river and, at one point on its course, the Root Creek flows within half a mile of the waters of Lake St. Joseph.

The improvements included the building of three dams, of timber cribs with stone filling, at Nattaway, Lynx and Flower Portages and of installing marine railways at these dams to move the scows from one reach of the River to the next. Each of these marine railways consists of a low heavy timber car running on wide gauge rails. A steel cable is attached to the car and by it a steam hoist pulls the car, with a loaded scow on it, out of the water of one reach and lowers it into the next.

The bed of the Root Creek was cleaned and widened to a point about three miles from Lake St. Joseph, where a dock was built and a hand derrick, later replaced by a steam derrick, was erected. To get over the remaining three miles to Lake St. Joseph a stretch of standard gauge railway was constructed. At the Lake St. Joseph end of this line a second dock was built and hand derrick erected. The "rolling stock" on this railway consisted of two flat cars and a 12-ton gasoline locomotive. This was later replaced by a 14-ton gasoline electric locomotive. The scows, coming up the river, were unloaded on to the flat cars. The freight was hauled to the north end of the railway and there unloaded into other scows on Lake St. Joseph. Recently the rails at each end of the railway have been run under water and some alterations made to one of the flat cars so that it can be lowered into the water, the loaded scow run on to it and the whole thing taken over the railway and launched into Lake St. Joseph. This has eliminated a great deal of freight handling and consequent breakage.

All the original work was carried out for the Northern Development Department by the Hydro Electric Power Commission of Ontario who were installing a power plant at the east end of Lake St.

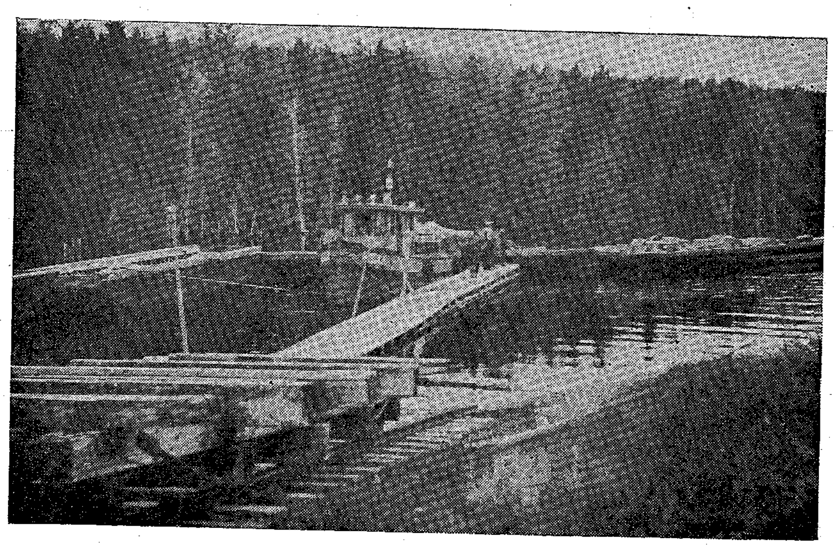


Photo: Canadian Metal Mining Association.

Portage at Nattaway Rapids.

*Manager.

Joseph. Raising of the water of Lake St. Joseph, as part of the hydro power scheme, greatly assisted navigation of the scows from Root Portage to Doghole Bay. Later alterations and improvements of the portage system were carried out by the company at its own expense.

Docks and storage warehouses were built by the mines themselves at Doghole Bay at the east end of Lake St. Joseph, where the freight is landed.

Some freight was moved over the portages in the late fall of 1934 but the first full season was 1935. This freight was stored at Doghole Bay as it arrived. Such of it as was required by the mines before winter was flown by plane a distance of 20 miles to Pickle Lake. The remainder was handled by tractor train as soon as there was enough ice and snow.

In 1936 an agreement was entered into between the company and the Provincial Government for the construction of a gravel road, eight feet wide, from Doghole Bay to the mines, the Dominion Government putting up a large part of the cost. Construction was started in August 1936 and the road was completed about a year later. Sufficient work had been done by the early winter to enable the 1936 freight, which had been stored at Doghole Bay, to be moved by truck instead of tractor train. In the summer of 1937, and in subsequent seasons the freight has been moved up the road by truck. The cost of maintaining this road and making improvements, such as widening and making more passing places, is borne by the company.

Funds for the operation and maintenance of the portages and for the upkeep of the road and the docks etc. at Doghole Bay are provided by the "portage dues" and "road tolls" which the company collects on every ton of freight moved over the route.

Six thousand tons of freight were carried over the system in 1939. This dropped to a minimum of 2400 tons in 1943. At present some four thousand tons move over the portages in a season of 5 months and one to two weeks. The two mines by whom this enterprise was started are not the only ones to benefit from it. The Hudson's Bay Company and a number of smaller concerns make use of the route in whole or in part.

Transportation to Pickle Lake Mining Area

by

Bertram H. Wilson*

DEVELOPMENT of any mining camp involves much more than the finding of the ore necessary for a producing mine. There has to be made available the facilities which will allow the profitable extraction of such ore. Of these facilities, one of the first essentials is adequate transportation". The foregoing, an excerpt from the February 20, 1947, issue of the "Northern Miner", provides a brief summary of the aims and objects of The Patricia Transportation Company Limited, a company incorporated in 1931 for the express purpose of providing year-round freighting services, by land and by water, to the Canadian Midwest Mining Area.

While the company's operations were initially carried out by water transportation by boat and scow during the summer months, and by tractor train during the winter months, the development and progress of what used to be isolated areas, and the opening up of those areas by all-weather roads, has been reflected in the operations of the company, which has now expanded its facilities to include all-year-round freighting services by highway truck transport. The company still maintains its freighting services by boat and scow during the summer months, and by tractor train during the winter months, to those areas not yet provided with connecting highways, and one of the areas so served is the Pickle Lake mining area in the Patricia District of Northwestern Ontario.

The railhead for all ground freight moving into the Pickle Lake Mining District is Hudson, on the main line of the Canadian National Railway, some 250 miles east of Winnipeg, and 10 miles west of Sioux Lookout, the chief base for flying operations in the area.

Diesel-powered tugboats, towing as many as seven or eight scows, each carrying up to 15 tons of freight, start out from Hudson on their 180-mile trip to Doghole Bay, at the northern tip of Lake St. Joseph, the northern terminal of the Pickle Lake water route. The scow "swings" cross Lost Lake at Hudson, head down the English River into the eastern arm of Lac Seul, and travel to its northeastern end to the mouth of the Root River. The swings proceed up the Root River to Nattaway, the site of the southernmost of four marine railways made necessary by non-navigable obstructions in the natural water route.

A marine railway consists of a track leading into the water at both ends of the portage, along which a flat car is drawn by means of cables attached to the drums of a steam-driven hoist, the hoist usually being located at or near the centre of the portage. The flat car is lowered down the slope into the water until it is completely submerged, the loaded

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scow or barge is floated over it, and the car is then drawn up the rails onto the portage, carrying the scow and its freight. The hoist lowers the flat car and its burden down the other side of the portage, and the scow is floated clear and tied up to await the balance of the scows in the swing, and, in some cases, the tugboat. The flatcar is then withdrawn, usually empty, but sometimes carrying an empty southbound scow, and returned to the other side of the portage to bring over the next scow. This process is repeated until the whole of the swing has been transported over the portage, and the swing is then re-aligned for the trip up the Root River to the next Marine Railway at Lynx Portage, where the portaging process is repeated. Portaging operations are again repeated further up the Root River at lower Portage, and the swing then continues up the Root River as far as it is navigable, stopping at South End, where the Root River becomes a small creek impassable to boats, and here it becomes necessary to make a 3-1/2 mile portage to North End, Root Portage, on the southwestern end of Lake St. Joseph.

Prior to the 1949 summer navigation season it was necessary to unload the freight from scows at South End, load it onto railway flat-cars, transport it across the 3-1/2-mile portage, then reload it into 35-ton capacity scows assigned to the Lake St. Joseph haul. During the winter of 1948-49, however, at the suggestion of The Patricia Transportation Company Limited, Root Portage was converted to a Marine Railway, and now that this conversion has been completed it is possible to transport the loaded 15-ton capacity scows across the portage without the re-handling of the freight being necessary. As a result of this conversion to a Marine Railway it is now possible to operate refrigerator scows over the route, the scows being iced and well cooled out before loading, so that perishables and meats may now be transported into the area by ground freight, instead of having to be moved by airtransport as formerly.

Four Tugboats are presently assigned to the Root River Route between Hudson and Doghole Bay: the diesel-powered *Billie S.* operates between Hudson and Nattaway, making three scheduled trips weekly, northbound on Mondays, Wednesdays and Fridays; the diesel powered *Hercules* travels from Nattaway, meeting the *Billie S* at that point, across Lynx Portage to Flower Portage, and the gas-powered *Alligator* operates between Flower Portage and South End, meeting the *Hercules* at the former point. The *Hercules* and the *Alligator* are operated by the same crew, and, while either the *Hercules* or the *Alligator* could handle the scows between Nattaway and South End during high water levels, the extra boat is used because some time is saved through eliminating the need for portaging a boat at Flower Portage, and also because it is desirable to have a spare boat available in case of breakdown to any of the other boats. The channels between Flower Portage and South End are narrow and shallow, and the shallow-draft *Alligator* is most suited to this particular portion of the route, while the *Hercules,* being operated by diesel power and powered by a more powerful engine, is more efficient than the *Alligator,* and hence is used over the larger portion of the route between Nattaway and South End. The *Hercules*, a 35 ft. boat with a 10 ft. beam, powered with a 110 h.p. Gray marine engine, while smaller than either of the boats operated on the lake haul at either end of the route, could fill in and take the place of either of them if such becomes necessary, while the *Alligator* could take over that portion of the route assigned to the *Hercules.* Through the interchangeability of the boats assigned to the run, the operation is protected from interruption due to breakdowns. The steel tug *Pentagon,* assigned to Lake St. Joseph, and running between North End and Doghole Bay, completes the northern end of the water operation. The whole trip by water from Hudson to Doghole Bay requires some 2-1/2 days, depending on weather conditions and payloads hauled.

At Doghole Bay the scows are tied up to the docks, and the freight is transferred to Patricia trucks for the 25 mile haul to Central Patricia and Pickle Crow Gold Mines, and to the hotels, stores and individuals in the area.

The area consists of three separate localities, Central Patricia and Pickle Crow, the townsites of the two mines, and Pickle Lake, a community on the shore of the lake from which it derived its name, and all three areas are regularly served by the Company's highway truck transport service, the regularity of service being determined by freight arrivals by boat and scow, and shipments originating in the area for furtherance south to the railhead. The whole operation is typical of a gold mining area, in that inbound freight governs the operation, outcoming freight being neglig-

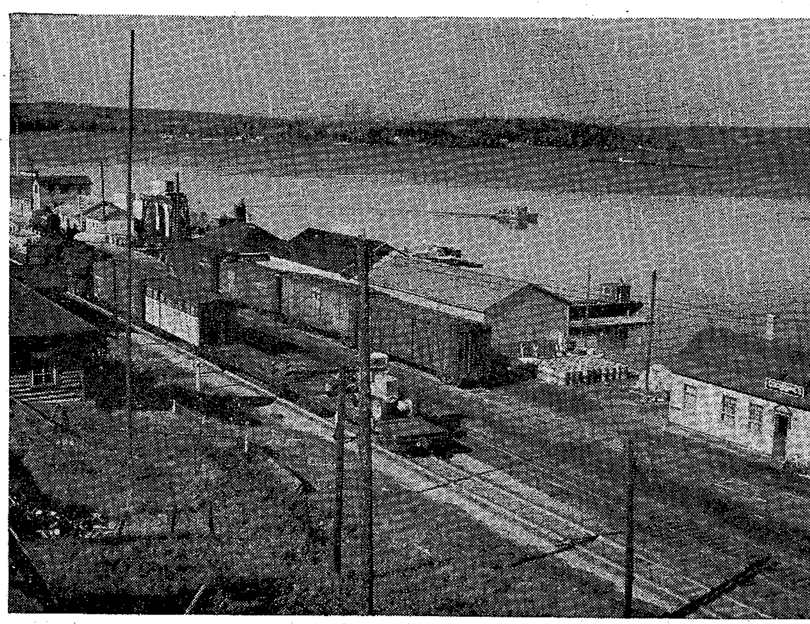

Photo: Canadian Metal Mining Association.
The Waterfront at Hudson, Ontario.

ible in comparison. In addition to operating the trucking service, the Company also undertakes the maintenance of the Doghole Bay -Central Patricia-Pickle Crow Road, and crews and equipment are kept busy during the whole of the summer freighting season in grading, dragging, filling holes, clearing brush, and all the necessary routines of keeping a road in good condition.

Equipment used for the truck haul from Doghole Bay to the Pickle Lake mining area varies with the amount of freight available for transport, though experience has indicated the desirability of having available three trucks for freight transport and a dump truck for road maintenance; a tractor is provided for the dragging and grading of the road. Tractor trucks and semi-trailers have been assigned to service on this haul, but it has been found that due to the comparatively short haul, and the heavy grades encountered on the road, that standard platform rack trucks with a capacity of between 6 and 7 tons best meet operating conditions encountered.

The Future

The Pickle Lake Mining Area hopes to be provided with an all-weather highway in the not-too-far-distant future, so that the area will not be cut off from ground transport during the break-up and freeze-up periods, and which would allow regularly scheduled highway truck transport service into the area. The present service, while founded on an efficient operation and reasonable rate structure, could not compete with a highway trucking operation, in that besides the actual cost of operating the freighting service by water, there is also a considerable expense necessary for the operation and maintaining of the portages, and for the maintaining of the private road from Doghole Bay to the area, all of which must be paid for by tolls on the freight moving to and from the area.

The Ontario Department of Highways is presently engaged in the clearing of a winter road from the railhead at Savant Lake, on the main line of the Canadian National Railway, some 73 miles east of Hudson, to connect with the Doghole Bay—Pickle Lake road, and the natural tendency will be for this road to be converted to an all-weather highway at some time in the future. When this happens, and the Pickle Lake area is linked up to the outside world by connecting highway, the duties of the scows and tugs will have been fulfilled, and, even though their place will be taken by a modern highway truck transport fleet, it will be a source of satisfaction to The Patricia Transportation Company Limited to have assisted in pushing back another frontier and opening up of another vast territory of Canada's Northland.

THE WORLD OVER

PROSPERITY FOR SOUTH AFRICAN GOLD MINES

South Africa has now had a few weeks to think over the devaluation of the pound. And it does not know quite what to make of it. Judging by the outward and visible sign of public opinion — the Stock Exchange — the reaction has been rather hesitant. That is possibly because in view of Mr. Snyder's firm refusal to raise the price of gold and Sir Stafford Cripps' repeated denial of the possibility of devaluing the British pound, South Africans were not ready to believe it when the latter course actually produced the former result.

Certainly share prices rose sharply at first, but the gains were checked quite quickly and everybody called it a "healthy market." What really happened was that the British Treasury cashed in on the South African end of devaluation by liquidating huge blocks of gold-mining shares that it had acquired from the French Treasury. This made the market so healthy that people who expected a two-day rise decided that they had better take their profits too or miss them. As a result, prices settled down to a level which, though well above the pre-devaluation position, still did not do justice to the greatly improved outlook for the gold-mining industry.

The industry, however, has in fact been raised on a substantial pedestal. At present everyone seems to be frowning at the base of the pedestal to see if anyone is nibbling at it. There will be considerable nibbling, but beyond having its stem shortened a little. the pedestal will still stand.

Profits Rise

Working results for September were spectacular. Of the gold produced during the month just over three-quarters was sold at the new price of 248s. 3d. an ounce and the remaining portion at the old price before devaluation — 172s. 6d. This lifted total profits of the mines in the big groups two and a quarter times, the figure rising from £2,036,532. in August to £4,516,892 in September. A further rise is probable when results of a full month's working at the new price are published.

But this aggregate increase does not reveal the enormous benefit enjoyed by individual mines in the groups. South Africa's richest mine — Blyvooruitzicht — succeeded in selling the whole of its output at the new price and its profits rose from £299,039 in August to £530,872 in September. This is a record for any South African mine.

At the other end of the scale, Transvaal Gold Mining Estates, a company which had little future left as a gold producer and was becoming chiefly interested in forestry found its profit from gold increased nearly 20 times; it rose from £1,250 to £23,148. This mine also managed to sell all its output at the higher price. For the rest of the producers profits jumped to anything from twice to seven times the former level.

Particularly interesting were the results of the mines which had announced that, under the old price of gold, they had not long to live. The latest mine to make that gloomy forecast was Glynn's Lydenburg; its profits jumped from £761 to £12,792. Nigel, in the same category, showed a rise in profits from £1,305 to £17,253, while Witwatersrand Gold Mining Company, which made a loss of £19,736 in August, chiefly as a result of a pressure burst, came out £3,-033 on the right side in September.

The variation in the profit increases were due to two factors. Some mines sold more of their output at the higher price than others were able to do, while some of the low-grade mines had been including quite a lot of unprofitable ore with their payable output and as the pay limit under the new price dropped from an average of 3.1 dwt. to 2.13 dwt. per ton much of this ore became payable.