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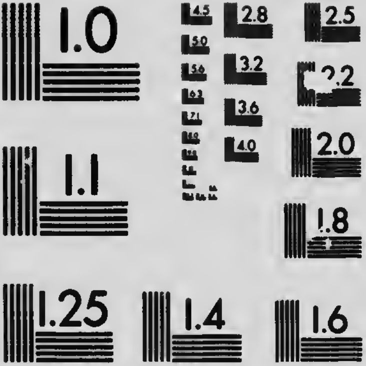

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The Mining and Smelting Operations

of the

International Nickel Company of Canada, Limited

WRITTEN BY THE STAFF

A paper read before the Canadian Institute of Mining and Metallurgy, at the Annual Meeting, Toronto, March 192[illegible]

TRANSACTIONS OF THE CANADIAN INSTITUTE OF MINING AND METALLURGY.

THE MINING AND SMELTING OPERATIONS OF THE INTERNATIONAL NICKEL COMPANY OF CANADA, LIMITED.

WRITTEN BY THE STAFF.

GEOLOGY OF THE CREIGHTON MINE.

THIS introduction gives a sketch of the geology of the Creighton [illegible] which, at the present time, supplies all the ore [illegible] the International Nickel Company of Canada. A fe[illegible] regarding the history of the mine are included.

The mine is in the Sudbury district, Ontario, six miles west of Copper Cliff where the smelter and general offices of the Mining and Smelting Division are situated. Mining was begun in 1901 and production increased so rapidly that within a few years the Creighton mine was known as the greatest nickel deposit ever worked—and it still retains that distinction.

The earliest records date back to 1856, when a surveyor noticed a strong deflection of the needle where his line passed close by the Creighton deposit. The locality was examined during the same season by Alexander Murray, a colleague of Sir William Logan, who in 1843 had founded the Geological Survey of Canada. Murray reported an "immense mass of magnetic trap" which he found to contain "magnetic iron ore and magnetic iron pyrites generally disseminated through the rock, the former in small grains; titaniferous iron was found in association with the magnetic ore, and a small quantity of nickel and copper with the pyrites."

Murray's description evidently refers to the norite with disseminated sulphides and he probably did not see the gossan covering the orebody at the foot of the ridge. Twenty-seven years passed without further discoveries, but immediately after construction of the Canadian Pacific Railway in 1883, many finds were made throughout the district and within a few years practically every deposit exposed at the surface had been staked. The Creighton deposit was re-discovered in the fall of 1886, and in 1887 a patent was issued to the Canadian Copper Company, which had been incorporated in January of the previous year. When the Algoma Eastern Railway was

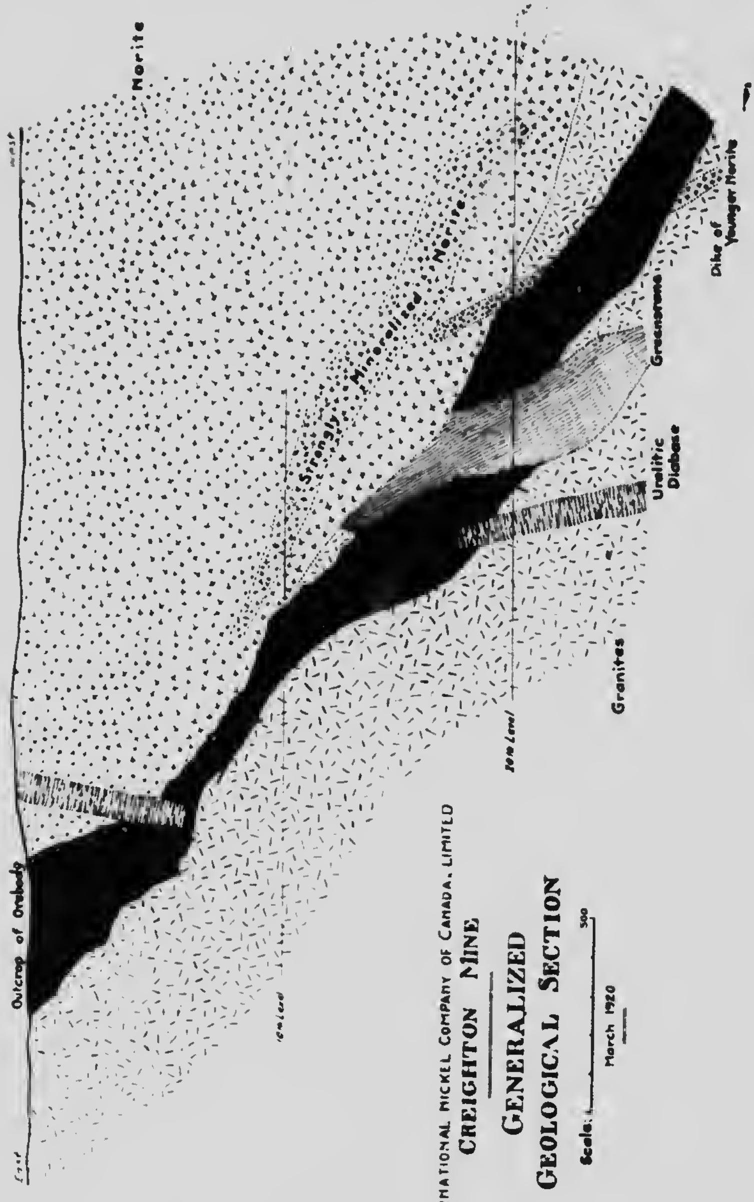

Norite
Outcrop of Orebody
Strongly Mineralized Norite
Greenstone
Uralitic Diabase
Dike of Younger Norite
Granites
THE INTERNATIONAL NICKEL COMPANY OF CANADA, LIMITED
CREIGHTON MINE
GENERALIZED
GEOLOGICAL SECTION
Scale:
500
March 1920

Figure 1.

extended to the mine in 1900, stripping was commenced and the first shipment made to the smelter at Copper Cliff in August, 1901. At the end of the year, the production was from 500 to 600 tons per day, and during the war period it exceeded 5,000 tons per day. The total production to the end of 1919 was 8,874,780 tons.

The greatest extension of the main orebody is along the dip, which is about 45°. It is usually sharply defined against the country rocks. The cross-section varies in form from oval to narrow lenticular, with wavy outlines, which, in some places, are strongly marked and irregular. The pitch length developed is over 2,500 feet. The level length varies from 1,000 to 400 feet. The horizontal width reaches the maximum of 375 feet with an average of over 100 feet. There is no gangue in

Photo by British & Colonial Press, Toronto.

Plate 1.—The open pit, Creighton mine.

the ordinary sense of the word but inclusions of rock are abundant in many places. The ore is very uniform as regards the proportions of rock and sulphides. The latter consist of pyrrhotite, chalcopyrite and pentlandite, with minute grains of magnetite disseminated through them. Several diabase dykes cut the orebody and the largest of these, as shown on the geological section (Fig. 1), exhibits a striking feature. The breaks in its continuity are not caused by faulting, but are due to the form assumed at the time of intrusion. This curious phenomenon is sometimes seen in small dykes that offset a short distance on coming to a soft bed lying between tougher ones.

Many writers have discussed the origin of the ore. The various theories may be divided into two classes: (1) magmatic segregation and (2) deposition from solution along zones of crushing and faulting. Both, in their various modifications, have had ardent supporters since the beginning of investigation down to the present time. Geological work at the mine has disclosed facts that indicate the origin of the ore by its intrusion in molten condition along a plane of shearing in the footwall rocks adjacent to the norite, after the latter had solidified. The most interesting evidence is offered by a dyke of comparatively fresh younger norite that intrudes the main body of norite and its footwall rocks. It is itself intruded by the ore, and also altered by it. The alter-

Photo by British & Colonial Press, Toronto.

Plate II.—Collar of No. 3 shaft, showing skips and man-cage.

ation is an unusual variety of contact metamorphism. It appears as a dark margin, varying in width from a quarter of an inch to six inches, against the ore. It is found in every rock with which the ore comes in contact, except the diabase dykes, which are younger.

Methods of Mining at the Creighton Mine.

The mine was first worked as an open pit, or glory-hole, from the surface. Later, when increasing depth made this system dangerous, underground methods were adopted.

From the open pit method, which was continued to a depth of 300 feet, the work has progressed through the various

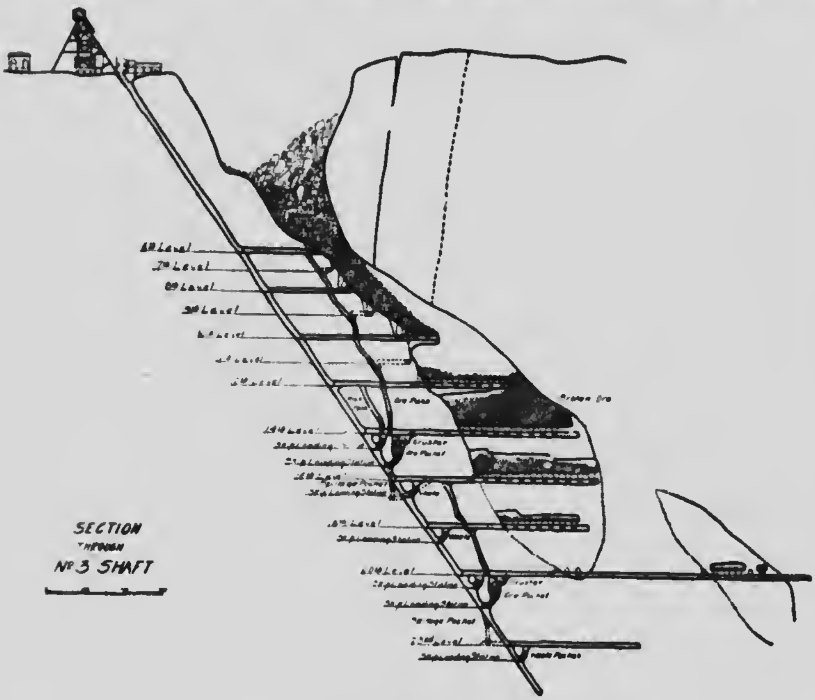


Figure 2.

stages of underground quarrying, shrinkage stoping with dry-walls and shrinkage stoping with under-drifts mainly in ore (in both of which round pillars were left supporting the hanging-wall) to the present method of shrinkage stoping with main drifts in the footwall, crosscuts in ore, and the use of rib pillars. This system has been employed in all work below the 6th level.

A synopsis of the development and stoping layout will show the orebody roughly determined in outline by diamond drilling and divided transversely into alternate stopes 6 feet wide and rib pillars of 15 feet. The shafts are in the footwall. Main haulage levels at intervals of 120 feet vertically, with drifts north and south from the shaft stations, are in the footwall at a convenient distance from the orebody. Crosscuts are driven through the orebody from footwall to hangingwall along the centre-lines of pillars, with boxholes at intervals of 15 feet on alternate sides. Intermediate drifts, with boxholes in the footwall are driven as required. A continuous system of ore-passes carries the ore to central underground crushing-stations.

Three shafts have been sunk. No. 1, a three-compartment shaft at an incline of 59°, was used to handle ore during the earlier operations in the open pit and from dry-wall stoping; it extended to the 5th level only. This shaft has since been dismantled.

No. 2 shaft, of four compartments, was sunk at an incline of 47°; it extends to the 12th level. It was in full operation until 1917, but is now used for men and supplies only.

No. 3 shaft was sunk at an angle of 55°. It is 33 feet 2 inches by 7 feet 6 inches, outside timbers, and is divided into five compartments, which are 5 feet 10 inches by 6 feet 6 inches in the clear. Sinking was commenced in April 1915, and ore was first hoisted from the 14th level crusher-station in April, 1917. Two compartments are equipped for handling ore, two for men, waste rock and supplies, and one for ladder, pipes and electric cables. The shaft has since been extended 380 feet below the 20th level, a total depth, on the incline, of 1,941 feet. The upper part of the shaft was sunk with 3⅛-inch piston drills. The centre V-cut was used, and rounds averaging seven feet were drawn. Below the 16th level, sinking was continued with Sullivan (D.R.6) machines, completing sections of, approximately, 200 feet by the use of a small auxiliary hoist and a rock pentice as protection from the operations above.

The sections through No. 3 shaft (Figs. 2 and 6) show in detail the method of shaft timbering, pocket construction, and loading-station equipment. British Columbian fir is used in timbering. Wall-plates are 8 by 10 inches, dividers 8 by 8 inches, and studdles 4 by 10 inches. Concrete piers are placed across the shaft beneath every fifth wall-plate, and also beneath station levellers.

Ore-skips of 9-ton capacity are operated in balance. Rock-skips, cages and supply trucks are interchangeable and are operated in balance. Thirty men are hauled in each cage. Waste-rock skips are of 5-ton capacity.

Skip and cage tracks are 4-foot 6-inch gauge. Rails are standard C.P.R., 85 lb. to the yard, connected with angle splice-bars and Harvey grip-thread track-bolts with spring washers. Lundy tie-plates are placed on each sill and the rails are held by 9/16 by 5½-inch spikes. One anti-creeper is attached to each rail. The entire absence of rail-creeping under conditions of heavy production has proven this construction most satisfactory. A 24-inch gauge track of 40-lb. rails is placed between the heavier rails in one of the cage compartments and is used to transfer haulage locomotives and cars.

Shaft rollers are made of six-inch tubing 12 inches long. The spindle turns in brass bearings held in an iron frame. All parts are interchangeable. The rollers are placed 60 feet apart and are offset alternately two inches on either side of the compartment centre; this allows the turning of the rollers end for end in the frame as grooves are worn by the rope.

A 16-inch diameter air-main is continued full size to the 20th level, and is held at its lower end, and again at the 16th level, by yokes resting on 18-inch I-beam bearers. Grooved blocks (6 inches by 8 inches by 3 feet 10 inches), placed at convenient intervals on the footwall plates in the manway compartment, support the pipe lines. The bend in the pipe at the shaft collar is held rigidly in concrete, and an expansion joint is provided near the surface. A six-inch pump column and four-inch fresh-water main are held by clamps through the wall-plates. Two four-inch pressure reducers are placed in the water main, and a two-inch reducer in each branch line.

Submarine power- and signal-cables are held on the end dividers. The power-cable is of three-inch diameter, three-conductor, 400,000 circular mill, with 2,600-volt insulation, the average load being 175 amperes at 550 volts. The signal-cable contains 15 wires. Charging-cables for electric storage-battery locomotives are two-conductor, 400,000 circular mills. Mine lighting is obtained by using from three to five K.W. transformers on each station, fed from the power circuit. Stations are equipped with return electric-bell system, call buzzers, signal lights and telephone.

Ore is drawn from two loading stations, one below the 14th level, and the other below the 20th level. The stations are equipped with measuring boxes having capacities of 7½ and 9 tons respectively. The spillage pockets are cut below the loading pockets as shown on the sections through No. 3 shaft (Figs. 2 and 6). Shaft rails are carried across these pockets on 18-inch steel I-beams 32 feet long.

Shaft stations are cut the same width as the shaft for a distance of 60 feet except on the crusher-station levels, where the full width is maintained beyond the crusher. Waste-rock pockets of 100-ton capacity, are placed with centres 52 feet horizontally from the grade-line of the shaft.

To control ventilation and as a means of fire protection, a reinforced concrete wall eight inches thick is placed 38 feet from the shaft across each station. Openings are left for protection against concussion due to blasting. These openings are covered with ⅜-inch steel plates swung on hinges. Double doors in the centre are made of ¼-inch steel plate lined with two-inch planks. These doors are automatically opened and closed by means of a 4-inch by 5-foot air-cylinder, the valve being conveniently placed for operation by the locomotive driver. A smaller door similarly constructed is placed near one end of the wall for use of workmen. Fire-hose and extinguishers are provided at convenient places behind the wall.

All timbering on stations, and for a distance of 25 feet above and below the station in the shaft, is fireproofed with a covering of expended metal and gunite.

Air-mains, water-mains and drainage launder are carried in a covered concrete conduit beneath and along one side of the station. Latrines are built into the reinforced concrete wall, against one side of the station, and are of the same construction as the main wall. They have concrete floors and drains, and are ventilated by a 12-inch pipe, which extends through the wall and a few feet into the upcast shaft.

Ore Passes.—A series of raises, 8 by 8 feet, in the footwall are driven between main haulage levels (Fig. 2). The raises are commenced from the sides of the station crossents, a brow point for the control gates is carefully determined and from this point continued to the level above, at an angle of 65°, forming a continuous ore-pass into which ore is dumped from the haulage levels and delivered to the crusher below. The ore is controlled on each haulage level by bent-finger rails and at the crusher by a baffle gate. All control gates are operated by air-cylinders.

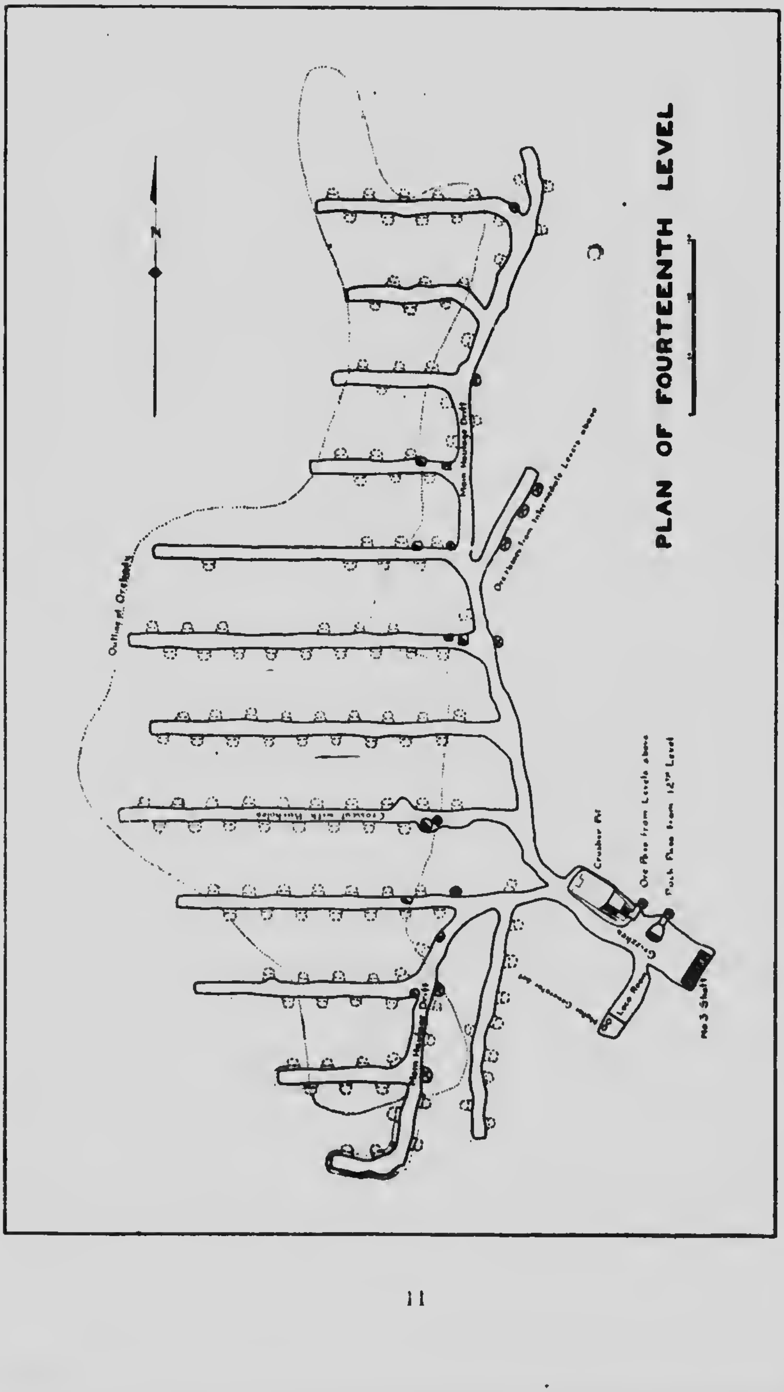

PLAN OF FOURTEENTH LEVEL
N
Outline of Orebody
Main Haulage Drift
Ore Passes from Intermediate Levels above
Crusher Pit
Ore Pass from Levels above
Rock Pass from 12th Level
Grizzlies
Loco Room
No 3 Shaft

Figure 3.

Level Development. — Footwall drifts, or main haulage drifts, are run at a convenient distance from the orebody (from 25 to 35 feet) with as few curves as possible. Crosscuts are turned off every 75 feet on the centre-lines of pillars and are driven to within a few feet of the hangingwall. Boxholes are raised from the sides of the crosscuts on alternate sides every 15 feet. In this manner a crosscut serves two stopes, with boxholes at 30-foot centres along the sides of each stope. Boxholes are also placed along the main drifts where necessary. Main haulage drifts and crosscuts are driven 11 feet wide and 10 feet high and give a clearance of three feet on each side of the tramcars. (See Fig. 3.)

In each pillar a manway raise is driven in the ore along the footwall contact from level to level, and equipped with ladder-way, pipe lines, and steel chute. These raises have the following functions:

(a) They form a systematic means of access to the stopes. As backstoping progresses, successive small openings are broken through from the stope on each side. Sharp steel is delivered through the chutes from the level above and dulled steel passed to the level below.

(b) They serve as a ready passageway between levels and are useful as an independent travelling way out of the mine.

(c) They form a simple and efficient means of ventilating the stopes and are of ample capacity for air currents to the lower levels.

(d) They assist in recovering the ore in pillars after the stopes are exhausted.

(e) They are used for compressed-air and water pipes from which branches are taken into the stopes.

In some parts of the mine intermediate levels are necessary owing to the flat dip of the footwall. These drifts are 9 by 9 feet in section and are driven in the footwall, but nearer the orebody than are the main haulage drifts, the work being guided as to actual location of footwall by the manway raises previously driven. There may be as many as three intermediate drifts at intervals of 30 feet vertically between haulage levels, the length of each being governed by the local dip of the footwall. Boxhole raises are driven beneath each stope and also to the manway raises. The latter are not equipped with chutes until such time as the pillar is to be drawn. Ore- and rock-passes are raised from the main haulage level to connect with the intermediate drifts above, the rock-passes to be used later as manways during the pillar-drawing stage. On the intermediate levels, tramming is done only at such times as the stoping machines are cutting back along the footwall, and at the end of the drawing-off stage, after the bulk of the ore has been drawn through the main-haulage-level chutes.

One of the lower levels has been equipped with a system of intermediate footwall drifts and branching ore-passes, which are directly fed by boxholes in the drifts. By this method it is hoped to handle the ore without intermediate tramming.

Rock Drills and Steel.—The footwall granite, in which a large part of the development work is done, is hard and of rather coarse texture; it contains almost no fracture planes. The greenstones and associated granites in the lower section of the mine are somewhat easier to drill and to break. The ore is easily drilled but being rather difficult to break in the comparatively small development openings, it requires as many holes drilled as in the rock.

Photo by British & Colonial Press, Toronto.

Plate III.—Drilling in the face of a footwall drift.

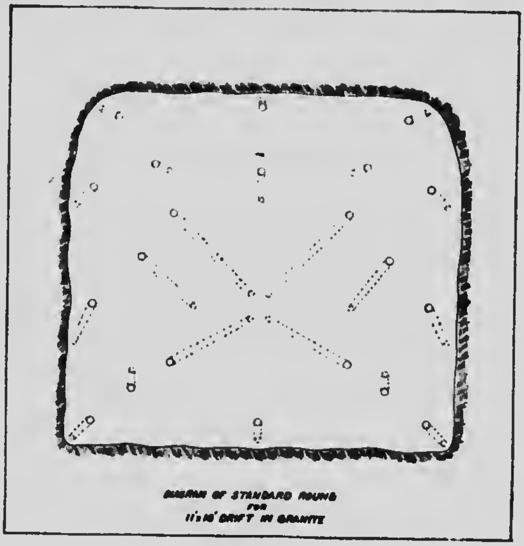


Figure 4.

The same rounds are drilled in all drift and crosscut headings, varying only in the number of 'easers.' The four-hole centre-pyramid cut is used, and from 17 to 22 holes, usually 19, are drilled in a round. Great care is taken to bring the cut-holes near to the point of intersection, and to properly place the 'easers.' Cuts are blasted and enlarged separately, before the 'square-up' is blasted. Rounds varying in length from 6½ to 7½ feet are broken. Polar Forcite of 40 per cent strength is used for all classes of work. Experiments with different types and lengths of rounds have been conducted. Rounds averaging ten feet in length were broken during a period of 30 days with the same speed per machine-shift and less powder per foot driven, but the large amount of broken rock to be handled interfered with the cycle of operations.

Figure 4 shows the positions of holes as drilled in tough granite to break a round of [illegible] feet. By placing the cut-holes slightly lower and doing away with the two lower 'easers,' the standard 19-hole round in greenstone will clear 7½ feet, to the bottom of the holes.

Two Sullivan (DRC) drills are used in each heading. The air pressure is 100 lb. Hollow hexagon steel 1⅛ inches in diameter is used. Cross-bits, with 14° and 5° taper and reaming edge, are gauged 2½ inches on two-foot 'starters' and decrease ⅛ of an inch per foot to a length of eight feet; thereafter the decrease is 1/16 of an inch per foot to 14 feet and there is a difference of ⅓ of an inch between the gauge of 14-foot steel and that of 20-foot. As it is necessary to drill 16-foot holes in the stopes, and as the bits of 16-foot steel are 1-5/16 inches in diameter, this bit gauge is used throughout the mine in order to avoid confusion and to standardize shop work. Experiments are being made with small-gauge bits on development work, commencing with two-inch 'starter' bits and finishing with a 1⅜-inch bit at 10 feet.

A 7½-foot round in granite is drilled in one and one-half shifts, or three drill-shifts. During the period from July to December, 1919, the advance per drill-shift in drifts and crosscuts was 2.3 feet, consumption of powder averaging 19.4 lb. per foot driven.

Drills are tested on a granite block in the repair shop and must cut three inches per minute using a 2½-inch bit.

Stoping.—The longitudinal section through the orebody (Fig. 5) shows the system of pillar-and-stope arrangement. The pillars extend throughout the mine from footwall to hangingwall and are at right angles to the long axis of the orebody. In a section of the mine, from the 12th to the 16th levels, near the south end of the orebody, the pillars were increased in width to 25 feet around the manway raises, with a corresponding decrease in the width of the stopes. This change was considered necessary on account of the more friable nature of the ore, but it has not affected the essential feature of the narrow pillars, namely, the cheap recovery of the ore in the pillars more or less concurrently with the finishing of the stopes.

Floor Cuttings.—Headings 20 feet wide are first carried along each side of the stope at an elevation of 18 feet above the grade line of crosscuts. These headings are connected across the stope along the footwall and are extended toward the hangingwall by ordinary stoping methods from one boxhole to the next, the benches being so arranged that broken material is thrown by the blast into the nearest boxhole. No shovelling of broken ore is necessary. With the stope-floor heading seven feet high, the ore broken per drill-shift amounts to about 37 tons.

The next stage in the cutting of floors is accomplished by raising the elevation of the stope along the footwall and carrying a breast stope toward the hangingwall. As the under side of this breast is the roof of each of the two headings just described, the result is a clear space 60 feet wide with a ridge in the centre of the floor. All the broken ore possible is drawn through the chutes in advance of the breast, in order to keep sufficient head room for the next bench above. In stopes in which this stage of floor cutting is carried on, the duty per drill-shift averages 110 tons, making an average drill-shift duty of 80 tons for floor cutting in these stopes.

Back stoping progresses from footwall to hangingwall, carrying a bench from 10 to 12 feet high. Eighteen holes, drilled horizontally from four separate set-ups, are usually sufficient to bring down a bench. Sixteen-foot steel is used. Broken ore is drawn in advance of the benches and a height of from six to eight feet below the back is maintained from the footwall to the working face. This gives comfortable head room and affords an easy opportunity for the inspection and sealing of the roof. As succeeding benches are carried forward it is necessary to cut upward along the footwall, and if this

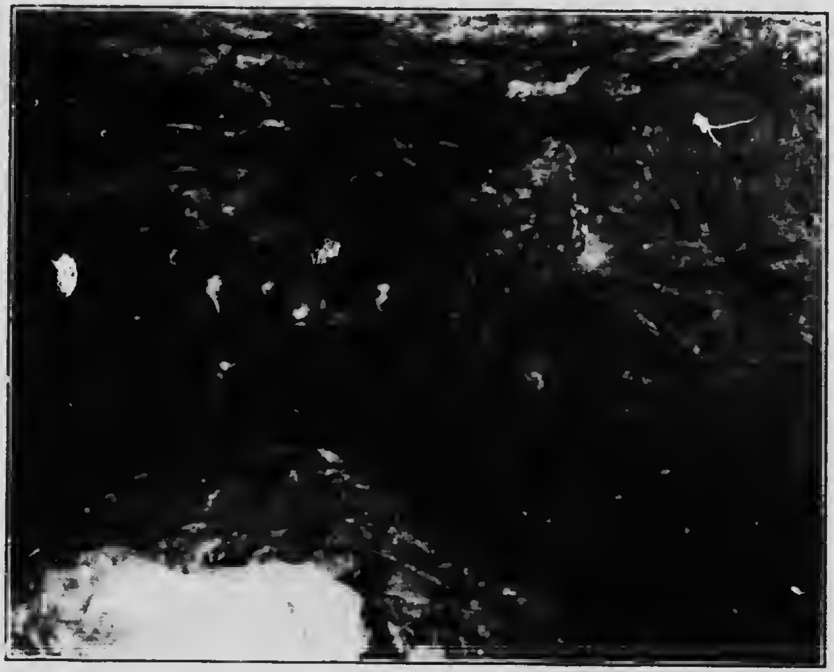


Photo by British & Colonial Press, Toronto.

Plate IV.—Back stoping.

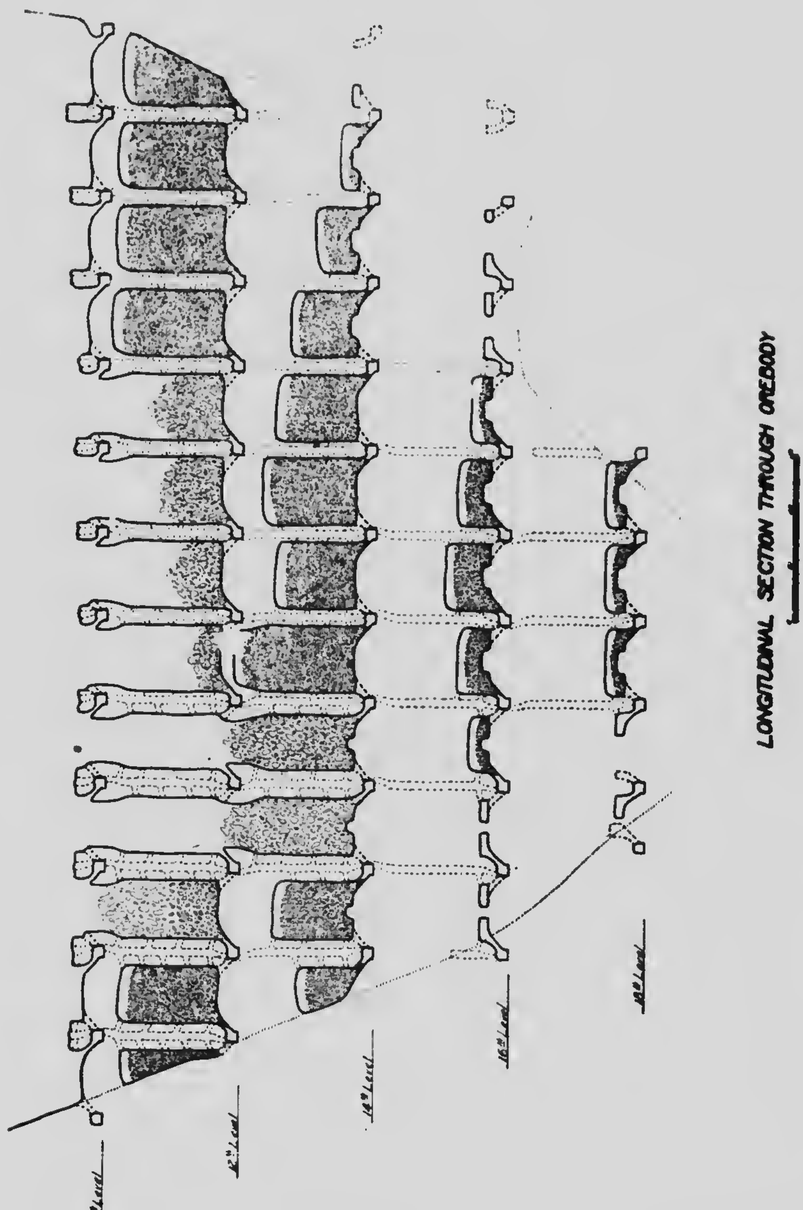


Figure 5

should be quite flat a somewhat lower duty per drill-shift is obtained.

Ore not broken fine enough to pass through the chutes below is blockholed. Rand (DDR13 and BC26) machines, fitted with chucks to take 1⅛-inch hollow hexagon steel are used in this work. Incidentally, short ends of broken steel are utilized here. It is inevitable that a certain number of pieces of ore, too large to pass the chutes, are either buried in the blast or carried beyond reach of the blockholer, and it is necessary to blast these in the chutes later.

Floor Removal.—When the ore has been broken to within a convenient distance from the floor of the stope above, the floor can be shot out any time after stoping above the next level is finished. A thickness of 25 feet is usually sufficient to maintain safe working conditions. To avoid the constant attention of scalers, which otherwise would be necessary in that section of the mine where the ore is softer or where a pillar shows signs of weakness, a few stulls of round timber resting on the broken ore are used to indicate possible movement in any part of the roof. These stulls are recovered before blasting. A section near the hangingwall is now thinned to about 18 feet in which vertical holes are drilled and an opening blasted through. Careful attention is given to the placing of the vertical holes to ensure a clear break-through and its subsequent enlargement. The entire floor is then removed by retreating in stages towards the footwall.

Duty in Stopes.—Sullivan (DR6) drills are used. Holes are drilled to a depth of 10 feet on the floor headings. In breast stoping, 16-foot steel is used, and for breaking down floors with vertical holes, 20-foot steel is often required.

The drill-shift efficiency taken over six months, with drills about equally distributed between floor headings and back stoping, is 83 tons per shift of eight hours, tonnage broken per man in stopes is 24, powder per ton broken (which includes blockholing powder) is 0.4 lb. All drilling in stopes is done during the single day shift, the blasting crew coming in on the afternoon shift.

Pillar Removal.—Pillars may be broken after the floors have been cut through and the ore has been drawn sufficiently to allow the pillar to fall when blasted.

Holes are drilled in the back and sides of the footwall manway between two levels and, if necessary to free the pillar from the hangingwall, in the floor and roof of the crosscut above. These holes are blasted with ordinary fuse, and usually at one time. Two hundred and fifty may be taken as the average number of holes necessary to bring down a section of fairly solid pillar.

The probable caving of the hangingwall on the removal of pillars was foreseen, and as shown in the section through No. 3 shaft (Fig. 2), part of the stope floor was left along the footwall of the orebody at the 10th level. The outer edge of this floor leaves a safe margin beyond the angle of repose of broken rock that might come from the area of flat-dipping hangingwall above. The last of the pillars above this point are in process of removal. Some of the pillars below the 10th

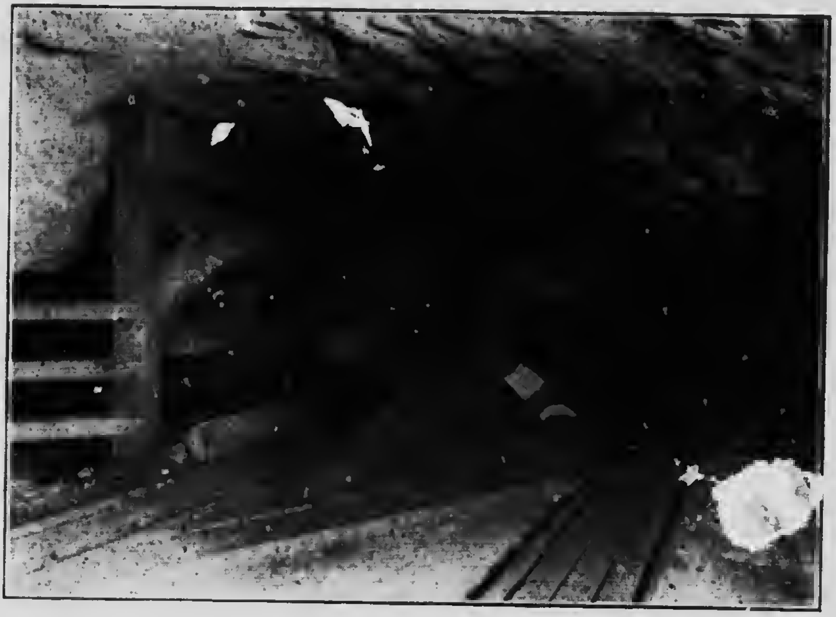


Photo by British & Colonial Press, Toronto.

Plate V.—Power tram-car dumping into an ore-pass.

level have been purposely weakened by cutting into them near the hangingwall. Some of these show signs of weakness above, and it is hoped to remove them with little trouble when the time comes, as the dip of the orebody is considerably steeper in this section.

Tramming.—On intermediate levels, and during the early stage of level development, 16-cu. ft. side-dump cars are used.

trammed by hand. All other tramming is done by storage-battery locomotives and 56-cu. ft. side-tipping cars.

Ten Baldwin-Westinghouse, five-ton storage-battery locomotives are in use, each equipped with 66 Edison Type A-10 cells, having a capacity of 375 ampere-hours.

A 50-K.W. D.C. generator placed on the 14th level will charge four locomotives at one time at a normal rate of 75 amperes each; this generator serves the charging stations from the 14th to the 20th level. A generator set of equal capacity is placed on the surface and is used to charge the locomotives above the 12th level.

An example of the life of locomotive storage-batteries may be shown by a set after 2½ years' service. This locomotive is delivering 210 cars of ore to the tipple over an average tramming distance of 600 feet on one charge. The length of time necessary to re-charge is five hours. The average service of cells is about three years.

Trains of seven cars are handled easily. Main-line tracks are uniformly graded 0.5 per cent, and branch lines in cross-cuts 0.75 per cent. Haulage cars weigh 7,000 lb.

Car tipples are constructed of timber fitted with steel wheel-tread, cast in sections six feet long, the curved surface of the tread being so developed that the dump wheel of the car is at all times at right angles to the plane of the tread. (See Plate V.)

No grizzlies are used at the dumping points into the ore-pass that feeds the crusher on the 14th level, but as the largest pieces of ore that can be drawn through the stope chutes sometimes give trouble at the crusher, the lower ore-pass has been equipped with grizzlies. The grizzly rails are constructed of 12 by 16-inch timber 17½ feet long, lined with 12-inch channel and 1¼ by 5-inch wearing plates. They are placed at an angle of 22° from the horizontal and spaced 28 inches at the upper end and 30 inches at the lower end. A row of discarded crusher plates, placed on a 30° slope at the upper end of the grizzly, takes the shock of the dumping load. Oversize is delivered to a blasting station at the lower end, where the large pieces are blockholed.

Sledging grizzlies over the rock pockets are made of 85-lb. rails turned base upward and bent about four feet from one end at an angle of 27°. This forms a slope on which the broken rock slides away from the car track, the sledging station being about three feet below the grade of the track.

The rails are spaced 10 inches and are held in place by wooden blocks made to fit the rails. Blocks and rails are held in place by a cover strip of 3/8 by 6-inch steel bolted through the b[illegible]r beneath.

The gauge of the track is 24 inches; 25-lb. rails are used

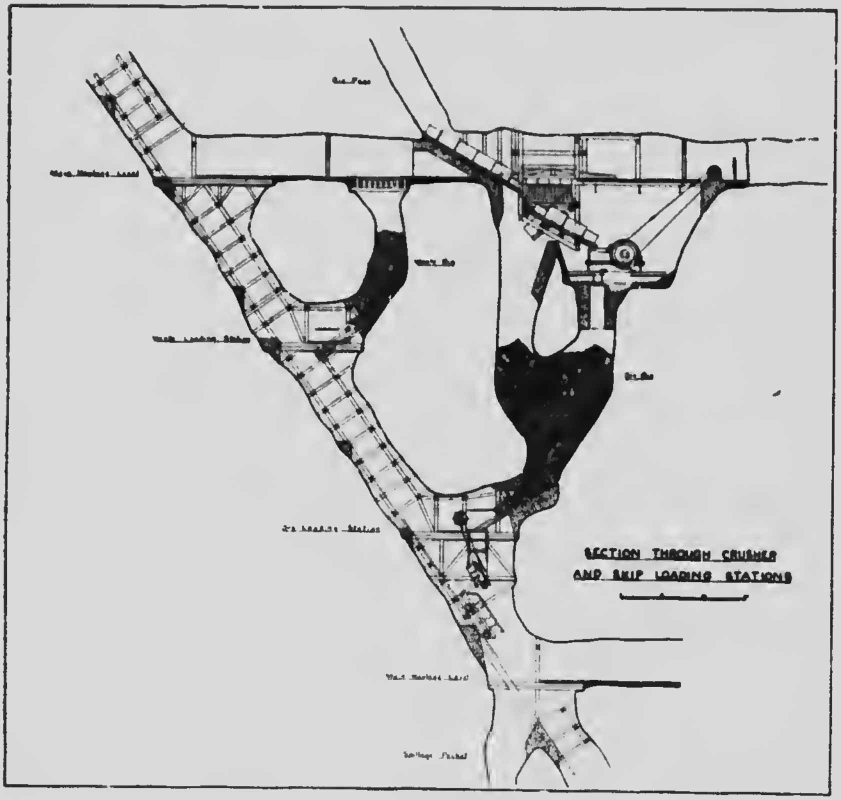


Figure 6.

on intermediate levels and 40-lb. rails for locomotive haulage. The minimum curve of 25-foot radius is maintained on haulage levels. Curves at the entrances to crosscuts are accurately laid out by the foreman by the use of a chart giving the points for different [illegible]gles of intersection. All ties are placed at two-foot cent[illegible].

The position of underground crushers is shown on the sections through No. 3 shaft (Figs. 2 and 6). Broken ore from the ore-pass and from the level on which the crusher is situated, is dumped over separate grizzlies (five-inch spaces) into a 3 by 42-inch jaw-crusher set at 6 inches, and the product falls into an ore-pocket from which it is loaded into the skips. Two of these crushers are in use, one on the 14th and one on the 20th level; they handle all the ore above their respective levels.

The crushers weigh 65 tons, and base and fly-wheels are sectionized for passing through the shaft. One of these crushers has handled 5,000 tons in two shifts of eight hours each. Water and grease are used to lubricate the pitman and journals, and cup grease for the jaw shaft. Waste water from the lubrication is collected in a sump and pumped to the drainage launder above. The crushers run at a speed of 235 R.P.M., and are connected to two 100 H.P. induction motors. One motor would furnish sufficient power to drive the crusher under normal load, but owing to the large starting torque required two motors are used; these also ensure constant speed under different conditions.

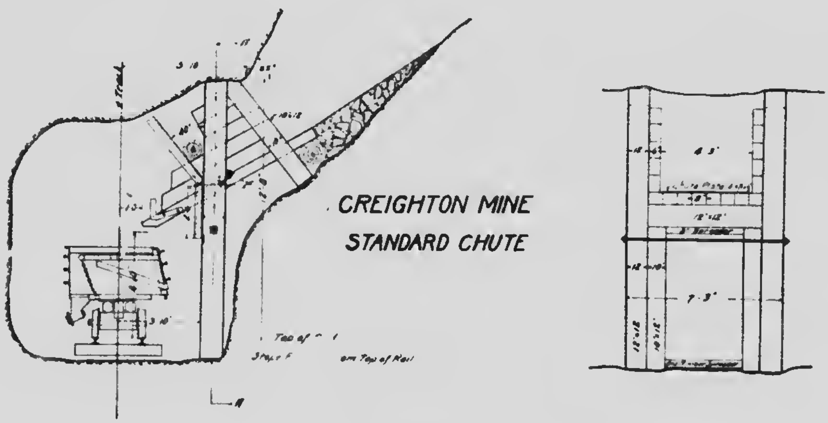


Figure 7.

Stope Chutes.—Chutes are shown in the drawing of the standard chute (Fig. 7). The essential features of the chute are height and location of the brow point upward to the stope floor, forming a funnel in which the larger pieces of ore can jamb near the lower end only. Careful attention to these details has lessened accidents due to chute blasting and has improved the tramming efficiency.

Tons trammed, taken over a period of six months, and including chute drawing and development shovelling, is 38 tons per trammer shift for power tramming, and 20 tons per shift for hand tramming. Tons hoisted per man underground is 7. Powder used in chute blasting is from 0.2 lb. to 0.3 lb. per ton trammed.

Pumps.—Seepage water from the surface is practically all that is encountered in the mine, aside from the fresh water brought down for rock-drills and drinking. During the rains and thawing periods, however, this becomes considerably more.

Two main pumping stations are provided, one on the 6th level, No. 2 shaft, and the other on the 16th level, No. 3 shaft, both pumping directly to the surface.

Photo by British & Colonial Press, Toronto.

Plate VI.—Loading tram-car from a stope chute.

Equipment on the 6th level consists of a Gould vertical 3-plunger pump geared to a 40 H.P. motor. Plungers are six inches in diameter and have a 12-inch stroke, the capacity of the pump being 250 gallons per minute, against a head of 457 feet. This is supplemented by a three-stage centrifugal pump and an air-driven plunger-pump, which are held in readiness in case of accident or during the wet season. The working time of this pump varies from three hours per day in the winter to full time during the first rains in the spring.

The equipment on the 16th level consists of a Gould pump of similar type, but of heavier construction. It is direct-connected to a 100 H.P. motor; plungers are 6½ inches in diameter and have a stroke of 16 inches. Suction and discharge diameters are 8 inches and 6 inches respectively, the height over all is 10 feet 3 inches and the capacity is 250 gallons per minute under a working head of 1,035 feet. The working time varies from six hours in the winter to 18 hours in the spring. The mine water passes through settling tanks placed between the launder discharge and the suction pump. These tanks are concrete lined and provided with by-passes to facilitate cleaning. The capacity of the sump is 17,000 gallons.

A small motor-driven vertical-plunger pump will be installed on the 23rd level. Small air-driven Cameron pumps are used in the various parts of the mine.

Ventilation.—Natural ventilation supplies ample air for the deepest workings. The two shafts and a winze that connects with the stopes at the south end are up-cast and the difference in the elevation between these openings on the surface and the bottom of the open pit, which is 300 feet deep, maintains a strong upward current of heated air. Short circuits to the shaft and between the north and south ends of the mine, above the 12th level, are prevented by doors in drifts and crosscuts. Below the 12th level the footwall manways become down-casts, owing to the bulkhead walls on the stations, the air current passing to No. 3 shaft through the lowest level. Propeller fans were placed temporarily in the station bulkhead walls on the 18th and 20th levels, but as development proceeded and more manways were opened, they became unnecessary.

Hoisting.—As all the production comes through the No. 3 shaft, every effort has been made to effect rapid and safe handling of ore, rock, materials and men. The ore is hoisted to the rock-house in nine-ton skips, working in balance, through the first two compartments of the shaft and the waste rock is hoisted through the third and fourth compartments to a rock-storage bin standing in front of, and lower down than, the ore dump. Men and materials are handled in steel man-cages, working in balance, in the third and fourth compartments, the skips being replaced by cages or *vice versa* as required. The changing of cages and skips is done quickly by storing the skips on a balcony above the collar and the cages at the collar, and transferring from either collar or balcony track to shaft-track by jump-rails which are swung into position by small air-hoists.

The ore hoist is of the Ilgner type and was built by the Wellman-Seaver-Morgan Company. The drums are 12 feet in diameter and 7 feet 6 inches wide, having a capacity of 2,100 feet of 1½-inch rope in one layer. One drum is keyed to the shaft, whereas the other is loose on the shaft and is provided with a multiple-tooth clutch, operated by an oil cylinder, which is supplied by the accumulator by means of a lever on the operator's platform.

Photo by British & Colonial Press, Toronto.

Plate VII.—The ore hoist.

Each drum is fitted with a post brake, set by weights and lifted by a brake engine operated by oil which is supplied by the accumulator. The clutch and brake are interlocked so that the clutch cannot be thrown out until the brake is applied, or the brake released until the clutch has been thrown in.

The hoist is provided with a mechanism for automatic acceleration, and an automatic slowing down and stopping device.

The accumulator equipment consists of a weighted accumulator having a capacity of 16 cubic feet, duplicate three-throw oil-circulating pumps with direct-current motors, also the

Photo by British & Colonial Press, Toronto.
Plate VIII.—The motor generator set.

necessary return tank and piping, connecting the brake cylinders with accumulators and return tank.

The hoist motor has a normal rating of 1,800 H.P. continuous at 550 volts and 40° rise, operating at a speed of 66½ R.P.M., and having a maximum rating of 3,600 H.P.

The motor generator set consists of a direct-current generator, a wound-rotor induction-motor, a steel-plate fly-wheel and a direct-connected shunt-wound exciter, all mounted on a common bed-plate.

The separately excited direct-current generator has a capacity of 1,500 K.W. based on a 40° rise.

The three-phase alternating-current, wound-rotor induction-motor has a normal continuous capacity of 1,400 H.P. and is designed for a frequency of 25 cycles and 2,400 volts. It is wound for six poles and has a full normal-load speed of 487 R.P.M.

The steel-plate fly-wheel is 12 feet in diameter, 21½ inches wide, and weighs 100,000 lb.

The direct-current inner-pole, shunt-wound exciter has a normal rating of 30 K.W. 40° rise, operating at a speed of 500/415 R.P.M. at 250 volts.

The hoist is designed to operate under the following conditions:—

Net weight of ore hoisted	Lb.	18,000
Weight of skip	Lb.	11,000
Weight of rope per foot	Lb.	3.5
Inclination of shaft		55°
Dept of shaft	Feet	1,800
Output in 7 hours	Tons	3,500
Output per hour	Tons	500
Hoisting speed	Feet per minute	2,500

Cycle—10 seconds for loading, 15 seconds for acceleration, and 10 seconds retardation.

The cables in use on this hoist are of 1½-inch diameter, six-strand, 16 wires ea[illegible] round 6 round 1 plough-steel Lang's lay with a breakin[illegible]in of 212,000 lb. Those now in use have hoisted over 6[illegible]0 tons per rope.

Rock and Man-Cage Hoist.—This is a Nordberg, two-drum hoist, and is driven through one reduction by a 350 H.P. Allis-Chalmers slip-ring variable-speed motor (alternating-current) rated at 480 R.P.M. at full load. The drums are 7 feet in diameter with a 4-foot face and each is fitted with parallel-motion post-brakes. The brakes are set by means of weights and released by oil cylinders. There is an automatic cut-off which operates in case of over winding or lack of current. A liquid rheostat, which gives a smooth and certain control of acceleration to the hoist, is used.

The clutch is of the axial friction type, set radially, and the motion of the moveable parts of the clutch is always parallel to the axis of the drum. The hoisting speed is about 1,100 feet per minute with a 5-ton load of rock, and the hoist is designed for a rope pull of 21,700 lb.

The cables are 1¼-inch, 6-strand, 16-wire, plough steel, Lang's lay with a breaking load of 138,500 lb. All cables are supported by six-inch idlers throughout the shaft, and by fleet wheels between the sheaves and the hoist, which is 140 feet from the shaft. The head sheaves are of the bicycle type, 12 feet in diameter with 4⅜-inch rope-groov,e and are carried on 12-inch journals 142 feet above the collar of the shaft.

The head frame is combined with the rock-house, which stands 71 feet from the collar of the shaft.

The four shaft-tracks of 85 lb. C.P.R. standard rails and 4-foot 6-inch gauge, extend from the collar of the shaft to the ore-dump in the rock-house and are supported by 20-inch I-beams throughout. The back legs are constructed of angles and plates and are anchored to concrete piers.

All four tracks from the mine pass over the rock-dump which is placed over a waste-bin situated about 20 feet in front of the rock-house and at a lower level than the ore-dump. This bridging of the rock-dump opening is arranged by sliding-rails which move in and out of line with the main track rails by means of a shuttle worked through a lever operated at the collar of the shaft. There is 47 feet of head room between the ore-dump and the sheave.

Ore-Skips.—Nine-ton, all-steel skips, with a capacity of 168 cubic feet and weighing 12,200 lb., are used for ore. These are 13 feet 4 inches in length over-all, 3 feet 11 inches in cross-section and are made up of ⅜-inch outside plate, ½-inch lining plate, and the whole is braced by ample angles and channels. The end and bottom are interlined by 2¾-inch hardwood to absorb the shock, and ½ by 6-inch manganese-steel bars are used to take up the wear of the sliding ore. The 16-inch wheels are of managanese steel, the front wheels having a 3-inch tread and the rear 6-inch. The axles are of 4-inch forged steel and are carried in tight housings which extend the full length and ensure ample space for lubricants. The bail is 15 feet 2 inches long, pivoted 3⅞ inches behind the end of, and 12 inches above the bottom of, the skip; it swings on a 3-inch shafting with ample bearings and reservoir for lubricants. The bridle bar is of 6 by 1-inch steel bars while the yoke is made up of heavy plate.

Rock-Skips.—These are of the same sturdy all-steel construction, being 11 feet long, with a 6-foot 6-inch wheel base; they have a capacity of 85 cubic feet and weigh 9,000 lb. The bail is pivoted 3 inches behind, and 10 inches above, the bottom of the skip-body, ensuring complete emptying, as in the case of the ore-skips. The wheels are also of manganese steel and of the same size as those on the ore-skips.

Man-Cage.—This is of $\frac{1}{8}$-inch steel plate riveted to trucks and weighs 8,800 lb. empty. The wheels are of manganese steel and the draw-bar is of Lowmoor iron annealed at frequent intervals. A screen door slides up and down, the weight being balanced by a spring roller, thereby entirely enclosing the men in a steel car.

Signals, Safety Appliances and Inspection.—The shaft, which is inclined at an angle of 55°, has no back-runners, and no chairs are used with cage or skips. The hoist, sheaves, cable, and shaft are inspected daily. A warning bell is installed to tell the hoistman when the skip is approaching the surface and he accordingly slows down. The load of the cage when carrying men does not exceed 85 per cent of the maximum weight of the other loads. The hoists have the ordinary safety appliances. The bell signals for hoisting and lowering are on the electric return-bell system. The switches are pull-type and the bells are rung simultaneously on all levels, collar of shaft, and hoist-house, the same signal being repeated by the hoistman for verification.

Operation of Hoists.—A Johnston and Johnston trip-recorder gives a continuous record of the trips of each hoist and some wonderful charts have been obtained from the ore hoist during 16 hours' operation. The best record has been 412 skips in 8 hours from the 14th level ore-pocket, 1,350 feet below the dump, with a load of $7\frac{1}{2}$ ton of ore. The average hoisting rate, however, is about 46 skips per hour with a $7\frac{1}{2}$-ton load from the 14th level, or 5,520 tons in 16 hours. This hoist has never been operated to its capacity.

Waste-Bin.—The waste is hoisted in the third and fourth compartments to a 22-foot by 36-foot (diameter) steel receiving-bin, holding 1,100 tons. This is either discharged through arc-gates directly into railway cars below, or else through finger-gates, on the side, into 56-cu. ft. cars to be hauled by electric locomotives to the open pit for disposal.

Rock-House.—This has been designed to handle a large tonnage and provide plenty of room for the required number of pickers. The different-sized particles are screened out and treated separately on different floors. The building, which has three picking floors, is of steel and brick, with cement floors, and stands on concrete pillars directly over the railway track on which the receiving cars draw the sorted product from the storage bins. The hopper-bottomed receiving-bin into which the entire ore product of the mine is dumped, is placed at the top of the building. From this the ore is drawn off through a gate controlled by rack and pinnion; it passes over 5-foot diameter feeding rolls which turn at a constant speed of about one revolution per four minutes undercutting the stream of ore. The ore then goes by gravity directly into the trommels (60 inches by 8 feet) on the upper floor, which are inclined at an angle of 8° and are revolved at the rate of 10 R.P.M. by 25 H P. back-geared Westinghouse motors. The under-size from the 6-inch openings in these screens goes directly to the second screen on the next lower floor. The over-size on the upper floor is discharged over cast-steel plates, inclined at an angle of 27°, into 36-inch Robin's rubber conveyor-belts travelling past the pickers at a rate of 35 feet per minute. The waste rock is taken out by the pickers and dropped down conveniently placed steel chutes which go down through the building to a sorted-rock storage-tank above the

Photo by British & Colonial Press, Toronto.

Plate IX.—Upper floor of rock-house.

railway tracks to be drawn off into railway cars, or the rock can be collected in cars on an intermediate floor for disposal in the open pit. The sorted ore remaining on the belt is carried to the end where it drops into crushers 48 by 15 inches set at 3½ inches. The product from the crusher is delivered into a screen, 48 inches in diameter inclined 8° and revolving at 12 R.P.M. The fines, which pass through its ¾-inch openings, are separated from the coarse by this screen. Both products are now finished and drop into a two-compartment steel storage-bin 50 feet in diameter and 23 feet high.

2nd Floor.—The under-size, which passed through the 6-inch openings of the trommel on the upper floor, drops through hoppers into trommels (48 inches by 10 feet) on the second floor which are also inclined at 8° and revolved at 12 R.P.M. by 25 H.P. motors. The openings in this trommel are 2½ inches in diameter and the under-size goes direct to the floor below, to be sorted, while the over-size goes to the sorting belts where the sorted ore is carried over the end as a finished product and drops into the same steel storage-bins into which the ore from the upper floor was discharged.

1st Floor.—The under-size from the second floor is again screened in a trommel (48 inches in diameter and 12 ft. long) driven by a 10 H.P. motor at 11 R.P.M.; this, however, is a double screen with holes in the main section 1½ inches in diameter. The over-size goes to sorting belts and the sorted ore goes to the coarse-ore storage-bins, while the under-size is again sized by another screen outside of the main one. This outside section is 7 feet long, has ⅞-inch holes, and the over-size and under-size (which are the middlings and fines) are finished products and are stored separately. As this is a general description of the sequence of operations, attention may be drawn to a few points regarding design and efficiency.

As the crushing is done underground, the sizing done in the rock-house is to aid picking and to ensure proper sizes and the proper separation of these sizes for reverberatory and blast-furnace products and for building the roast-heaps. The products shipped are termed 'coarse,' 'middlings' and 'fines,' and the proportion of each is 55 per cent, 10 per cent, and 20 per cent, respectively, of the hoisted ore, the sorted rock making the remaining 15 per cent. The underground crusher is set at a 6-inch opening while the rock-house crushers are set at 3½ inches. The rock-house is made up of two units, either of which can be operated independently as the output demands. The efficiency in sorting is rated by tons of rock

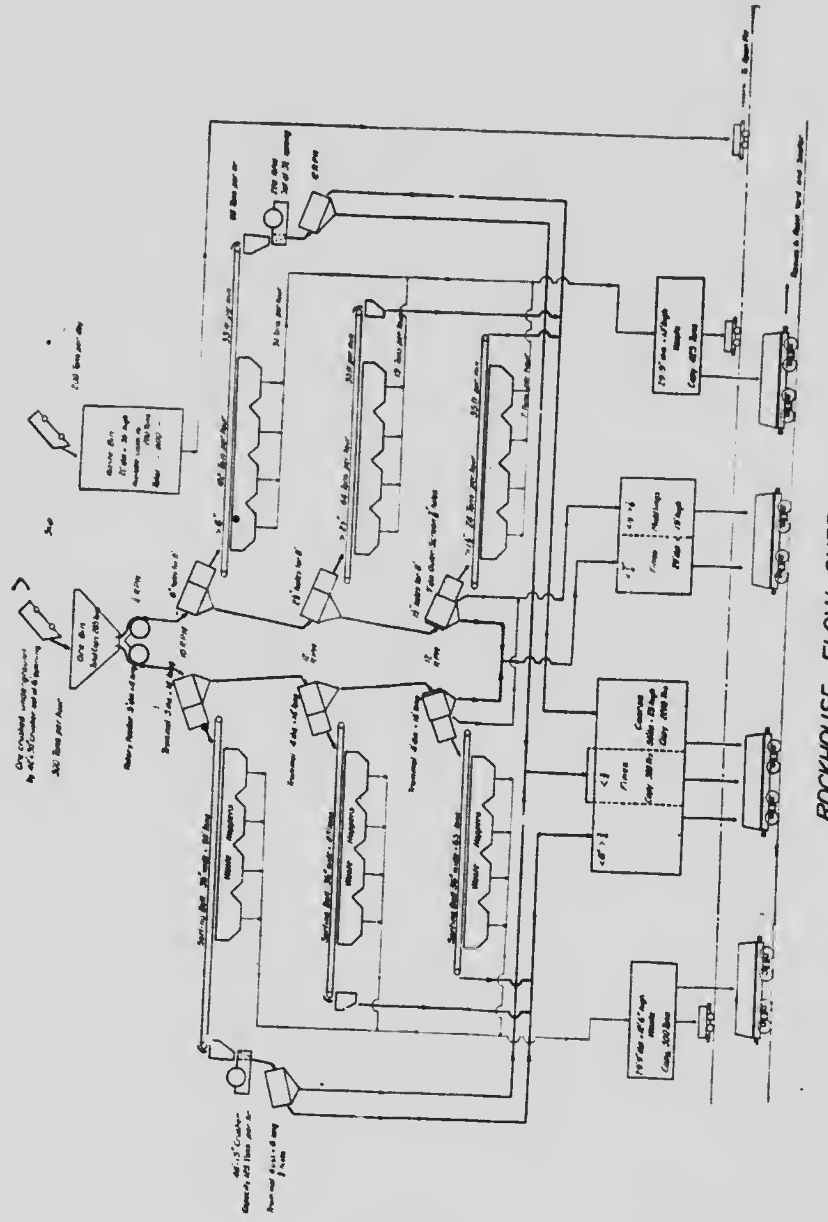


Figure 8.

picked per picker shift of eight hours and averages about 12.

All trommels are inclined at an angle of 8° from the horiozntal and the discharge chutes to the belts are at from 27° to 29°, depending upon the fineness of the product. To allow for the 'riding up' of the over-size, all trommels are set 14 inches off the centre-line of the conveyor belts onto which they feed. All conveyor belts are 36 inches, 5 by 7-ply Robin's, supported on 5-section rollers set at 4-foot centres; they have about 60 feet of picking length on each. These are driven at a speed of 35 feet per minute by 5 H.P. constant-speed motors through James speed-reducers having a ratio of 705 to 5.6.

The belts on the upper floor have each caried about 500,000 tons and are still in service. It might be added that the design of the rock-house has been so satisfactory in every detail that nothing has been changed in it since it was put into operation two and a half years ago. The flow sheet (Fig. 8) shows the layout of the rock-house.

Steel Sharpening.—As about 1,200 pieces of steel are handled daily in and out of the shop, convenience in working has been carefully planned. It will be seen by referring to the plan of the blacksmith shop (Fig. 9) that the tracks are laid through the building so that the dull steel is brought into one side of the building on trucks, and deposited on racks in front of the five heating-furnaces. The heated steel is carried by the furnace operator to his punch. He punches the hole and hands it to the sharpener to form the bit, gauge the steel, and deposit it in the waiting truck on the other side of the building. These trucks are conveyed back to the tempering and grinding room where the bit is tempered, shanks are ground and the finished steel is again placed on trucks ready to be sent undergound for storage on the various levels until needed. There are five sharpening machines in operation, and each has its own rack, furnace, and punch, thereby making a complete sharpening unit. Each sharpener handles only certain lengths of steel and thus minimizes the changing of dollies and gauging blocks. All of the steel is re-heated in a single furnace and tempered by plunging into a tank supplied with running water. The shanks are tempered in oil, the oil being cooled by pumping it through a multiple-tube inter-coolor and also by jacketing the oil tank in running water. Three Sullivan and two Leyner types of sharpeners are in use and a third Leyner is used when the amount of steel exceeds the capacity of the other five. A sharpener and his furnace-man average 250 pieces of steel for the eight-hour shift and two temper-men are ample for handling all the steel. At present, one eight-hour shift is suficient for 1,200 pieces of steel.

The crude-oil furnaces are of local design, being made of brick and steel; they have the usual needle valve, with ample air supply and compressed-air inlet. The ignited gas traverses the combustion chamber and, striking the opposite end of the furnace, is deflected up into the reverberatory chamber immediately above, and through the side of which the steel is thrust for heating. The waste gas passes out of the furnace of the top, and at the same end at which the oil entered. The amount of oil consumed is about one gallon for every 11 pieces of steel put through the shop; this includes the two heats per steel.

These furnaces have proven very satisfactory and the repairs only amount to re-bricking from time to time as some slag forms in the combustion chamber and the fire-bricks are eaten through. To a furnace running steadily eight hours per day, this happens about once in two or three months.

The wear on the bits, a cross-type with 14° and 5° taper, is heavy in the machines drilling in the granite; hence, in the shop, a great deal of dollying is required to form the bit. The aim of the shop is to produce the greatest amount of steel that can be well formed and tempered, and to make each operator responsible for certain steel put through.

General Blacksmithing.—Three forges are in use and are served by steam-hammer, power-shears and punch. Another small shop off the collar-house for skip repairs, etc., is in operation. The mine cars are repaired in a shop at the end of the collar-house.

Other Shops.—The situation of these important service departments can be seen by referring to the accompanying plan showing the general surface arrangement of the buildings (Fig. 10). The machine shop is well situated and has modern equipment in the way of lathes, radial-drill press, shaper and planer, bolt machine, pipe threader, etc.

The carpenter shop is small but is sufficiently well equipped with machines and tools for the proper handling of the work turned over to it.

Warehouse.—The materials and supplies at the mine are stored in a large and well laid-out building which is served by railway and narrow-gauge tracks at its door. Overhead crawls unload the heavier materials and the overhead track

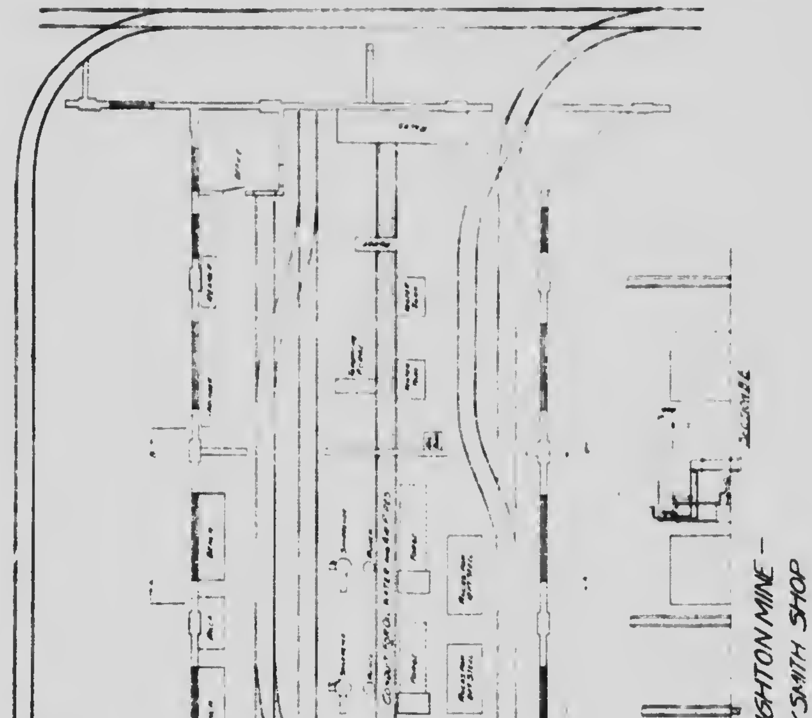


Figure 9.

extends into one section of the building and throughout its length.

Powder Magazine.—About 1,000 feet from No. 3 shaft, a tunnel has been driven into a hill and a room of sufficient size excavated to hold 2,500 cases of dynamite. Another room has been made off the tunnel to serve as a thawing-room. This is heated by hot water coils, the water being heated outside by electric heaters. A railway siding runs to the portal of the tunnel, and mine-gauge tracks are laid so that a truck of dynamite can be loaded in either the storage or thawing room and conveyed to the shaft. The tracks in the rooms are of hardwood, the doors of steel plate, the electric lights enclosed in air-tight glass globes, and recording thermometers give the daily temperature record. A powder-man, who keeps the stock records, is always in attendance.

The cap house is of cement and brick construction and is about 500 feet away from the powder magazine.

Air Compression.—Three air compressors placed in a building by themselves compress the air for use in the mine. These have a capacity of 5,000 cubic feet each; one is vertical and was manufactured by Bellis & Morcom, and the other two, which are exactly alike, are horizontal, being manufactured by the Ingersoll-Rand Company.

The one made by Bellis & Morcom is a 50-drill, electric-driven, two-crank, two-stage, vertical, enclosed self-lubricating compressor, direct-connected to an auto-synchronous motor, manufactured by the General Electric Company of Sweden, and rated at 900 K.W., 2,400 volts, 3-phase, 25-cycle, 187 R.P.M. This compressor is designed to compress approximately 5,000 cubic feet of free air per minute, at 975 feet above sea level, to a final pressure of 100 lb. per square inch.

The Ingersoll-Rand 50-drill compressors, are Ingersoll Roegler, class 'R.P.E.2,' duplex, electric-driven, horizontal, cross-compound, two-stage, direct-connected to a Westinghouse self-starting, sychronous motor rated at 932 H.P., 923 K.V.A. 2,400 volts, 3-phase, 25-cycle, 136.3 R.P.M. Each compressor is designed to compress approximately 5,000 cubic feet of free air per minute to a final pressure of 100 lb. per square inch.

The intakes to each of these are from wire-enclosed boxes standing about 10 feet above the ground. The diameter of the intake pipes is 24 inches and vents are left at joints of pipe-lengths to allow for the backing up of the stream of air when the valves automatically cut off. In the summer, the jacketing water is cooled by being sprayed into the air in a cooling

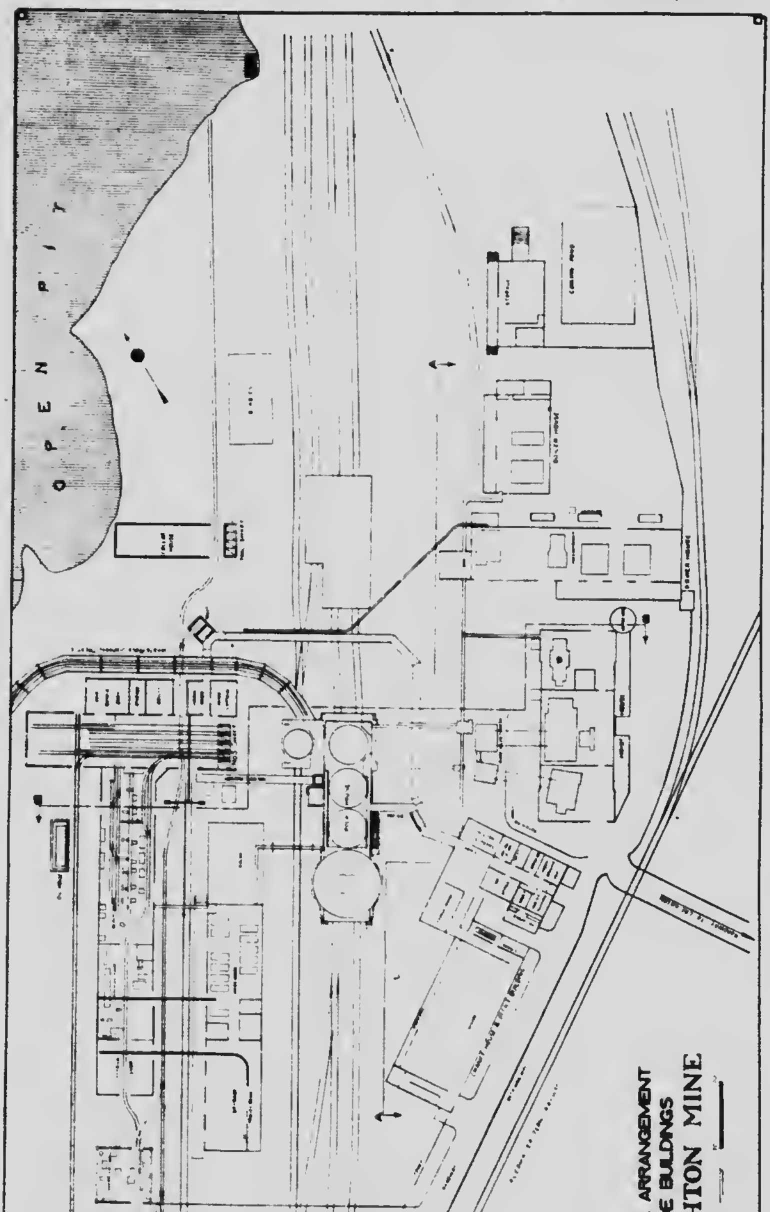

OPEN PIT
GENERAL ARRANGEMENT
SURFACE BUILDINGS
CREIGHTON MINE

Figure 10.

pond, 50 by 80 feet, situated a short distance from the compressor building. The compressed air is collected in receivers from where it is delivered into the 16-inch pipe which carries it underground with a loss of about 3 lb. in transmission.

Buildings, Etc.—The mine buildings and their general layout can be seen in Figure 10 (General Arrangement, Surface Buildings).

These are all of substantial concrete and brick construction and each has been carefully planned to fulfill all requirements as to space, light, warmth and fire protection. All frame work is of steel, all walls of brick, all floors of concrete, all roofs of good asbestos shingles and windows are of steel sash with wire-reinforced glass panes.

The two hoists for No. 3 shaft are in a building by themselves. Travelling electric cranes are installed in the hoist and compressor buildings.

The collar house at No. 3 shaft provides room for a drill repair shop, skip repair shop and a car repair shop, and is traversed by four tracks of the shaft gauge on which extra skips and man-cages stand ready for immediate transfer to the shaft when required.

The Change House.—This is so situated that the men coming to work come into it directly, and, having changed, go through the clockroom and thence through a covered passage leading over the railway tracks to a warm room at the collar of No. 3 shaft. The building is of two stories, there being on each floor 660 individual lockers arranged in 19 banks,

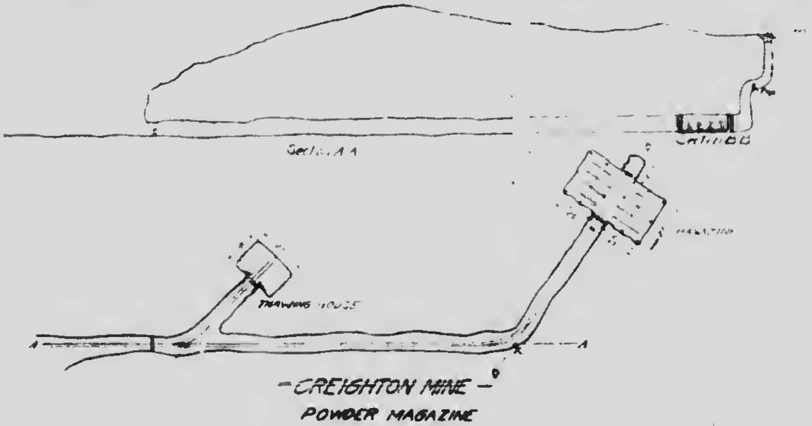


Figure 11.

together with 12 wash troughs and shower baths supplied with running hot and cold water. The lockers are of two compartments with screen doors and are connected to an exhaust fan that sucks out the foul air and draws in the warm air supplied by the hot-air pipes which heat the building.

Heating.—A central boiler plant supplies steam to steam coils in the various buildings and fans draw fresh air from out of doors through these coils, and send it through pipes leading to hot-air registers where needed. This heating has been very successful, but to cut down the cost of coal, etc.,

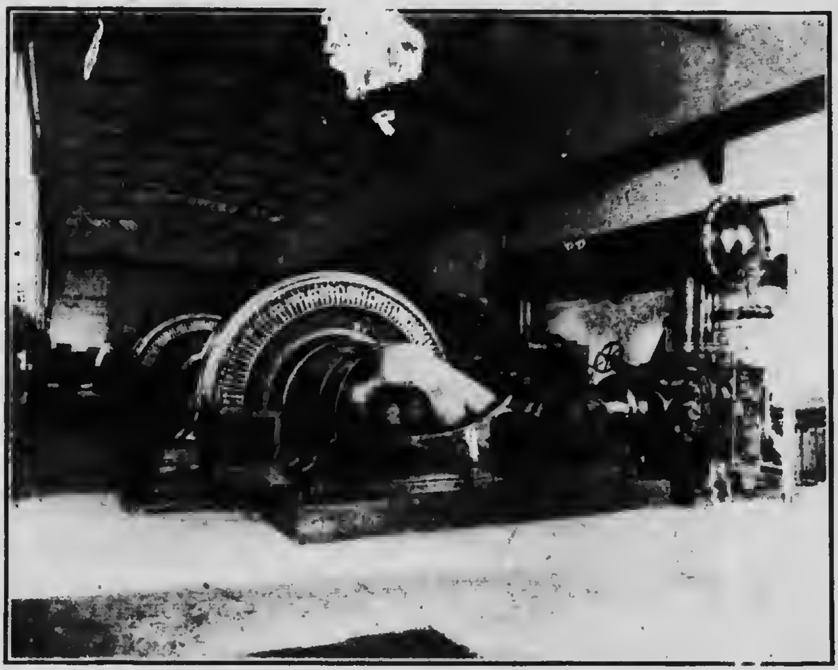


Photo by British & Colonial Press, Toronto.

Plate X.—Interior of compressor building.

electric grids have been substituted for the steam coils and the cold air drawn through them, thereby obviating the need of any coal. The water for the change house is heated by electric heaters inserted in long tanks connected with the water main. This method of heating the water has proved successful during over a year's trial.

Yard Arrangement.—In laying out the surface buildings, etc., plenty of room was provided for storing timber, machin-

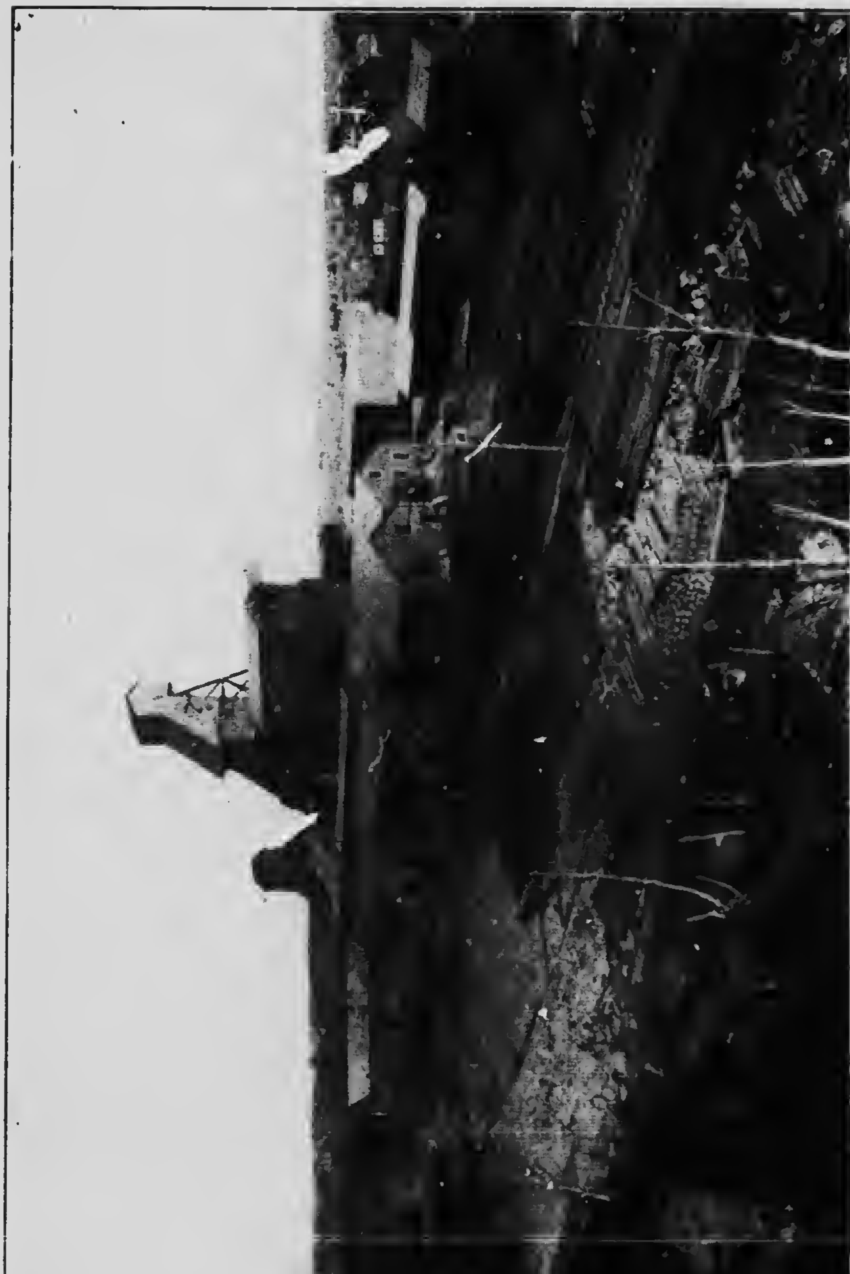


Photo by British & Colonial Press. Toronto.

Plate XI.—Surface plant and buildings, Creighton mine. Timber yard in foreground.

ery, etc., and this yard is well served by railway sidings and narrow-gauge tracks traversing each section. The narrow-gauge tracks are on zero grade; they have been well tamped to prevent movement by frost, etc., and have been laid as carefully as those underground.

The waste rock is conveniently disposed of in the open pit and the usual problem of how to get rid of it is thereby cheaply solved.

Smelting Methods.

The method of treating the ore as now practised, includes roasting, smelting in a blast furnace or reverberatory, and converting the matte in basic converters.

The ore as shipped from the Creighton mine, which is now the only operating mine of the company, consists of two main products, coarse and fines, in the proportion of about 60 per cent of the former and 40 per cent of the latter. The ore is crushed at the mine to pass a 6-inch ring, and the fines is the portion that passes through circular openings of 1½ inches in the trommels.

A typical analysis of the ore would be:—

Cu	1.50
Ni	4.00
Fe	41.50
S	24.00
SiO_2	17.00

On account of the high sulphur content of the ore, direct smelting is not practicable. The grade of matte produced would be only about 10 per cent Cu. Ni, which would throw an unduly large amount of work on the converters, and a large quantity of limestone or other basic flux would be required in the furnaces to make a slag that would flow at a reasonable temperature. From the above analysis it is seen that the iron in the ore offers a basic flux quite as effective as limestone, and at a much lower cost. To make it available, however, it must first be changed from the sulphide, in which form it exists in the ore, to the oxide, in which form it can combine with the silicious rock material that has to be fluxed. One method of doing this would be to oxidize it in the furnace by means of the blast, or in other words, to smelt the ore pyritically. Many and persistent attempts have been made to do this with these ores, but so far without success. On this account roasting in some form or other is the only alternative. Mechanical roasters require a finely ground feed, and a certain portion of the ore is treated in Wedge roasters

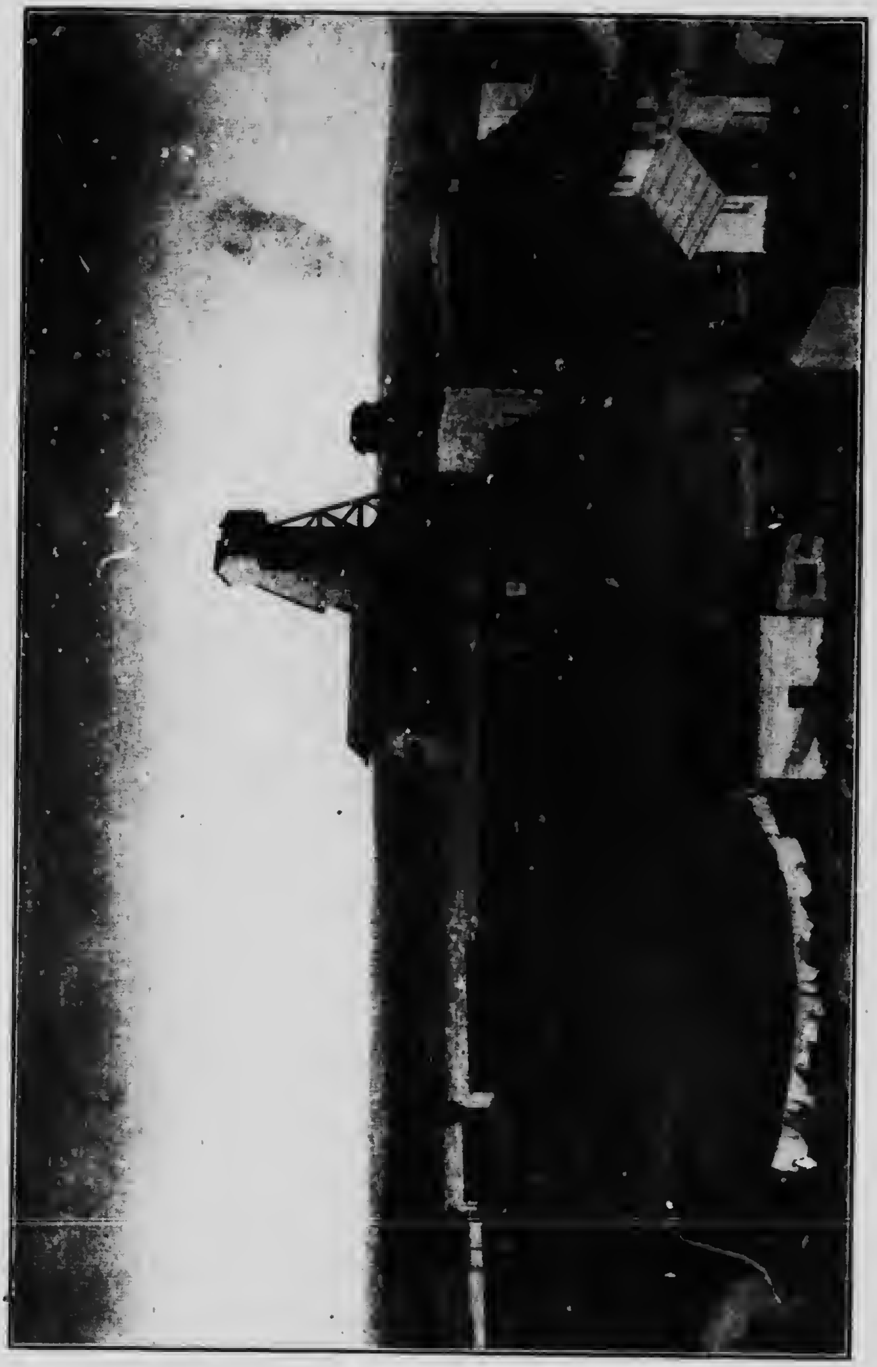


Photo by British and Colonial Press, Toronto.

Plate XIa.—Surface plant and building, Creighton mine.

and charged to a reverberatory furnace. The smelting equipment, however, consists principally of blast furnaces, which are not suitable for smelting fine material. In order, therefore, to provide a product suitable for use in the blast furnaces and at the same time one which will produce the proper grade of matte, it is necessary to roast the ore in the lump form. This is done in large heaps in the open, and is the practice that has been followed from the earliest days of the smelting industry in Copper Cliff. For many years the roasting was done in the immediate vicinity of the town, but as operations increased in magnitude , it was finally decided, in 1915, to move the beds to a pont where the minimum of damage would be done to vegetation.

The site chosen is in Graham township at mileage 16½ on the Algoma Eastern Railway, and is about nine miles in a direct line, almost due west from Copper Cliff, and about four miles west of the Creighton mine. By the railway, the distances are 13 and 5 miles respectively. Accommodation for the men employed at the roast yard has been provided by the company, and the village of O'Donnell has been built. There are a club house and several boarding houses for the unmarried men, and numerous well-built houses and cottages for those who are married. Special attention has been given to sanitation and the supply of drinking water. The latter is treated by one of the most up-to-date methods of chlorination (the Wallace Tiernan system) which renders the water perfectly safe without making it unpalatable. Its good quality is further ensured by taking frequent samples for bacterial examination in the laboratory at Copper Cliff An additional check is obtained by periodically sending samples to the laboratory of the Provincial Board of Health at Toronto. The same care, in fact, is taken with the drinking water at all points of the company's operation.

Roast Yard.

The plan of the roast yard is shown in Figure 12. The numerous running-tracks and storage-tracks assist in the rapid and economical handling of the ore. The ore is brought from Creighton by the Algoma Eastern Railway in 50-ton steel drop-bottom cars and placed on the receiving track at O'-Donnell. It is then taken by one of the company's locomotives to be weighed on the 100-ton standard railway scales, after which it goes to a transferring plant near-by, where the ore is dumped into a sunken hopper and thence fed by means of a short pan-conveyor under the hopper, to a 42-inch

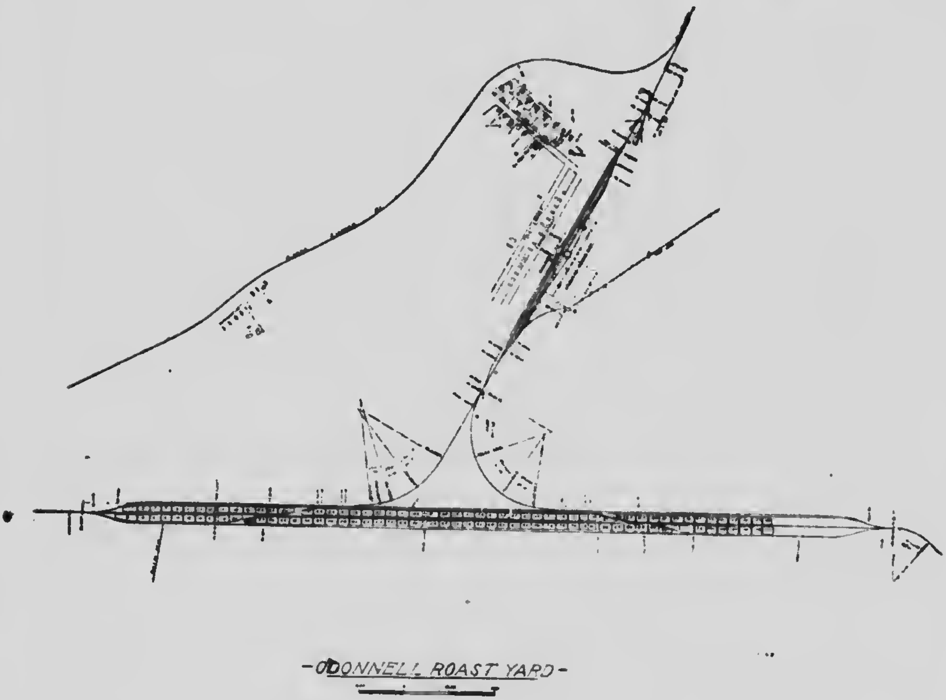


Figure 12.

Stephens-Adamson elevator which delivers the ore into specially designed 100-ton side-discharging steel cars. It requires about 10 minutes to fill one of these cars. As there are 12 cars in use, 1,200 tons of ore can be held in them at any one time. Some are kept loaded all the time in order to prevent any delay in the next step ,which. is the distributing of the ore on the beds.

The loaded transfer cars are taken to the roast yard proper and 'spotted' at the unloading bridge. The bridge has a clear span of 170 feet and travels on the inside rail of the standard gauge tracks that run along each side of the roast beds and are used as loading tracks when the roasted ore is picked up. As shown in the plan of the roast yard there are two parallel rows of roast beds with one track between them, one track along the south side and two along the north side. The last are 14 feet apart from centre to centre, and when the special transfer car is 'spotted' on the outer track on the north side (and opposite the bridge), it is in position to dis-

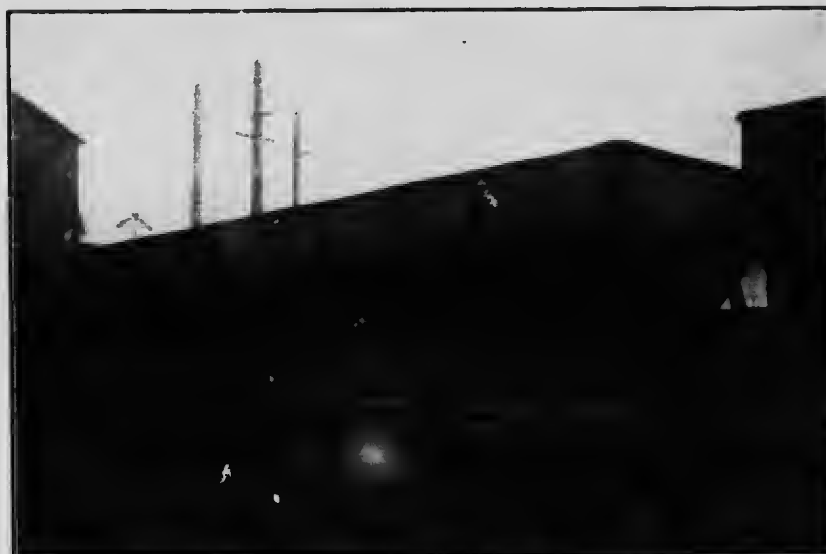


Plate XII.—100-ton transfer car.
Plate XIII.—Transfer plant. roast yard.

charge the ore into the receiving hopper of the bridge. In the transfer car there are 13 discharge doors, each of which is opened and closed independently of the others. The receiving hopper of the bridge can take the discharge from three doors simultaneously. Commencing at the end of the car, the practice is to open the two end doors and when the flow of ore slows up somewhat, the third is opened. By this time the ore opposite the first outlet is fully discharged, the door is closed and the bridge moved one door's width, when the process is repeated until the car is emptied, which takes from 13 to 17 minutes. One man operates the various controls on the bridge, and two others attend to the opening and closing of the doors on the transfer cars and to the guiding of the ore to the receiving hopper.

From the receiving hopper the ore is fed by a 60-inch pan-conveyor to a 36-inch pan-conveyor which elevates it to another hopper at the highest point of the bridge, from which it is discharged in a continuous stream on to a 30-inch shuttle conveyor belt. When building a particular bed the direction of travel of this belt is always the same, but the travel of the carriage on which the belt operates is automatically reversed at limits that can be set as desired. The belt travels towards the transfer car when the bed nearest the car is being built, and in the opposite direction when the one farthest away is being built. By the operation of the shuttle and the travel of the bridge itself, the beds are built uniformly over

Plate XIV.—Unloading bridge, roast yard.

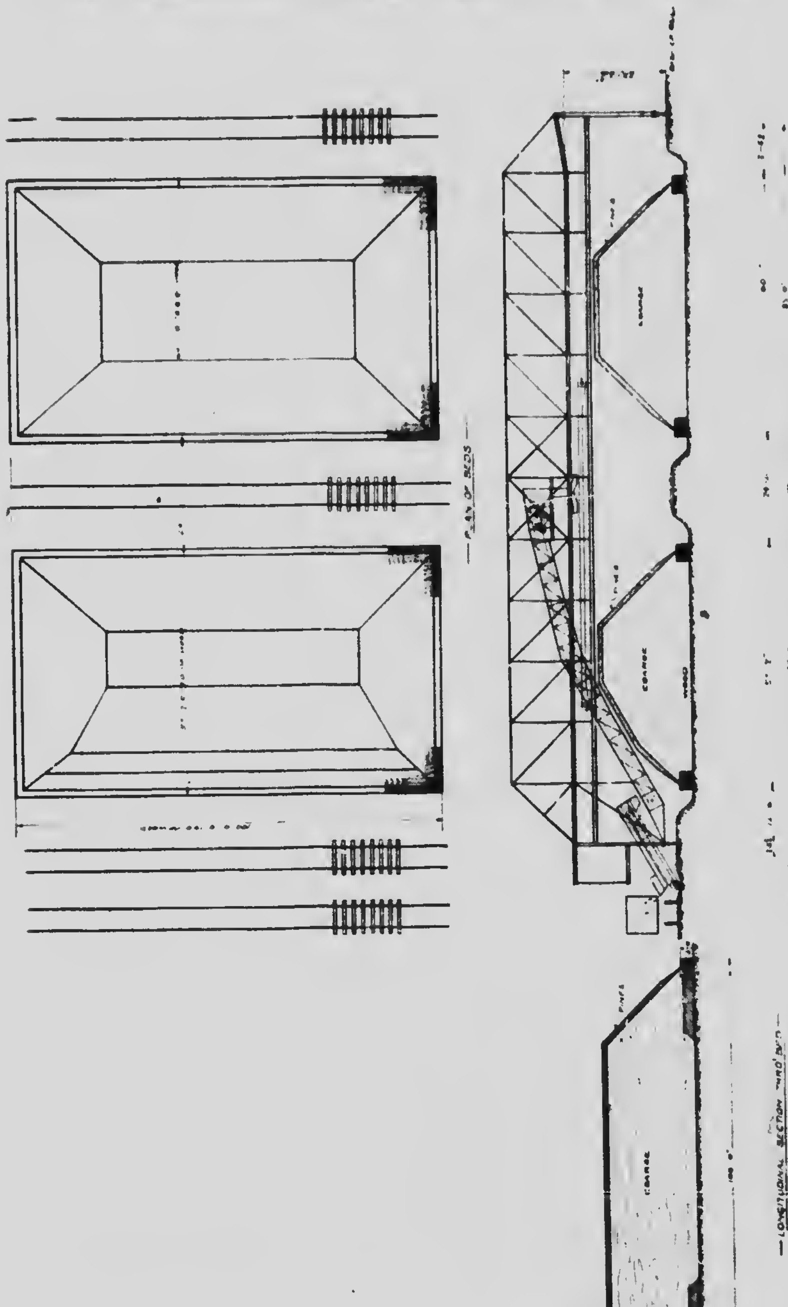

PLAN OF BEDS
CROSS SECTION THRO' BEDS
LONGITUDINAL SECTION THRO' BED

Figure 13.

their whole area, but if it is desired to build up a certain part more quickly than another, this can readily be done by stopping the conveyor carriage; the belt will then discharge the ore in the one spot as long as the bridge is stationary.

The ground on which the beds are built is about 4 feet below the level of the tracks. This difference enables the bridge to build larger beds than it could if the tracks and bed bottoms were on the same level. The only preparation required before beginning to build a bed is to lay the wood necessary to start the roasting. The wood for each bed covers a rectangular area about 100 feet long and 60 feet wide. The width is determined by the distance between the tracks, but the length is largely a matter of convenience in building and lighting. The wood used is the usual 4-foot cordwood, and it should be of good quality. The finished bed contains about 5,000 tons of ore.

The beds are lighted as soon as possible after the building is completed. As the wood burns away, the ore settles down and cracks develop in the surface of the bed. This is the critical time of the process, and bed trimmers are constantly on the alert, during the first week or two of the burning, to

Plate XV.—Steam shovel loading roast ore.

close these cracks as quickly as possible. After the subsidence is completed no more openings are likely to form, and the bed requires practically no further attention. It will probably burn for six or seven months, by which time the sulphur will be reduced to about 10 per cent.

Plate XVI.—Unloading green ore by hand.
Plate XVII.—View of roast beds built by hand.

The roasted ore is re-claimed from the beds by two Atlantic steam shovels having a dipper capacity of 2½ cubic yards. They load the roast ore into the same 50-ton cars that bring the green ore from the mine.

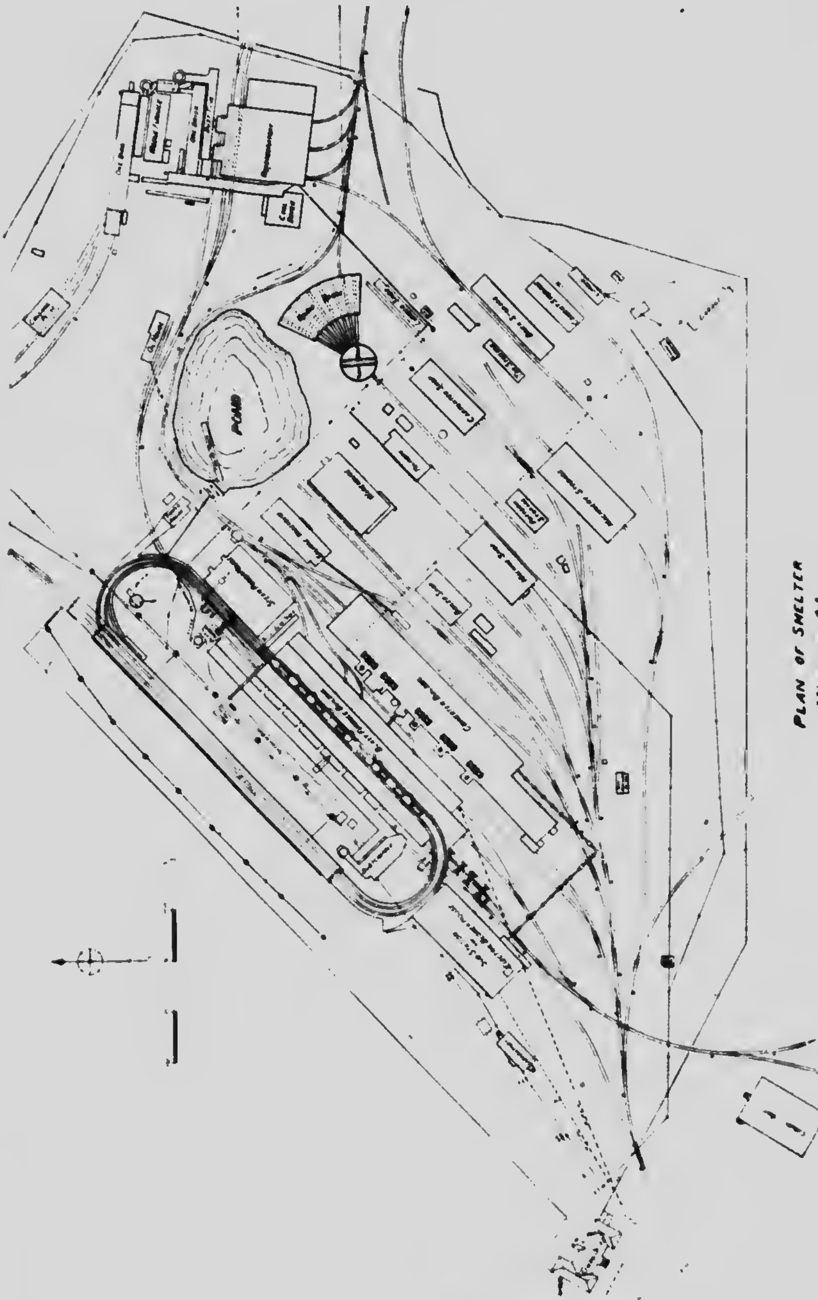
The bridge was put in operation early in 1919. Previously, the ore was unloaded from flat-cars by men with wheelbarrows (Plate XVI) and the beds could not be made so high as at present because the men would not wheel the ore up much higher than the floor of the car on which they were working. An average bed then contained about 2,500 tons, and burned out in three or four months. The larger bed, which it is now possible to build, has several advantages over the smaller one, the chief being that (1) more ore can be stocked in the same space, (2) the ore roasts better, and (3) less wood is required. The same quantity of wood had to be used to start the roasting of the low beds as is required for the higher, and therefore larger, beds. The better roasting is due partly to the fact that the large beds roast more uniformly and for a longer time, but also because they have a smaller percentage by weight of the outer part of the bed that is roasted very little in either case.

For a number of years before the bridge was built, various mechanical methods of handling the green ore had been under consideration, but while labour was plentiful and cheap they did not appear attractive, and it was mainly the shortage of labour during the war that finally caused the decision to be made to adopt the present method as the best of a number investigated. It has proved even more satisfactory than was anticipated, and after some alterations in the elevators which were considered advisable after operating for some time, and which were made in 1919, it readily handles the required tonnages. The operating costs, also, have been quite as low as anticipated.

Blast Furnaces.

There are eight furnaces in this department. Five of them are 17 feet in length, one is 21¼ feet and two are 25½ feet, giving a total furnace length of 157¼ feet. All have the same width at the tuyeres, namely, 50 inches, and are similar in construction throughout except for such differences as are due to the different lengths.

The cast-iron hearth-plates are supported by 12-inch I-beams laid transversely to the furnace length. No cooling is provided for these plates beyond that due to the air naturally circulating under them. The crucible of the furnace is built



PLAN OF SMELTER
Figure 14.

of chrome bricks, which, at the sides, come up to within 6 inches of the tuyeres and are stepped down to the centre in the form of a 'V', the minimum thickness of brick being 18 inches.

The sides of the furnace are formed of sections 4 feet 3 inches wide, containing six jackets for the full height of the furnace. First there is a pair of cast-iron tuyere-jackets (in which are embedded 1¼-inch water pipes), each 2 feet 1¼ inches wide and 4 feet 7 inches high, resting on the hearth plates. Each of the pair has, near the top, two tuyere openings 6 inches in diameter. Above the tuyere jackets is another pair of cast-iron jackets of the same width and of similar construction, but only 3 feet 11 inches in height. Above these is a steel water-jacket 3 feet 6 inches high and 4 feet 3 inches wide, and finally another steel jacket 6 feet high and 4 feet 3 inches wide. The use of the small steel jacket was made necessary when the height of the furnace was increased several years ago. The cast iron jackets are given a slope outward to form the bosh of the furnace, the maximum inside width being 5 feet 10 inches, an increase from 2 feet 10 inches at the hearth plates. The steel jackets are vertical. The 17-foot furnace requires four of these sections to the side, the 21¼-foot furnace, five, and the 25½-foot furnace, six. At the dead end of the furnace the lowest jacket is of cast iron wth water pipes imbedded in the same way as with the cast-iron side-jackets. It is 3 feet 9 inches high and above it are three steel water-jackets, 4 feet 9 inches, 3 feet 6 inches and 6 feet high, respectively. At the front end the lowest jacket is of copper 32 inches wide and 5 feet long, with the lower edge 1 foot 11 inches above the hearth plates. This forms the trap of the furnace. The top of the copper jacket is the same height above the hearth plate as the top of the cast-iron jacket at the dead end, and hence the remainder of the jackets at the front are the same as those at the dead end. Each furnace is provided with a small side tap jacket fitted into a notched tuyere-jacket and placed near the middle of the furnace.

The furnace spout is of chrome brick built against the copper jacket. It is carried on a 4-inch cast iron plate about 4 feet wide and 5 feet long, which rests on the bottom plate of the furnace and the side of the settler. Cast-iron water-cooled side-plates retain and protect the brick. A water-cooled cast-iron lip is placed where the mixed matte and slag flow from the spout. Further protection to the brick is given by another cast iron cooler placed under the lip. The effective

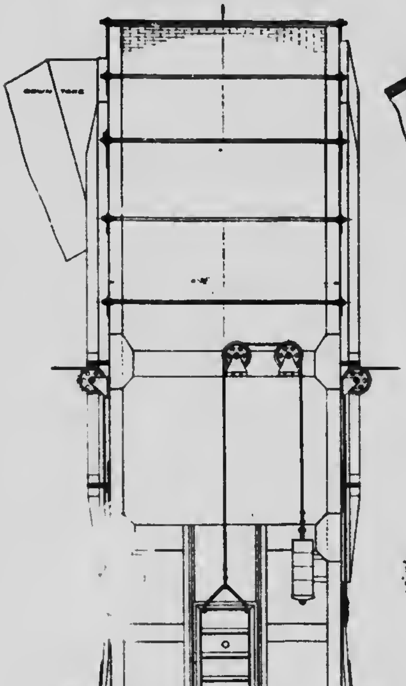


Figure 15.—End elevation blast furnace No. 6.

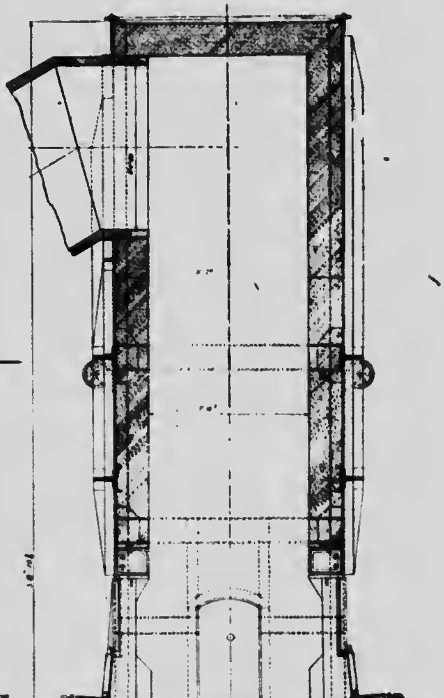


Figure 16.—Cross-section, blast furnace No. 6.

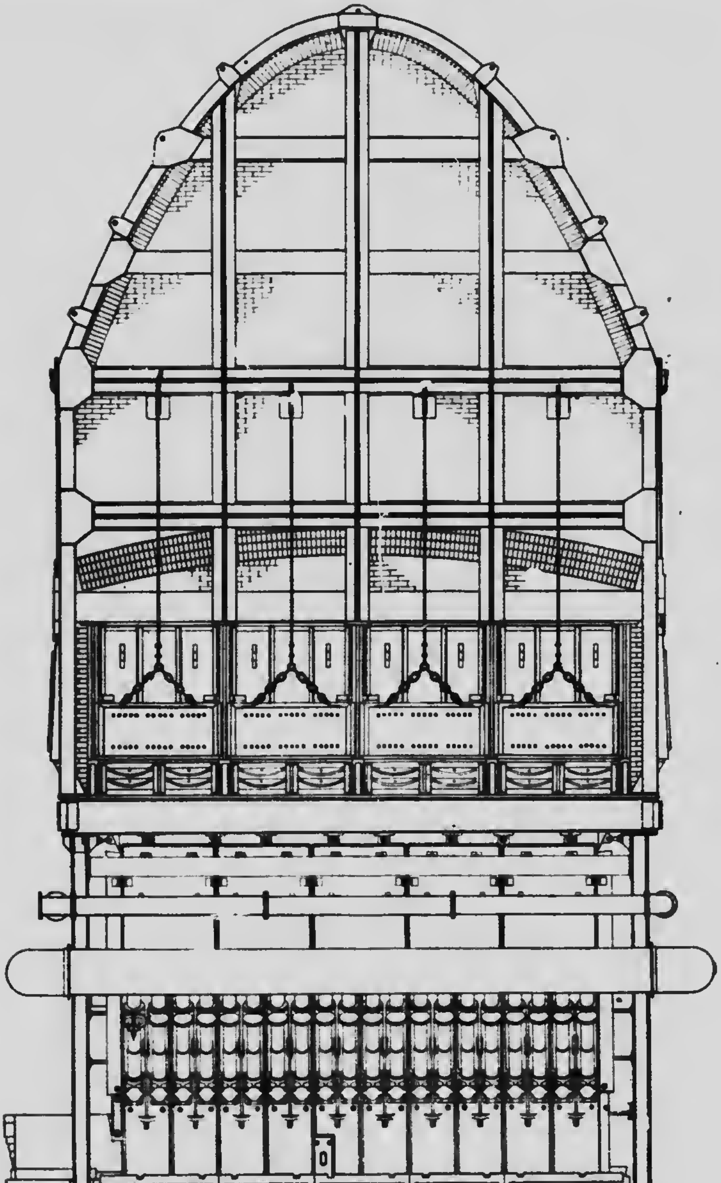
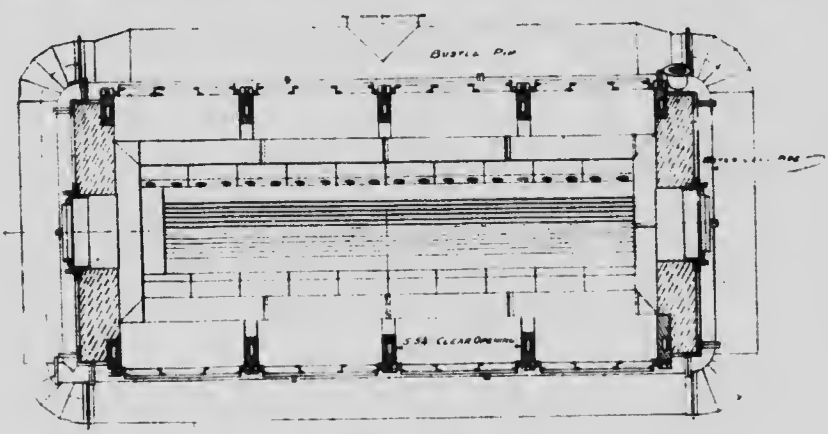


Figure 17.—Side elevation, blast furnace No. 6.

depth of the trap formed by the spout and the copper jacket is about 9 inches. This type of spout was developed at the plant to overcome trouble with corrosive low-grade copper-nickel mattes, and has proved very satisfactory. A cut-out of the spout is practically unknown.

The furnace tops are of fire brick stiffened with steel work. They rise about 33 feet higher than the charging floor. The upper 15 feet is in the form of a catenary arch sprung over the length of the furnace. In the smaller furnaces an 8-foot downtake connection is just under the centre of the arch. In the larger furnaces the diameter is correspondingly greater. The downtakes are made of steel plate, the first 20 feet being lined with fire brick. They are about 64 feet long and run down at an angle of about 30° with the horizontal to a 20-foot balloon flue, at the rear of the furnaces. This flue is of steel and is about 530 feet long. It is provided with clean-out chutes, which discharge the collected dust into a standard-gauge steel-car. Approximately 2 per cent of the weight of the charge is recovered in this way. Additional dust is also caught in a wire-hung dust collector placed between the balloon flue and one of the two stacks that serve the furnaces. Each of these stacks is 210 feet high (above the yard level), with an inside diameter of 15 feet at the top. They are connected to opposite ends of the balloon flue.

The water for use in the jackets is obtained from a small lake near the town (Copper Cliff). As the supply is limited,



48"×255" BLAST FURNACE-SECTIONAL PLAN

Figure 18.

Plate XVIII.—Balloon flue and stack, blast furnaces.

it is used over again after being collected in a cooling pond at the smelter. It is treated with alkali to neutralize any acid present.

The settlers are 5 feet 6 inches high and the shell is made of ½-inch boiler plate. It is held in position by lugs and bolts sunk in the concrete base. The dimension on the longer axis is about 20 feet inside the lining. The lining consists of chrome brick about 12 inches thick at the bottom and 18 inches thick in the walls. At the point where the stream from the furnace falls into the settler, and also around the tap-holes, extra protection is given by the use of cast-iron cooling-plates. The tap-hole is formed by two chrome blocks 14 inches square and each 6 inches thick. The hole through the blocks is 2 inches in dameter. There are two tap-holes in each settler, and the pouring lip for converter slag is placed between them. The settlers have the shape of a distorted

Plate XIX.—Charging floor, blast furnace building.

oval, and are placed at the ends of the furnaces with the longer axis at right angles to the longitudinal axis of the furnace.

All material required to make up the charge for the blast furnaces is delivered to a system of bins which are parallel to, and about 200 feet from, the furnace-building proper. The ground on which the bins stand is at the same level as the charging floor of the furnaces, and the top of the bins is 35 feet higher. Along the top of the bins run two parallel standard-gauge tracks and the whole is covered with a train shed. The sides of the bins are also completely housed in except for the lower 6 feet on the side nearer the furnaces. The length of the building is 600 feet and the width about 30 feet. Heavy timber construction is used throughout.

There are 50 separate sections or compartments, each having a capacity of about 6,000 cubic feet. Twenty-four of these are generally reserved for coke, and 12 or 13 for the roasted ore, giving storage room for about 2,000 tons of the former and 4,000 tons of the latter, sufficient to keep the plant in operation through any ordinary tie-up in transportation. The remaining bins contain converter slag, scrap, green ore, quartz and limestone, and one is also used to hold coal for the heating boilers and smelter locomotives. As far as possible all material is brought to the storage bins in bottom-dump or side-dump cars, in order to save both time and labour in the unloading. Grizzlies are placed over all except the coke bins to prevent large lumps of ore or 'revert' falling in and possibly blocking the chutes below when the material is withdrawn.

Underneath the bins are two narrow-gauge tracks on which are operated small charging trains drawn by electric locomotives. The tracks continue around in the form of an oval past each side of the furnaces, and back again under the bins, travelling always in the one direction. The trains consist of nine side-dump cars with a capacity of 25 cubic feet each, and are filled from chutes under the bins with the materials desired to make up the charge. The usual charge consists of three cars of coke, one car of 'revert,' one car of green ore and four cars of roast ore. Sometimes two cars of green ore are used, in which case a car of roast ore is dropped. The cars containing the charge are weighed on a scale at the end of the bins, and the coke adjusted to the exact percentage required at the time. This is usually between 10 and 11 per cent of the weight of the charge, depending on the quality of the coke and the nature of the charge. A typical charge would have about the following composition by weight:

Roast ore	12,000 lb.
Green ore	5,000 lb.
'Revert'	2,500 lb.
Total charge	19,500 lb.
Coke	2,000 lb.

When using well roasted ore, a charge of this kind will give the grade of matte desired for the converters.

As a rule no flux is required, though both limestone and quartz are available if conditions demand their use. For the whole month of January, 1920, only 15 tons of quartz and four tons of limestone were used.

When the charge train is brought around to the furnace, the doors of the furnace are raised with the assistance of counter weights and the charge dumped in. The coke is put in first, and everything is spread as evenly as possible by moving the train backward or forward as the cars are discharging their contents. Charging is done at intervals of from 20 to 30 minutes.

Plate XX.—Furnace blowers.

The blast for the combustion of the coke and the oxidation of the sulphur is furnished by four Connersville blowers, three with a capacity of 44,800 cubic feet per minute, and one of 33,000 cubic feet. The latter can be operated at three different speeds, furnishing 33,000 cubic feet, 23,800 cubic feet, or 16,800 cubic feet as required. There is also a Nordberg blower of 24,000 cubic foot capacity, which is now seldom used, but is available as a spare. It also is a three-speed machine giving 16,000 or 11,000 cubic feet at the lower speeds. All blowers discharge into a common pipe, the supply to the furnaces being controlled by a gate at each furnace between the common pipe and the bustle pipe. The pressure in the bustle pipe is maintaned at about 25 oz. The volume delivered to the furnaces, measured by rated blower capacity, is about 1,050 cubic feet per minute per linear foot of furnace. A larger volume will give a faster running furnace for a time, but blow holes soon develop and the tonnage decreases. There is also additional work barring down accretions and the attendant disadvantages of erratic operations.

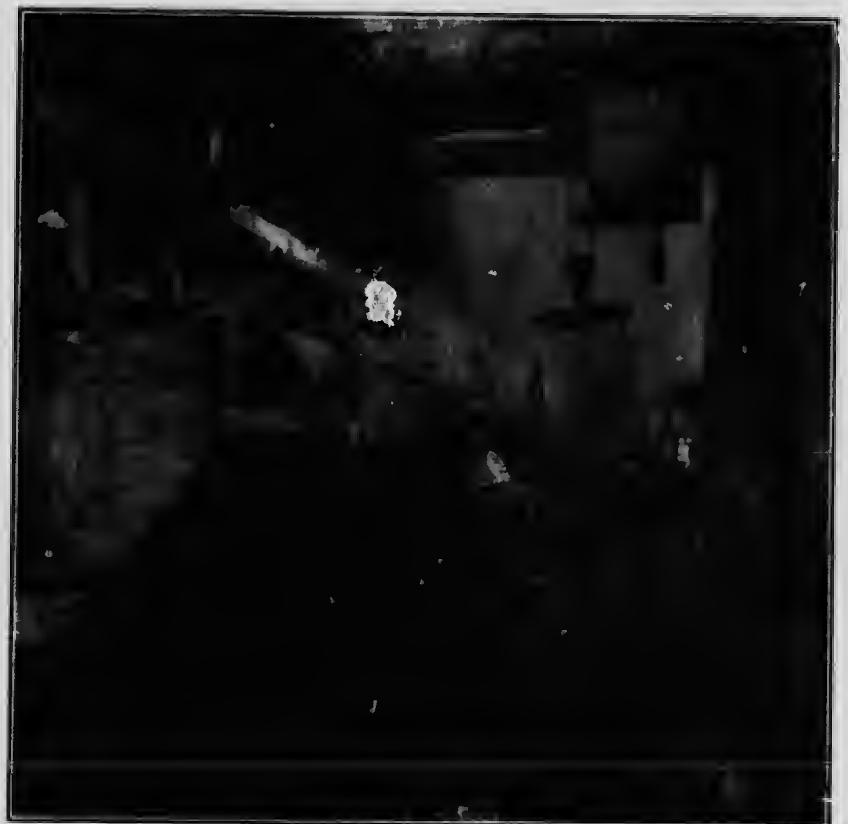
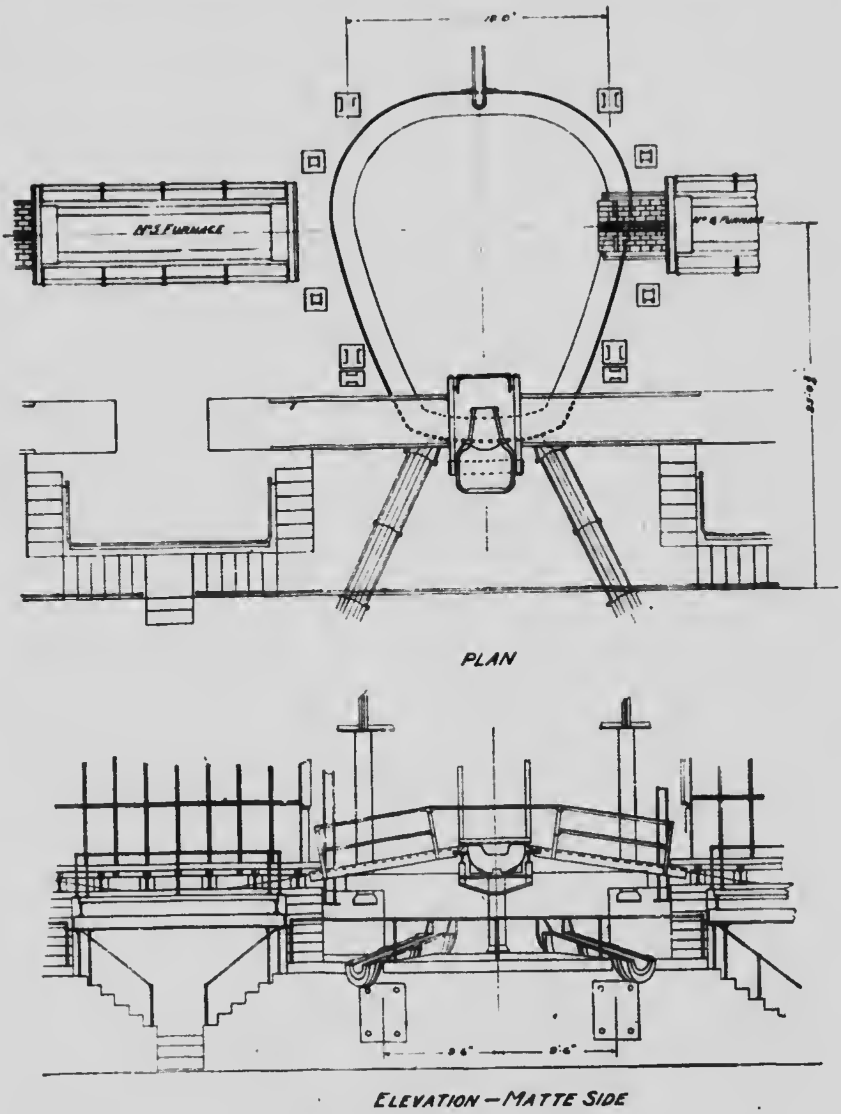


Photo by British and Colonial Press, Toronto.
Plate XXI.—Tapping floor, blast furnace building.



BLAST FURNACE SETTLER

Figure 19.

The molten slag and matte from the furnace flow into the settler close to the side and nearer the back of the settler where the slag overflows than the front where the matte is tapped. This is far from being an ideal arrangement, but was the best that could be done in the existing building (without incurring excessive expenditure) when the settlers were enlarged to provide for pouring converter slag into them. A greater distance of travel for the furnace slag before overflowing would be a decided improvement However, for the converter slag, the maximum distance of travel is provided.

The slag from the settlers overflows into pots of 225-cubic foot capacity, standing on standard-gauge tracks. When full, they are drawn away to the slag dump where the liquid slag is poured. The turning down of the pots is done by an electric motor. Before being placed under the slag stream, the pots are given a lime wash to aid in the removal of the 'skull' at the dump.

Before adopting the practice of pouring the converter slag into the settlers, a long series of experiments was undertaken. Two settlers, connected to furnaces smelting similar charges, but one having converter slag poured into it, were sampled independently over a period of nearly two years. Not only were the usual ladle samples taken at the settlers, but channel samples of the dump were also taken. This duplicate sampling gave added confidence in the results obtained.

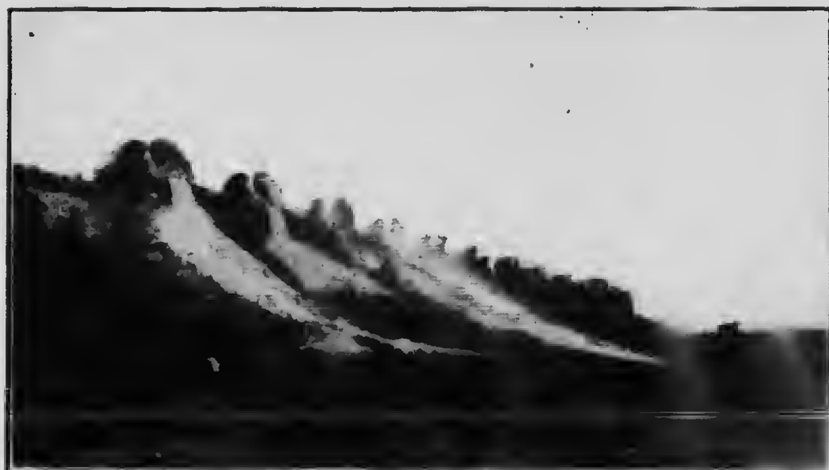


Photo by British and Colonial Press, T[illegible]

Plate XXII.—Slag cars at dump.

Over a period of 21 months the average of the dump samples showed only a difference of 0.008 per cent Cu.Ni from the ladle samples.

In cases where the converter slag is required as a basic flux in the furnaces, the saving may not be so pronounced as in this plant, where the tendency is to the use of a silicous flux rather than a basic one. The amount of slag poured into a settler is approximately equal in volume to the matte tapped from it.

An attempt is made to pour in the slag just after tapping. It is also poured as slowly as possible. A too rapid overflow into the furnace slag pot is prevented by damming the slag chute for a time.

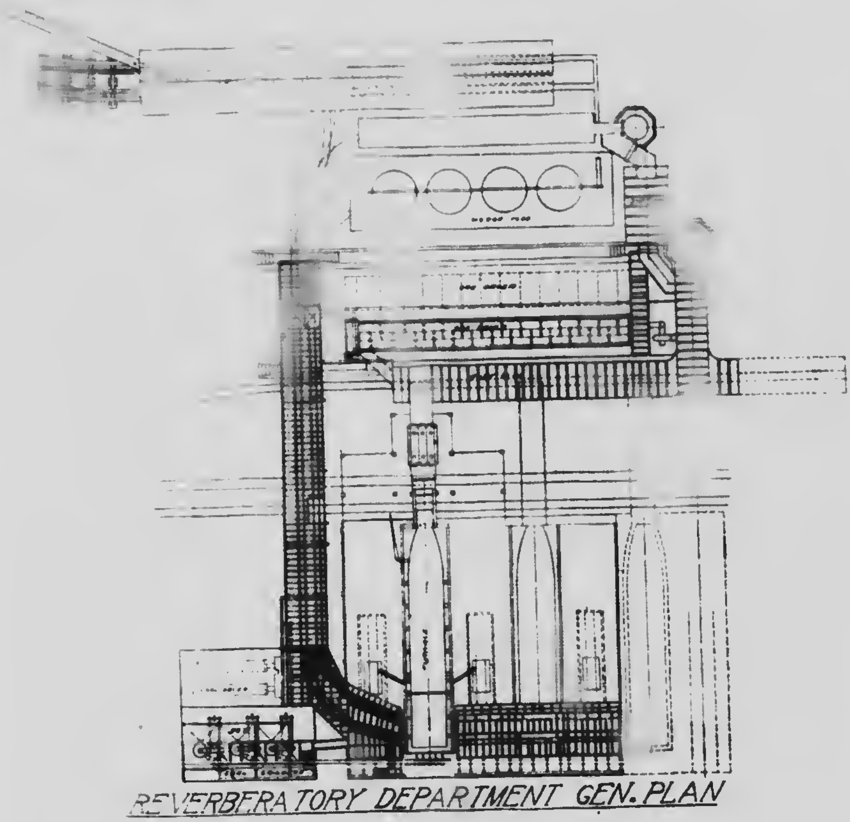


Figure 20.

Furnace matte is tapped into sectional cast iron ladles which hold about 8 tons. The ladles are given a protecting lining of converter slag before being used for matte. The matte is transferred to the converter building over a short narrow-gauge track provided with a 35-ton scale.

From the standpoint of both converters and blast furnaces, the most economical grade of matte to make is found to be from 25 to 27 per cent copper-nickel, but as there is now considerable excess converter capacity, advantage is being taken of this fact to use up some green ore that is on hand, and at present the matte is somewhat lower.

Reverberatory Furnaces.

The reverberatory plant was first put in operation about the end of 1911. The complete plant as now operated includes a system of steel storage-bins, a ball-mill department, a Wedge furnace roaster plant, a pulverized-coal department and the reverberatory furnace itself, all housed in separate buildings.

The principal part of the charge to the reverberatory is fine ore from the mine. This ore has passed through a screen having 1½-inch circular openings, and about 50 per cent of it is less than ¼-inch size. As the ore has no tendency to decrepitate on heating, it is necessary to crush it still smaller before it will roast satisfactorily, and for this purpose four No. 8 Krupp ball mills are used. Standard-gauge tracks, at the same level as those that serve the blast-furnace bins, bring the ore to the ball-mill plant, and coal and other material to the reverberatory storage bins. About 1,000 tons of coal and 3,500 tons of ore, besides smaller amounts of flue dust, fettling and flux can be stocked at the bins.

The ball mills are driven by 60 H.P. motors and rotate at a speed of 21 R.P.M. They are ftted with manganese-steel grinding-plates and carry a load of 4,000 lb. of balls five inches in diameter. The consumption of balls is about 0.5 lb. per ton of material crushed. Ore is fed into the mills by a 24-inch link-belt pan-conveyor, intermittently operated by an adjustable rachet and pawl device. Before escaping from the mill, all ore must pass through five-mesh steel screening, which gives a product of which 65 per cent will pass a 20-mesh screen, and 25 per cent a 100-mesh screen. When operating in this way, each mill will grind about 200 tons per 24 hours. Three mills, operated on two eight-hour shifts, are at present producing the necessary tonnage. Finer material is sometimes required and is secured by replacing the five-mesh steel screens with ten-mesh bronze screens. In this size, steel screens would clog badly and would also have a much shorter life than the bronze. The discharge from the mills is

Plate XXIII.—Coal burners at reverberatory plant.
Plate XXIV.—Top floor, Wedge roasting furnaces.

taken, by a system of 16-inch conveyor belts, to a section of the storage bins holding about 3,500 tons.

The roaster building, which is parallel to, and about 25 feet from, the storage bins, contains four Wedge furnaces, 22 feet 6 inches in diameter, equipped with seven hearths besides a top or drying hearth. The central shaft carries four arms on the seventh floor and on the drying hearth and two on each of the others. The arms are cooled by air supplied from two fans, one of which is usually found to be sufficient. The air pressure on the discharge side of the fan is equivalent to about three and a half inches of water. The air used for cooling the arms also supplies the oxygen necessary for the combustion of the sulphur and the oxidation of the iron. The furnaces are driven from a line shaft operated by a 35 H.P. motor and rotate about once in three minutes.

The fine ore in the storage bins is taken, by means of conveyors and an elevator, to bins above the Wedge furnaces from which it is fed automatically to the drying hearths, where practically all the moisture is driven off. When entering the furnace the ore contains about 24 per cent sulphur, which is reduced to 10 or 11 per cent when discharged. Lower sulphur can be obtained if desired, but at the expense of tonnage. Each furnace has a capacity of about 115 tons of charge per day under the conditions at which they are now being operated. The heat of the furnaces is easily maintained by the roasting reactions, the temperature on the fourth, or hottest floor, reaching about 750° C. under normal operating conditions. Should a furnace become cooled, as when changing arms or through some other delay, it may be necessary to throw in a few shovelfuls of coal to assist in bringing up the heat again, and after a shut-down the heating up is done with oil.

Before escaping to the stack the gases pass first through a steel balloon-flue (with a narrow-gauge track below it for removing any collected dust) and then through a wire-hung dust chamber similar to the one at the blast furnaces. In this way about 250 tons of dust is recovered each month. The gases from the roaster plant usually pass up the same stack that serves the reverberatory furnace but may be diverted to a steel stack serving the roasters only.

The hot calcines from the Wedge furnaces are taken away in 3½-ton bottom-discharge cars and dumped into bins at the back of the reverberatory furnace, first passing over a scale

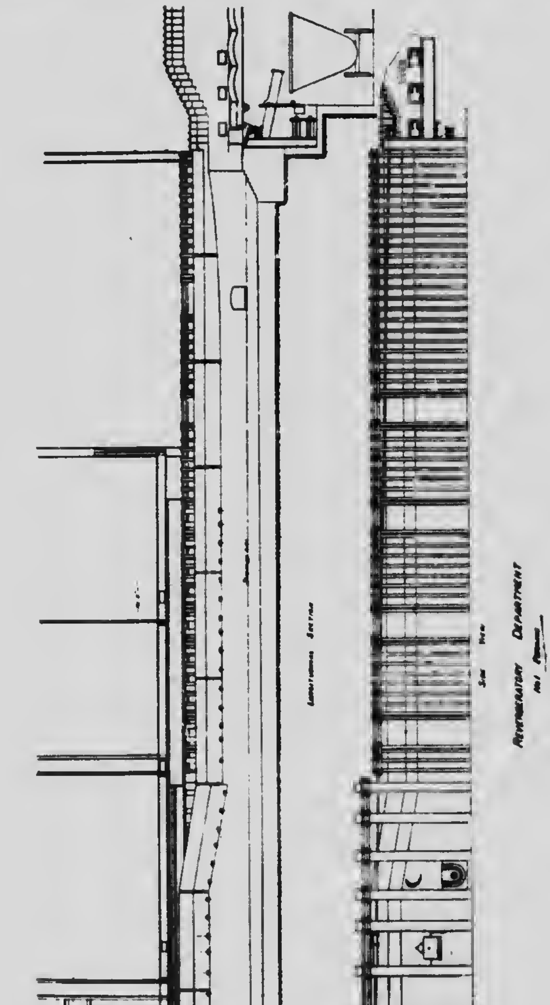


Figure 21.

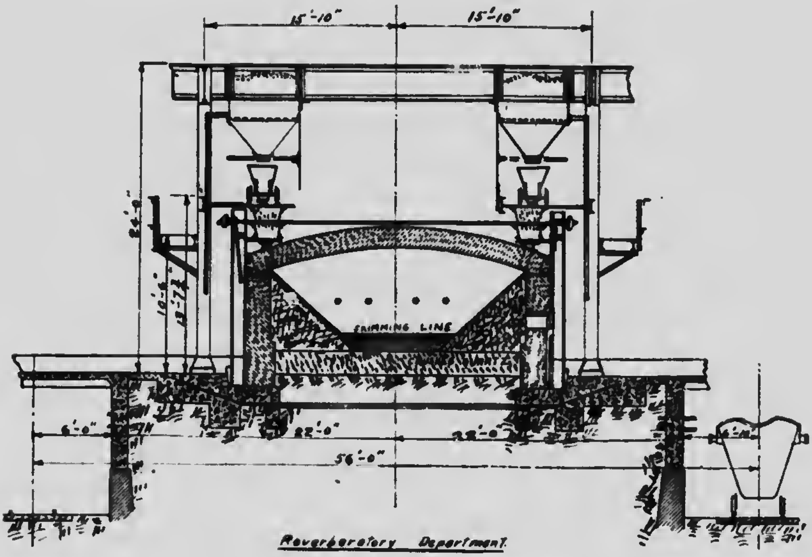


Figure 22.—Cross-section of reverberatory furnace.

to be weighed. Other ingredients of the charge are drawn from the storage bins in a similar manner. The haulage is done by six-ton electric locomotives similar to those used at the blast furnaces. From the bins at the end of the reverberatory the calcines, etc., are drawn into small bottom-dump cars which operate on tracks above the side walls of the furnace. Just below each track is a trough, extending the length of the furnace and kept filled by means of the small cars. Six-inch pipes, spaced two feet apart, extend from the bottom of the trough to holes in the roof of the reverberatory near the side walls, by which the charge enters the furnace. As the trough is always kept filled, the charge piles up against the inside walls of the furnace and reaches up to the roof. It sinks down only as smelting takes place and is immediately replaced by fresh material from above. In this way the charging is made absolutely continuous, the side walls are always protected and the same area is constantly exposed to the action of the flame.

The reverberatory furnace itself is 112 feet long by 19 feet wide and has a cross-sectional area at the throat of 48 square feet. For the first 30 feet from the bridge wall the roof is two and a half feet higher than at the flue end and a 12-

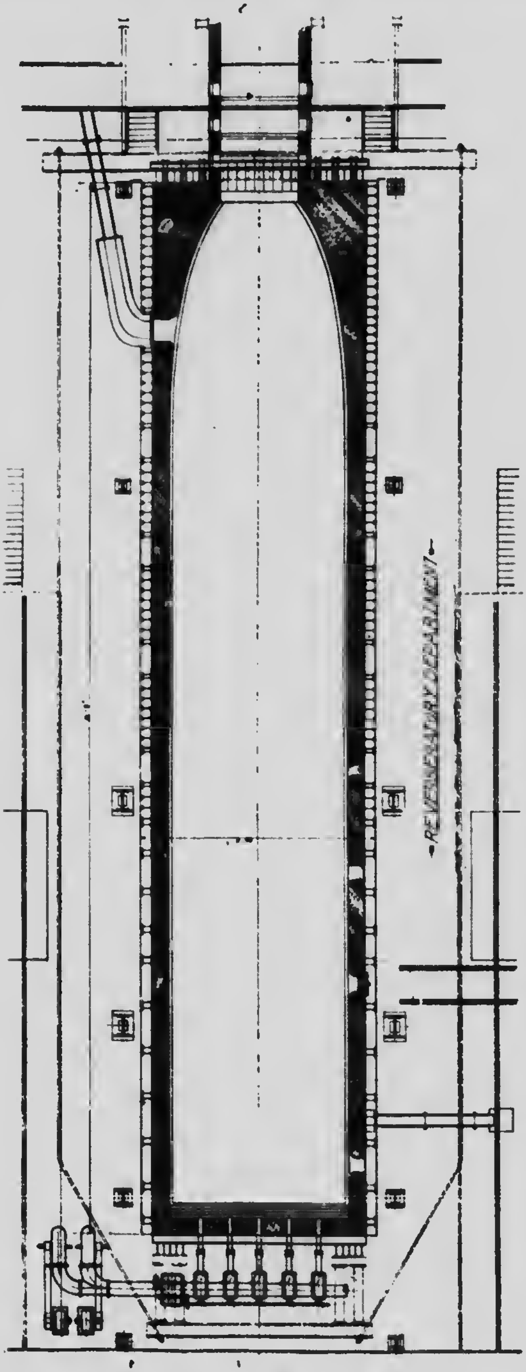


Figure 23.—Sectional plan of No. 1 reverberatory furnace.

foot inclined section connects the two levels. The roof is built of silica brick, the first 42 feet being 20 inches thick and the remainder 15 inches. The side walls are also of silica brick except at the tap-hole and skimming bay where some chrome and magnesite bricks are used and at the bridge wall where fire brick is used. There were originally a number of doors along each side of the furnace but these were bricked up when the side charging was adopted. The skimming bay is in the side wall of the furnace, 15 feet from the throat, and the tap-hole is close to it. Skimming is done intermittently at intervals of about two and a half hours and the matte is tapped as required, but is never allowed to accumulate to a higher level than four inches below the skimming plate.

The fuel used is pulverized slack coal, having the following composition:—

	Per cent.
Volatile matter	37.00
Free carbon	50.00
Ash	13.00
S	3.30

The coal, after being drawn from the storage bins, passes through a Jeffrey coal cracker, which breaks up any lumps to three-quarters of an inch or less. A 16-inch conveyor belt now carries the coal about 300 feet to a storage bin of 70 tons capacity. The Merrick weightometer weighs the coal on the belt. The coal as received contains from 8 to 10 per cent moisture and before going to the pulverizers it is passed through one of two Ruggles-Cole double-cylinder rotary driers which reduces the moisture to about 2.5 per cent. Further moisture escapes in the subsequent handling so that the coal, as finally delivered to the burners, contains about 1.0 to 1.5 per cent moisture. From the drier, the coal is elevated to three 50-ton bins, from which it is fed to three four-roller Raymond pulverizers. When fine enough, the coal is drawn up by the suction of a No. 11 Sturtevant special exhaust fan and delivered to a dust collector on the roof of the building. It is fed from the collector into the trough of a 16-inch helicoid screw conveyor, which delivers it to a 60-ton bin above and behind the reverberatory furnace. The fineness of the coal is very important and a screen test should show at least 85 per cent passing through a 200-mesh screen, and none larger than 100-mesh.

Five short screw-conveyors take the pulverized coal from the storage bin and drop it in front of a blast, which carries it into the furnace through five five-inch pipes, projecting through the bridge wall. Two No. 8 Sturtevant fans are installed for supplying the blast, but ordinarily only one is used. The blast furnishes only part of the air required for combustion, the balance being drawn in by natural draft through openings in the bridge wall. (Plate XXIII.) Control of the draft is given by a damper in the flue about 30 feet from the throat of the furnace. The speed of the screws which supply the burners is adjustable and the amount of coal fed to the furnace can be varied from 60 tons to 120 tons per 24 hours when all burners are operated. At present about 80 tons of coal is being burned and the furnace is smelting about 500 tons of charge per day, made up as follows:—

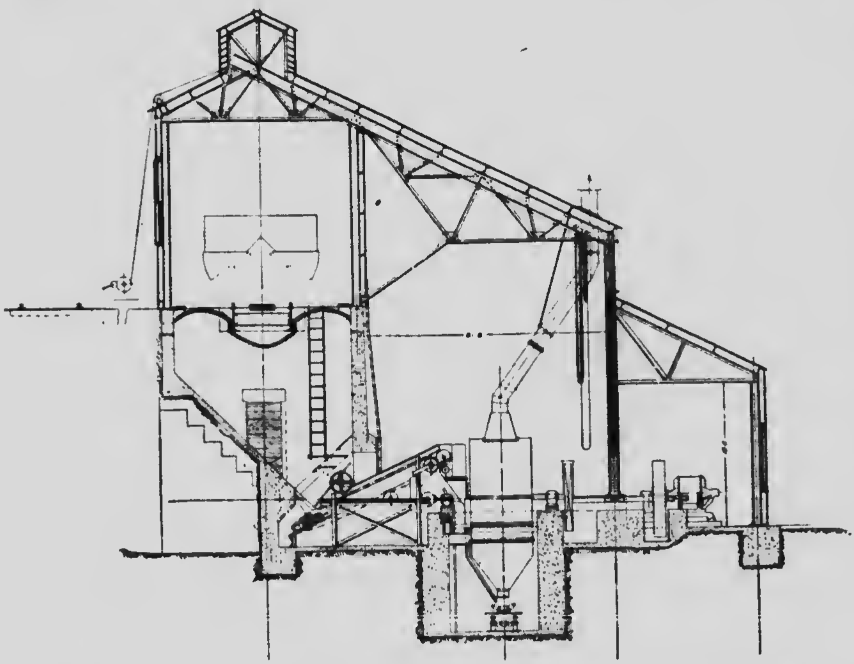


Figure 24.—Cross-section of crushing house, reverberatory plant.

	Tons.
Hot calcines.	**400**
Roaster flue dust	**10**
Blast-furnace flue dust	**30**
Green ore	**30**
Sample-house discards	**30**
	500

This mixture requires no flux and gives a slag containing about 33 per cent silica and a matte with about 16 per cent copper-nickel. If a higher grade matte is made, a silicious flux must be added to keep the slag loss from increasing. The reverberatory slag contains approximately the same percentage of copper-nickel as the blast furnace slag.

In the above charge it will be noted that 80 per cent hot material is shown, and that about six tons of charge is smelted per ton of coal burned. This is about the average practice, but on occasions the amount of calcines is as high as 90 per

Figure 25.

cent of the charge and the coal ratio then becomes about seven to one.

The temperature at the hottest part of the furnace is about 1,580° C. (2,900° F.) and the gases escape at 1,100° C. (2,000° F.) A 400 H.P. boiler is placed in the flue to utilize this heat. After leaving the boiler the gases pass through a brick flue before entering the stack. Provision is made for recovering any dust that settles in the flue, but very little is ever found here. The gases are finally discharged through a brick stack similar to those at the blast furnaces and of the same height.

Converters.

The converter department contains six basic-lined Peirce-Smith converters. The shells are 37 feet 2 inches long by 10 feet in diameter, and are constructed of ⅝-inch steel plate. The throat for the escape of the gases is placed at the centre of the shell and is about 5 feet 6 inches in diameter. On either side are two small openings through which the flux is charged. At the front of the shell there are two spouts for skimming slag or casting the finished matte, but only the one near the control levers is used.

The converter rotates on four 12-foot tread rings carried on equalizing trucks which are mounted on concrete piers. The shell is turned by means of two wire cables anchored to the front and back just below the throat and operated by a hydraulic piston working in a horizontal cylinder. On account of the low temperatures during the winter, oil instead of water is used to transmit the pressure. In addition to a hand-controlled operating system, each converter is provided with an automatic safety device which brings the converter off the tuyeres should the electric power be unexpectedly cut off.

There are 44 tuyeres 1½ inches in diameter on each converter; only 28 of these, however, are regularly punched. The blast is supplied to the tuyeres from a 16-inch bustle pipe running the length of the converter about three feet above the tuyere line. The pressure in this pipe is maintained at about 10 lb., under which condition there is no trouble in keeping the tuyeres open. If more than 28 tuyeres are kept open the blast pressure falls and the punching becomes very labourious. The tuyeres just below one stack opening are seldom punched on account of their tendency to throw molten material out of the shell.

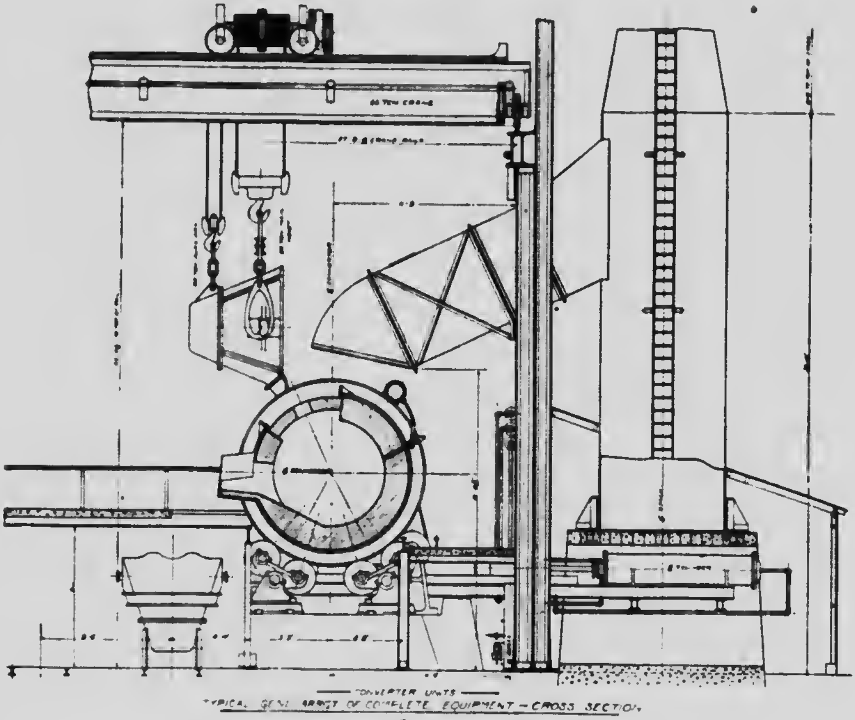


Figure 26.

Probably the most interesting feature of the smelter power plant is the Rateau Battu Smoot turbo-blower installed in 1917, which is the chief source of supply for the converters. This blower has a capacity of 42,000 cubic feet of free air per minute to a discharge pressure of 13 pounds gauge. A constant pressure is maintained by an automatic unloading valve in the blower inlet. This automatic unloading and stabilizing equipment is such as to permit a constant pressure of 13 pounds for all volumes up to, and includng, 42,000 cubic feet per minute, within a variation not exceeding 4 per cent, except such momentary variations as may occur at the instant one or more converters are thrown on or off the tuyers. The regulating equipment is specially adapted to meet these conditions, and the variations in pressure are confined to a few seconds' interval, after which the regulator assumes control and normal pressure is regained. The synchronous motor, direct-connected to the blower engine operates at 2,400 volts, 25-cycle, 3-phase, alternating current, with a normal speed of 1,500 R.P.M. This with a similar installation at the Creighton mine helps to maintain a power factor of around 95.

The magnesite lining of the shells is 24 inches thick at the bottom, 18 inches at the back or tuyere section, and 15 inches at the front. The top arch is of 9-inch magnesite. The life of the linings varies considerably, owing mainly to the difference in the grade of matte treated, but a converter in constant use should last for from six to nine months without repairs of any kind, and usually all that is required then is small repairs above the tuyere line and the removal of the accumulations that have solidified at the bottom. From records kept over a long period, the average consumption of magnesite is about six lb. per ton of iron oxidized or about 12 lb. per ton of converter matte produced.

The building is served by two 50-ton Morgan cranes having a travelling speed of 250 feet per minute. They are used for charging matte scrap and flux into the converters, casting the Bessemer matte, 'sculling' ladles and loading excess scrap into steel cars. The converter slag is handled by a 'Dinkey' locomotive on a narrow-gauge track and the matte is brought from the reverberatory plant in the same way. The matte from the furnace is brought in over a short narrow-gauge track connecting the two buildings, the ladle being placed on a car operated by an endless cable. As already mentioned, the furnace-matte pots are of sectional cast iron and have a capacity of about eight tons. Similar pots are used for the reverberatory matte. The slag pots are also of sectional design, but the side sections are of cast steel instead of cast iron. They have a capacity of about nine tons, and are occasionally used for reverberatory matte as well as for converter slag. The slag pots are held in a yoke on the slag cars and can be turned down to spill their contents on the ground, or can be lifted by the crane for pouring converter slag into the settlers in the furnace building or for pouring reverberatory matte into the converters.

The finished converter matte is poured into cast iron moulds, of which there are 14, situated in a matte shed along one side of the main building. Each mould is about 36 feet long by 6 feet wide, and, when full, holds matte to the depth of about 4½ inches. There are 18 sections in each mould and every third section has a rib on the upper side extending across the mould except for an 18-inch gap in the middle.

This rib creates a line of weakness across the matte after it has solidified and facilitates breaking. Before the matte is poured into the moulds a cast-iron lifting-block with an aperture for a hook is placed in the centre of the area marked out by a pair of ribs, and the moulds are given a lime wash to prevent the matte sticking to the iron. Soon after the matte has solidified, and while it is still quite hot, an air lift which travels on an overhead I-beam is hooked to the lifting block and a slab of matte about six feet square and

Photo by British and Colonial Press, Toronto.
Plate XXV.—View of converter building.

four inches thick, weighing about 1½ tons, is picked up and transferred to a set of grizzlies at the floor level. Below the grizzlies is a small pocket in which stands a side-dump steel car that has a capacity of about 2½ tons of matte. There is one of these loading pockets at each end of the matte shed. The matte is broken by sledges to pass through the grizzlies and when the car has been filled it is drawn up a short incline by an electrically operated hoist into a standard box-car for shipment to the refinery at Bayonne, N.J., or to Port Colborne, Ontario.

At one end of the main building there is equipment for handling and storing the clay and flux required in the operations. There are two Ruggles-Cole rotary driers through which all flux passes before being elevated to the storage bins. These bins hold about 700 tons and have compartments for quartz and mine rock, the fluxes used. By means of a system of small hoppers and conveyor belts below the bins, straight quartz or mine rock, or any desired mixture of the two, can be sent to the converters. The flux is discharged into a 10-ton steel box which has lifting trunnions by which it can be handled by the crane. Two Carlin pug-mills prepare all clay required at the different departments of the smelter. Most of the clay used is obtained locally, but a small amount of fire clay is used for 'budding.'

Converters are started with a charge of 70 or 80 tons of furnace or reverberatory matte, and 5,000 or 6,000 lb. of flux. The first blow is continued until a good slag has formed, which may require an hour or more. Subsequent blows are usually of 35 or 40 minutes' duration. If everything is running as it should, this is sufficient time to raise a pot of slag. The skimming of the slag and the charging of fresh matte and flux takes from 10 to 15 minutes. The operations of blowing, skimming and charging are repeated until the converter has received from 300 to 400 tons total blast furnace and reverberatory furnace matte. The amount of matte that a converter will take depends on the grade of the matte, and the condition of the interior of the converter. The accretions on the sides and walls, and the accumulations at the bottom of the shell reduce its capacity after it has been in commission for a few months and it is found advisable to shut the converter down periodically to have these cleaned out, even though no repair to the brick is required.

When all the iron in the matte charged has been oxidized, no further slag rises, and any additional blowing simply reduces the sulphur content. This, however, cannot be carried on to the extent of removing all the sulphur, but matte containing only 13 per cent sulphur is readily obtainable if desired. It is found that practically all the iron is removed when the grade of the remaining matte is about 75 per cent Cu.Ni., but at this plant it is customary to raise the grade to 79 or 80 per cent before casting. Matte of this grade will contain from 0.30 to 0.50 per cent iron, the balance being almost entirely sulphur. Further blowing has very little effect on the iron. The amount of iron oxidized per minute of blowing is about 190 lb. and is used as a measure of the efficiency of the converter work.

As already explained, under the blast furnace practice, part of the converter slag made is poured into the furnace settlers. This amounts to about 70 per cent of the total, the remaining 30 per cent consisting of the 'sculls' from the ladles, and a portion that is poured on the ground and smelted as part of the blast furnace charge.

The slag usually made contains about 27.5 per cent silica, which is higher than is found at the majority of copper smelters. It is quite feasible to make the silica 20 per cent or even lower, and it is advantageous to do so from the standpoint of the converters alone, but when the practice of pouring the slag into the settlers was adopted it was found necessary to keep the silica around 27 per cent in order to get the best results. A low silica converter slag soon caused the settlers to fill in with a sticky mass not conducive to good settling.

As mentioned above, quartz and mine rock are the fluxes used. The former contains about 91 per cent silica, but no metal values, the latter about 53 per cent silica and 1 per cent copper-nickel. As the rock is an otherwise waste product from the picking belts at the mine, but contains some metal value, it is advantageous to use a certain amount of it, but its low silica content, none of which is present as free silica, makes it an inefficient flux. It is found possible, however, to use it in about equal quantities with the quartz, but for the last three or four blows of each charge it is preferable to use quartz only. The silica content of the average mixture used is about 72 per cent, which after fluxing its accompanying bases to a 27.5 per cent silica slag, leaves about 62 per cent to do useful work in the converter.

Power.

Hydro-electric power, developed at High Falls on the Spanish river, is used at both mines and smelter. High Falls is a small settlement reached by a company-owned spur from Turbine, a flag-stop on the Soo branch of the C.P.R. and on the Algoma Eastern about 28 miles from Sudbury.

The river flow past High Falls is not sufficient at all times of the year to furnish enough energy. To offset this disadvantage, a natural storage has been developed on the upper reaches of the Spanish in the vicinity of Biscotasing. There, are situated several large lakes upon which the lumbermen years ago had built water storage and regulating dams to facilitate the delivery of logs to their sawmill in Biscotasing. The right to use these dams for conservation purposes has been acquired and the structures themselves greatly improved. Their use in regulating the supply of water, while very materially improving the stream flow, is, however, necessarily wasteful because of their distance and the three and a half days' time required for the water to reach High Falls.

A new concrete dam is now under construction, about three-quarters of a mile above High Falls power house. This structure will create a large lake 25 miles long, which, although not duplicating the Biscotasing storage, will, on account of its position, provide for efficient regulation of the water, with practically no wastage. The actual flow, save in flood periods, will come through three large automatically operated Johnson valves. The control for these valves will be operated by floats placed below the new dam on the forebay above the water wheels of the High Falls power house. This control will be transmitted electrically to the valves themselves. The new storage will also conserve the summer rains over at least three-fourths of the Spanish River drainage area above High Falls. Thus, except in the three months' period of extreme high water, the best average flow of the river can be maintained with practically no loss.

There are two power houses at High Falls. Both are steel frame buildings with brick walls. The older building has a wood roof 4 inches deep with tin roofing, but on the new structure the roof is of tile with a built-up Barrett specification covering. The floors are plain and of reinforced concrete where necessary. In the new building the window frames are of steel, with wire glass. Only in exceptionally cold weather has it been found necessary to heat these buildings, and for this purpose in the old building there is placed a hot water heating boiler.

In the new power house there is a vertical type I. P. Morris water turbine rated at 7,500 H.P., driving a 5,500 K.V.A. Westinghouse generator. This machine has an over-all efficiency of 89 per cent at full load. In the old building there are four horizontal type I. P. Morris water wheel turbines with Crocker-Wheeler generators having a nominal rating of 2,000 K.V.A. each.

In the old building there are also two exciter units of 320

H.P. each and two triplex oil pumps, one a spare, furnishing hydraulic power for the governors to control the supply to each of the water wheels. Back of the wheels, on a high platform, is the switching equipment. All of the generators' output from both buildings is here distributed and handled by remote electric control through the various breakers. These breakers are all situated over the transformers to the rear of the switching and alongside the substation, together with the switching performances of the machines. The electric energy developed is a 3-phase, 25-cycle current and is generated for the transformers at a nominal voltage of 2,600. There it is stepped-up to 33,000 for transmission to the mines and the smelter.

The transmission line consists of two sets of three No. 1 B. & S. gauge copper wires arranged with equilatoral triangle 4-foot spacing. The wires are supported by stands made up of A-poles with a common cross-arm. The stands are spaced at about 150 feet and number around 1,150 in all. The poles are untreated and, after 14 years of service, are seemingly good for as many more years. Practically no replacements have been made. The cross-arms have been painted the usual mineral red and during the current year it was found necessary to replace about 10 per cent on account of dry rot.

The current delivered to the mine at Creighton and to the smelter at Copper Cliff is stepped-down at these points to 2,200, 550, and 110 volts for various uses. The lighting circuits everywhere are 110 volts and most of the motors about the plants are 550 volts. Motor generator sets furnish direct current to charge the battery and trolley locomotives underground, on the furnace charge floor, and to dump slag pots at the smelter. The 2,200-volt circuits drive the larger machines in the power houses at both mines and smelter.

At the mines, the maximum portion of the power is consumed by the hoists, the air compressor, and the rock-house motors. In the hoist house, the 1,500 K.V. capacity flywheel motor-generator set furnishes D.C. energy for one 1,800 H.P. Wellman-Seaver-Morgan hoist. In the compressor building, two Ingersoll-Rand compressors use 923 rated K.V.A. each at full load, and a Belliss & Morcom compressor uses 900 K.V.A. at full load. Each of these furnishes approximately 5,000 cubic feet of air at 100 lb. per square inch. In the rock-house, about 500 rated horse-power is used when in full operation.

In the event of the current being interrupted from any cause, there is in constant readiness at the smelter an auxiliary steam plant capable of furnishing 1,200 boiler horsepower at 160 pounds pressure.

Safety and Welfare Work.

This paper would not be complete without a brief description at least of the safety and social work carried on by the company for the protection and betterment of its employees.

Accident prevention was organized as a separate department in July, 1913, and placed in charge of an experienced safety engineer. The movement was decidedly successful and the accident rate was considerably lowered. In 1915 the Workmen's Compensation Act became effective in Ontario, and the work of reporting accidents to the Board was added to the duties of the Safety Department. In this way a record is kept of each accident, from the date of its occurrence until final compensation is paid by the Board and the case settled. Assistance is given the employees in their correspondence with the Board and every effort made to follow up the cases, to see that workmn, prevented by disability from following previous occupations, are given employment at suitable work, to prevent malingering and lessen in every way the great economic waste caused by lost-time accidents.

Hospital and Medical Attention.—In order that employees and their families may have proper medical attention in case of accident or sickness, the company has provided a commodious hospital equipped with the latest approved devices for the treatment of surgical and medical cases.

Eight experienced doctors are employed and one is on duty continuously, ensuring prompt attention in case of accident.

Education and Entertainment.—In the town of Copper Cliff, primary education is provided for by a large public school of modern design and equipment, where a school population of over six hundred is accommodated. All grades up to High School work are taught and on completion of the public school course, the High School at Sudbury is available for Copper Cliff pupils.

At Creighton and Crean Hill mines, modern public schools have been erected, and the cosmopolitan population of the nickel district places these schools in a class by themselves in diversity of nationalities in attendance.

At Copper Cliff the company has erected, at a great expense, what is conceded to be one of the finest club buildings in the province. All employees are eligible for membership. The club is equipped with bowling alleys, pool and billiard tables, swimmng pool, reading room and a large assembly hall where weekly dances and card parties are held. Membership in the club is restricted to boys over eighteen and girls over sixteen, those under these minimum ages being taken care of in the Junior Copper Cliff Club. In the latter club, a specially qualified Lady Superintendent teaches domestic science, dancing, deportment and physical training. In connection with the club there is also a Cadet Company of over eighty members, and a well-equipped gymnasium with classes for boys of all ages, also workmen's class in boxing, wrestling and physical training.

Insurance.—Shortly after the Dominion regulations regarding group insurance were altered to permit of such insurance being written in Canada, the company was enabled to protect the families of employees by issuing policies, effective November 1st, 1919, as follows:

Each employee that has completed one year's continuous service is insured for $500.00. This is increased to $800.00 at the end of two years, to $1,200.00 at the end of three years, $1,600.00 at the end of four years, and the maximum policy of $2,000.00 is issued on completion of five years' service. This insurance is payable to the beneficiary named by the employee in event of death or total disability. The initial payment is 20 per cent of the total, with the balance in twenty-four semi-monthly instalments.

Pensions.—In order to provide for aged employees who have given long service to the company, the policy has been adopted of pensioning employees who have become incapacitated after twenty years' continuous service. Pensions are paid monthly from the general funds at a figure based on the pensioner's average earnings for the last year worked (based on full time). After twenty years' service the pension is fifty per cent of the average earnings, after thirty years' service, 62½ per cent, and so on in proportion to the number of years in the employ of the company.

After twenty years' continuous service, old age, sickness, injury or incapacity from any other cause, may be considered sufficient reason for granting a pension.