Developments in Mining at Giant Yellowknife

*By J. R. SMITH**

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Introduction

GIANT MINE in 12 years of operation from May, 1948 to June, 1960 has produced 2,918,019 tons of ore averaging 0.787 ounces of gold per ton. In the last ten years, the milling rate has risen from approximately 300 tons per day to slightly over 1,000 tons per day. Underground operations have been extended laterally over a strike distance of 10,000 feet and vertically to 2,000 feet.

At the 1953 Annual C.I.M. Meeting, Mr. D. C. McDonald, then Mine Superintendent at Giant, presented a paper "Mining at Giant Yellowknife" which very thoroughly documented the mining practice up to that time. This paper will deal mainly with the last four or five years of the operation and any phases which have undergone little change since 1953 will be dealt with very briefly or omitted.

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Shafts

The mine was originally developed through three shafts — A, B and C. (See Figure 1).

A Shaft, a three compartment shaft, 793 feet deep was abandoned as a working shaft in October, 1957 when mining in that area was completed. It is now used only for a ventilation airway.

B Shaft, also three compartment, is still in operation as a service shaft.

C Shaft, a five compartment shaft, handles all ore and waste and the bulk of the servicing. This shaft was originally sunk to 1,029 feet in 1950, deepened to 1,529 feet in 1954 and further deepened to 2,124 feet in 1959.

Originally this shaft was equipped with a CIR 60 x 36 PE-I hoist handling a skip-cage and a skip in the two skip compartments. In 1955 this hoist was placed in operation with a large service cage in the centre compartment and a CIR 96 x 54 PE-I hoist was installed to take care of the skipping.

In the original shaft, a loading pocket was installed at the 900 foot elevation. During the first shaft deepening to 1,529 feet, a crusher station was cut and loading pocket installed at the 1,300 foot and 1,400 foot elevations respectively and in 1956 a 36″ x 48″ Buchanan Jaw Crusher was installed. At present all ore is passed through this crusher and reduced to 5 inches before being hoisted.

After the last deepening the 900 loading pocket was moved to the 1,700 foot level to handle development waste from the 1,500 foot and 1,650 foot levels and to eventually handle ore produced from stopes between the 1,250 foot and 1,650 foot levels.

In addition, a new pocket was installed at the 2,050 foot elevation to handle development waste from the 2,000 foot level. The pockets at the 1,400 foot and 1,700 foot levels

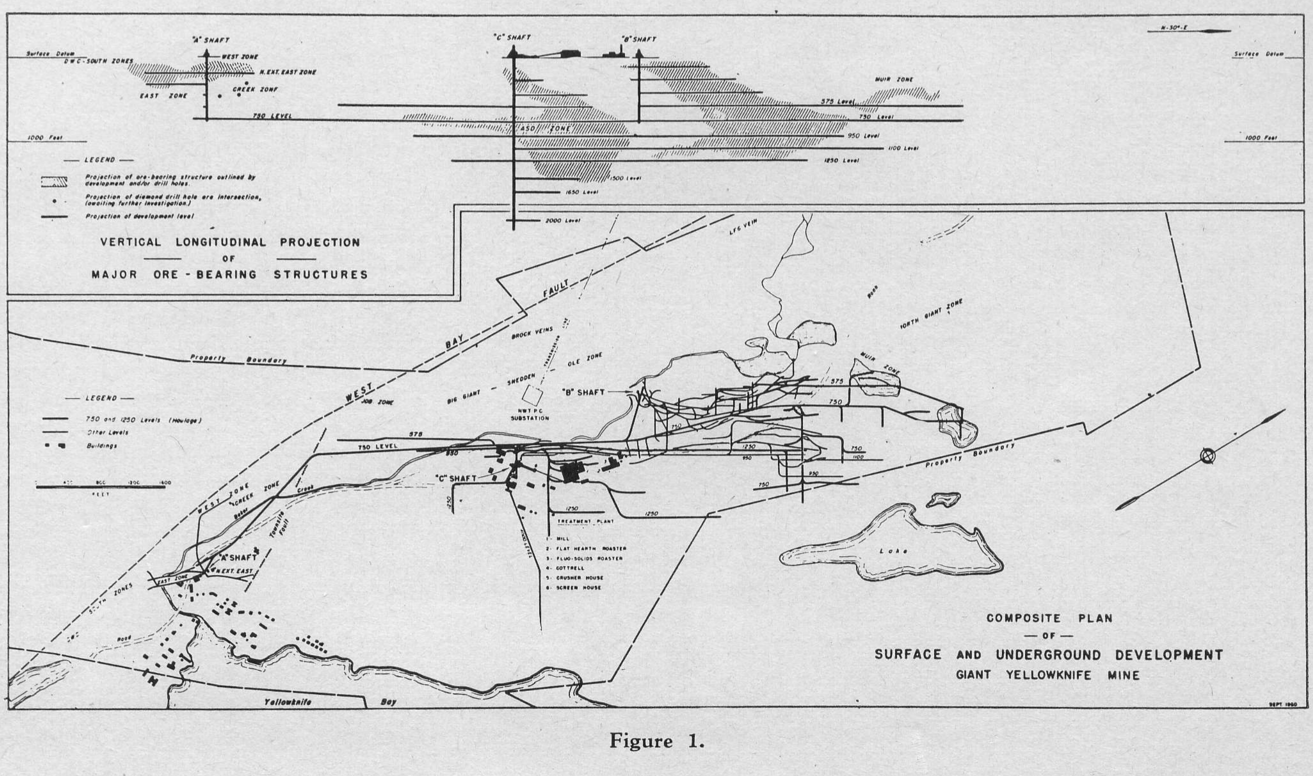


Figure 1.

are arranged to skip double ore or single ore and single waste. The pocket at the 2,050 foot level is designed for single waste hoisting with provision for expansion to double skipping when the need arises.

Development

The ore bodies occur as irregular masses in schist zones which vary in attitude from vertical to horizontal. For a detailed description of the ore bodies see (2).

Base development drives are planned from information obtained by diamond drilling from levels above or other headings on the same level. The usual approach is to drive in waste in either the foot wall or hanging wall of the ore and thoroughly outline it by diamond drilling before planning the extraction.

Drifts and Cross-Cuts

Drifts and cross-cuts are driven 8 feet by 8 feet in cross section by a two man crew using airleg rock drills, 7/8 inch hexagonal collared steel and tungsten carbide slip-on bits. Mucking is done with an Eimco 21 Loader. Trackage is 24 inch gauge with ties at 30 inch centres and may be laid with either 20 lb. or 30 lb. rail depending on expected tramming tonnage in the heading. The water line is normally 2 inch and the air line may be 4 inch or 2 inch depending on the distance to be driven. All pipe is lightweight rolled groove victaulic. All blasting is done with 1 inch Cilgel B initiated with tape fuse and ignitor cord.

The two man crew mucks out, drills and blasts an 8 foot round each shift and puts in pipe and track as required. For development performances see Table I.

Stoping

Originally, cut and fill, shrinkage and open stoping methods were used. Shrinkage and open stoping have been gradually abandoned and during the last five years, with the exception of pillar recovery, all ore has been mined by horizontal cut and fill.

Until the end of 1956, all filling was done with gravel or development waste. At that time, classified mill tailings were first used for stope filling. The use of tailings fill has been gradually increased and at present 77 per cent of the stoping tonnage is produced from tailings fill stopes and 20 per cent from stopes using waste fill. The remaining 3 per cent is produced from pillar recovery and development.

Table I

Development Performances

	1957	1958	1959	1960*
Drifting, Advance Per Man Shift.....	2.70	2.54	2.52	3.09
Sub Drifting, Advance Per Man Shift.	3.55	3.34	3.31	3.61
Raising...........................	3.71	3.61	4.05	3.93

*Note: Fiscal Year End — June 1st.

In waste fill stopes, fill raises are driven 6′ x 6′ or 5′ x 10′ as combined manway — fill raises, at 100 ft. — 150 ft. centres. In tailings fill stopes normally two 5′ x 8′ service raises are driven. In short ore bodies only one central raise is driven. These 5′ x 8′ service raises carry ladders, 2″ air and water lines, 4″ fill line, electric cable, and a large steel slide made as wide as possible to facilitate the moving of large equipment in and out of the stope. A 26″ x 9′ steel bucket operates in the slide powered by a tugger.

Mill holes are carried at 75′—90′ spacing, and each stope usually has two combined mill hole-manways. In both waste fill and tailings stopes, mill holes are constructed of 8″ rough-squared local spruce dapped to give a 4″ spacing between cribs. Inside timber dimensions are 6′-2″ x ′6-2″ (excluding the lining). Mill holes are lined with 6 inch material flatted three sides and cut in 8′ lengths. All mill holes are equipped with grizzlies consisting of 4—85 lb. rail with one 85 lb. cross rail giving 6—18″ x 34″ openings. In tailings fill stopes the sides of the mill hole or mill hole-manway are covered with 1″ mesh chicken wire and 10 oz. burlap.

Breasts are normally carried 8′ high, but if the walls are good, a 9 ft. breast is taken. In the case of 8′ lifts, the fill is topped at 10′ from the back, if a 9 ft. lift, 11 ft. is left between back and floor. Breasts are drilled with a 48″ spacing and 30″ burden. Usually 10 ft. steel is driven

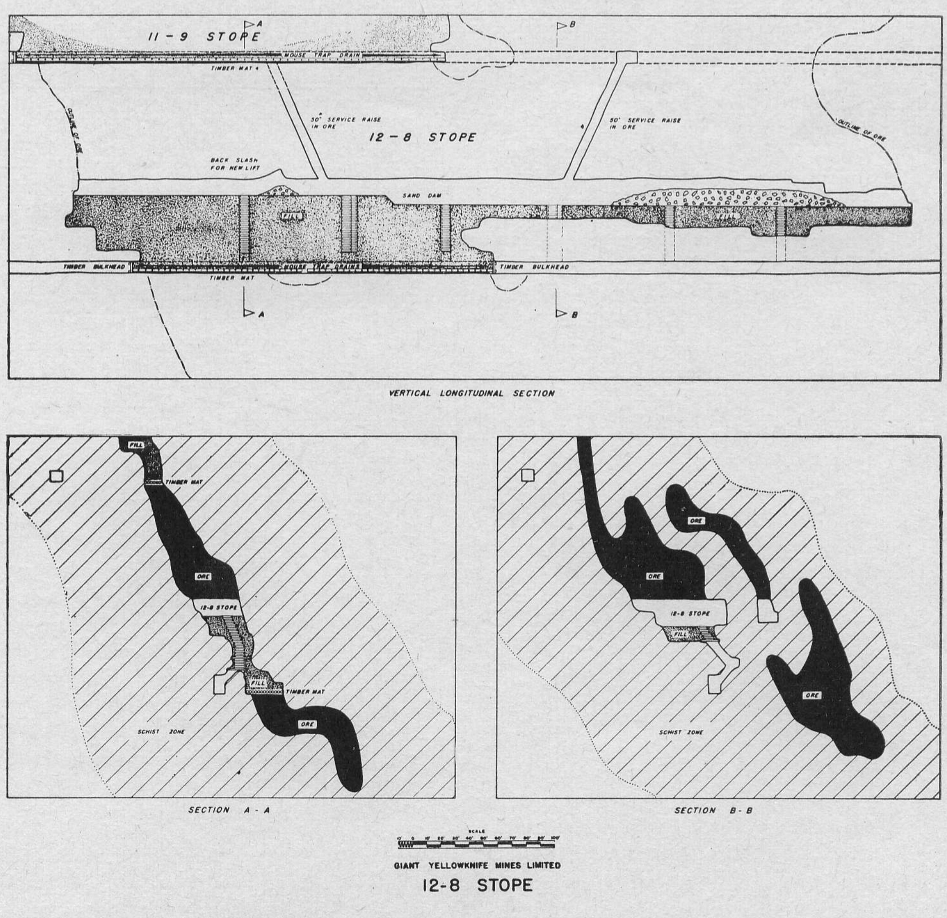


Figure 2.

but 16 ft. holes are used as long as the breast can be drilled off in the shift.

Lagging 16 ft. long with 5″ butt diameter is used for flooring. It is placed at 24 inch spacing, spiked to sill lagging at 8 ft. spacing.

Due to the irregular nature of the ore bodies, stope planning must be kept flexible. While stoping procedures are standardized as far as is possible, nearly every stoping area presents different problems. Several of these problems are illustrated by Figures 2 and 3.

As may be seen from the following tables, stoping efficiencies and costs have improved markedly over the past few years.

While some of this improvement is no doubt due to the mining of wider ore bodies, a great deal has been accomplished by improvement in equipment, technique and better training for stope miners.

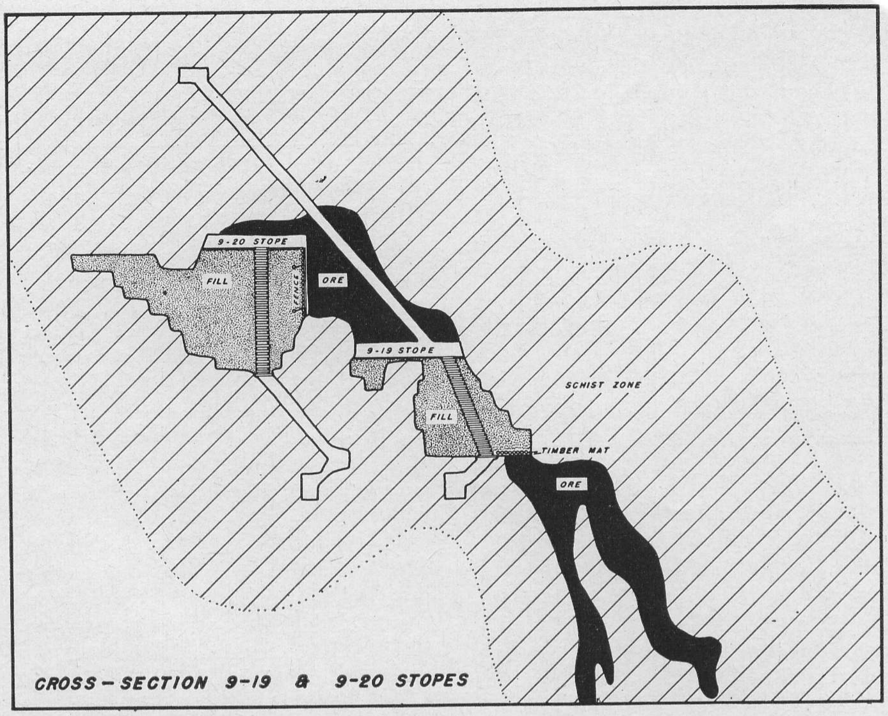


Figure 3.

Stoping Equipment

All stope drilling is done with light weight airleg rock drills equipped with 7/8″ hexagonal collared drill steel and slip-on tungsten carbide bits. For some time, it was found necessary to start with a 1½ inch bit due to excessive gauge loss, but recent experience has shown that a 1⅜ inch starting bit is satisfactory.

Blasting is done with 1 inch Cilgel B and short period delay caps. Powder consumption averages 0.65 lbs. per ton broken.

Expansion shell rock bolts in 5 ft. and 8 ft. lengths are used as required for support of walls and backs. Ground conditions generally are good and little support is needed.

Slushers range in size from 7.5 to 30 H.P. in both air and electric, but for the larger stopes the standard slusher is a 30 H.P. three drum electric with a 48 inch Lok Lip scraper fitted with side retainers. Pull rope is ½ inch best plough and return is ⅜ inch.

Power is supplied to the stopes at 550 volt 3 phase 60 cycle through trailing cables with screw connectors at the slushers.

TABLE II

STOPING PERFORMANCES

	1957	1958	1959	1960
Breaking Tons/Man Shift	60.2	64.2	88.0	87.4
Slushing Tons/Man Shift	54.0	69.2	95.0	88.3
Slushing Waste Fill Tons/Man Shift	71.0	82.7	84.0	91.0
Placing Hydraulic Fill Tons/Man Shift	122	85	119	159

TABLE III

DIRECT STOPING COSTS

	1957	1958	1959	1960
WASTE FILL STOPES				
Tonnage Broken	198,717	100,471	86,832	74,506
Direct Cost/Ton Broken	$ 2.50	$ 2.24	$ 1.76	$ 1.85
TAILINGS FILL STOPES				
Tonnage Broken	70,778	172,323	217,939	280,721
Direct Cost/Ton Broken	$ 2.74	$ 2.11	$ 1.69	$ 1.66
All Stoping Cost/Ton Milled (Direct)	$ 2.24	$ 2.05	$ 1.63	$ 1.69

TAILINGS FILL

As previously mentioned, classified mill tailings were first used for stope filling in December, 1956. The fill plant is extremely simple and was set up with a minimum of expense.

The 400 ton capacity storage tank is a disused 25 ft. diameter agitator, with the rake mechanism removed and fitted with a series of high pressure nozzles in the bottom of the tank. The nozzles are ¼ inch UH 10006 V Jet nozzles with a 1/16 inch jet orifice and are made of hardened steel. They are replaced approximately once a year.

Tailings are pumped by a 6 x 6 SRL 40 H.P. pump to four 10 inch Krebs Cyclones in parallel, mounted over the storage tank. Feed to the cyclones is at 22 lbs. per square inch and 25 per cent solids. From the cyclones the slime portion passes to tailings disposal and the sand product drops into the tank. For a typical screen analyses of the feed, overflow and sands, see Table 4. Percolation rate of the fill produced averages 3.6 inches per hour.

From the storage tank the fill is fed through a cone and 6 inch line fitted with a pinch valve into a surge tank. From here it is pumped at 65 per cent solids by a 25 H.P.

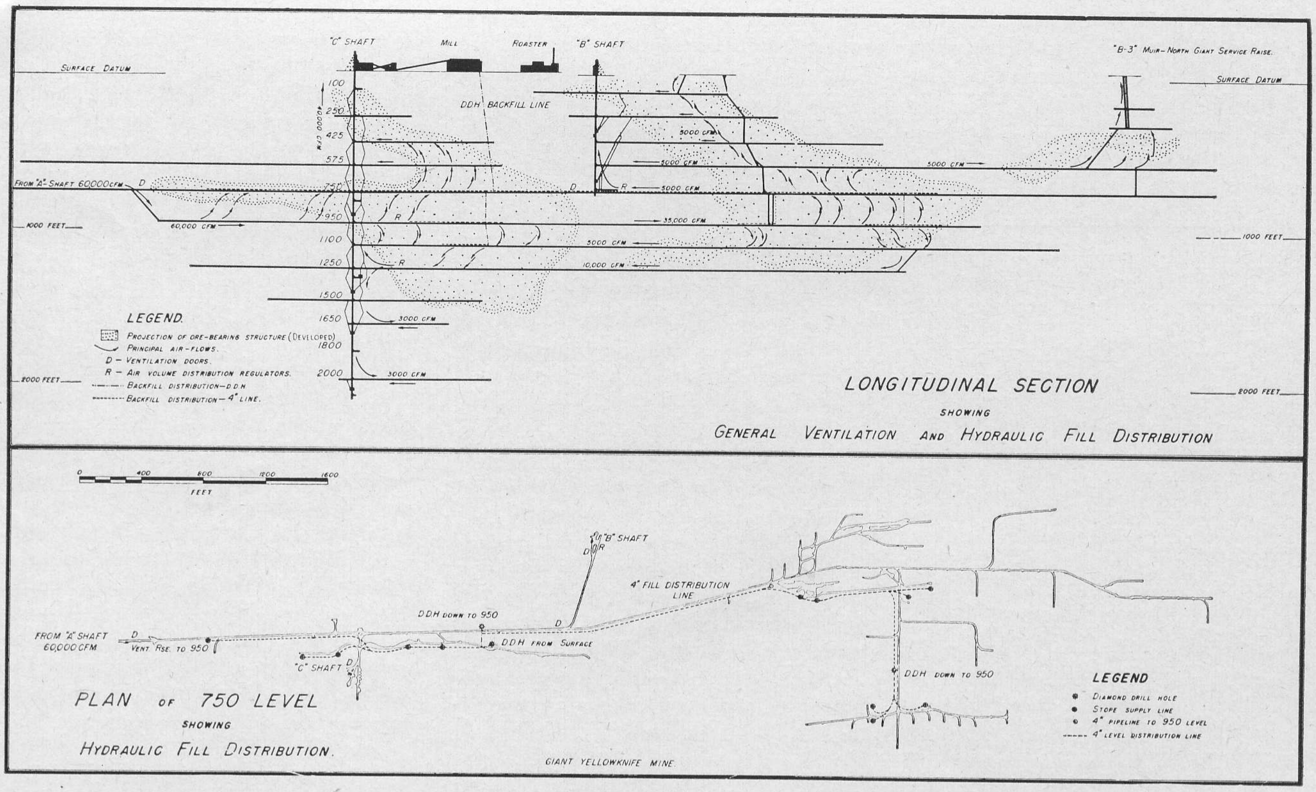


Figure 4.

TABLE IV

	Wt. Per Cent for Mesh Size					
	+100	+150	+200	+270	+325	+325
Feed to Cones.......	5.6	5.8	8.0	7.3	6.8	66.5
Cone Overflow......	0.1	0.2	0.4	0.7	1.7	96.9
Sand Product........	12.3	12.6	17.5	15.4	12.5	29.7

Wilkinson 5 x 5 Linatex Lined Centrifugal pump to a diamond drill hole collared in the mill and connecting with the 750 foot level as shown in Figure 4. This diamond drill hole was an old EX surface exploration hole. It was reamed to 2.94″ diameter and both top and bottom collars were further reamed to $4\frac{1}{2}$″ diameter for 20 inches, and $3\frac{1}{2}$″ diameter for an additional 12 inches. A steel fill hole adapter was inserted into the reamed portion of each collar and secured to the rock by two rock bolts.

The fill is distributed to stopes by a system of 4 inch standard weight victaulic pipes and 3 inch vertical and horizontal diamond drill holes. The steel pipe is carried part way down the stope service raises and 3 inch carlon pipe used for distribution within the stope.

The longest horizontal 4 inch line is 4,200 feet. At present fill is distributed down as far as the 1,100 foot level for filling stopes between the 1,100 ft. and 1,250 foot levels.

To date, slightly over 300,000 tons of fill have been handled through the system with the present monthly average being approximately 11,000 tons. As yet no appreciable pipe wear has been encountered. Periodically all lines are checked and horizontal lines are rotated 90° to even out any wear. In a few cases wear has started at the victaulic groove and caused a break, but it is planned to eliminate this weakness by the use of Plain Lock Victaulic fittings which require no groove.

Since fill is required only on day shift, during the afternoon and graveyard shifts the fill is allowed to settle in the storage tank and the excess water drawn off. At present practically all fill made in 24 hours can be poured in 4 to 6 hours.

All filling underground is handled by a two man fill crew working on day shift only. This crew looks after all cat walk construction and damming in addition to pouring fill. The operating procedure for fill crew and mill operator is as follows:

(1) The fill crew checks the lines into the stope to be filled, enters the stope, hooks up their portable 'phone and calls for flushing water from the mill.

(2) The mill operator starts the feed pump and runs clear water into the system until a flow is reported underground. When the fill crew are sure the system is clear they call for fill.

(3) The mill operator turns on the high pressure water to the jets and opens the pinch valve. The dilution and feed rate are held at the desired level by adjustment of the pinch valve and high pressure water.

(4) On completion of the filling underground, or when the tank is empty, communication is made by 'phone and the flow is cut off. A clear water flush is made and the pump shut down.

The fill crew, while pouring is in progress, make their own dams by banking the fill from the inside to form a barrier at the open end of the section being poured.

Pouring is continued until the fill is level with the top of the raised mill holes. The lagging floor is then laid. Small dams are then raised around each mill hole using scrap or 1″ rough planking and burlap. A final topping fill is poured bringing the finished level flush with the top of the lagging.

Operating costs for hydraulic fill are as follows:

	Per Ton of Fill	Per Ton Broken
Placing Labour (Mine).........	16.5¢	7.4¢
Placing Supplies (Mine)........	6.5¢	2.9¢
Maintenance of System.......	0.5¢	0.2¢
Fill Preparation (Mill).........	7.0¢	3.1¢
	30.5¢	13.6¢

Ventilation

Until the fall of 1957 the mine was ventilated by forcing 12,000 cu. ft. of air per minute down a combination ventilation and fill raise at B shaft and exhausting through A and C shafts. Additional ventilation was obtained by forcing air down a similar raise at A shaft.

As the mine developed laterally and in depth it became necessary to revise the ventilation system. On completion of mining in the A shaft area in 1957, a new ventilation — heating plant was installed, utilizing the abandoned A shaft as the downcast airway. All other openings to surface now become upcast, thus avoiding the usual problems associated with freezing air entry. Main ventilation flows are illustrated in general form in Figure 4.

Control of air flow is maintained by regulating doors at strategic points but it is anticipated that as the mine develops further the system will be expanded to a 'push-pull' type with exhaust fans placed to achieve more positive control.

Long or isolated development headings, particularly those operating on a three shift basis, are ventilated by a blowing system using 5 to 15 H.P. electric or air driven booster fans coupled to 20 inch diameter polythene ventilation tubing.

Ventilation — Heating Plant

Since temperatures drop to —40°F and lower and the mine is not opened to a depth sufficient to supply natural heating, it is necessary to heat the intake air during the winter months.

Three 48 inch diameter Woods two stage 48 J2VG fans are operated in parallel, each with a matching 48 inch diameter Cob 480 Horizontal Heat Exchanger. See Figure 5. Fan delivery is 60,000 cfm at 3.4 inch SWG on two stage operation and 40,000 cfm at 1.5 inch SWG on one stage.

Each heating unit is rated at 1,300,000 BTU output/hour with an overall thermal efficiency of 80 per cent. The heating units are operated on Esso Marine Diesel fuel with a rating of 163,800 BTU/Gal. On single stage the air temperature rises averages 20°F per burner or a total rise of 60°F with all burners operating.

Temperature control was originally effected by thermostat relay in the downcast, set to regulate the number of burners and fan stages in operation. Using this system led to frequent "on-off" operation of the burners particularly in certain temperature ranges. This meant that the combustion chambers were subjected to repeated extreme temperature fluctuations which resulted in the development of thermal stress failures.

After the first year's operating experience, the manufacturer replaced the heat exchangers with redesigned units which were found to be better able to stand extreme thermal stresses. However, it was still considered desirable to avoid the repeated stresses caused by "on-off" burner control. The alternative of continuous burner operation and automatic volume control was considered to be wasteful of fuel.

Fortunately, drainage water flows to A shaft were available. This water was utilized for the alternate formation and thawing of ice in the shaft and station areas to take care of rapid intake air temperature fluctuations rather than turning individual burners on and off.

Downcast air temperature is continuously recorded and the number of burners in operation is adjusted daily, or more often if required, to maintain an average downcast temperature of 32° to 33°F. Due to a rapid drop in temperature of the intake air it is possible at certain times to be sending air underground at 0°F, but due to freezing of water in the downcast, this air will be received in the main distribution drift at 32°F. Thus it is unnecessary to use dampers to reduce one stage delivery of the fans even at temperatures below —50°F, provided temperatures do not hold in this range too long. The lowest mean temperature of —26°F recorded during the month of January can be handled by three burners in continuous operation.

Toward the end of the winter season burners are cut back and ice allowed to build up to reduce fuel consumption. Naturally ice build-up cannot be carried to the point where it seriously interferes with the air flow in the downcast. Frequent inspections are made of the shaft and station areas to ensure that this does not happen.

Haulage

Standard track gauge is 24 inches with a grade of 0.25 per cent in favour of the load. For headings handling large tonnages, 30 lb. rail is used, with 20 lb. being used in secondary headings.

Equipment used on the main haulage level (950 Foot), consists of a 3 ton electric trolley and 60 cu. ft. Granby cars. The cars are loaded using a 3½ ton battery locomotive and left at a gathering point for the trolley which hauls them 3,200 feet to the main ore pass.

Other haulage levels use 1½ ton or 3½ ton battery locomotives hauling 38½ cu. ft. side dump or 60 cu. ft. Granby cars.

Development headings use 38½ cu. ft. cars and 1½ or 3½ ton battery locomotives depending on the tramming distance.

Dump points at the main ore pass are equipped with retractable ramps and ore pass covers operated by air cylinders. When the underground crusher was installed, all grizzlies on the ore passes were removed with the exception of the B shaft passes.

The majority of the chutes are equipped with a vertical gate operated manually by a cantilever arrangement. Chutes handling heavy tonnage are equipped with a cramp chain gate actuated by an air cylinder fed by a 4 way valve. From the valve, ¼ inch cable is strung along the drift enabling the motor man to do his own loading while seated on the motor, spotting the cars by means of wall markings. This practice eliminates a chute puller and has been a major factor in the improvement in tramming productivity. (See Table V).

Mining Costs

As may be seen from Table 6, in spite of labour costs which per hour have risen approximately 20 per cent from 1955 to 1960, mining costs have been substantially reduced over the past five years.

Some of this reduction is no doubt due to the volume effect of tonnage increase, the closing of A Shaft and the mining of wider ore bodies, but a great deal of the saving has been effected by improved techniques, the use of more suitable equipment, and the stabilization and better training of the working force.

Mining research has played a very large part in reducing costs. In addition to a continual program of performance testing on the existing equipment, investigations are presently under way on such projects as semi-automatic hoisting, the use of drill rigs for stoping, the use of steel in plate or rail form as a substitute for lagging for stope flooring and many others.

It is to be expected that much of this research will bear fruit in the coming years.

Acknowledgments

The author wishes to thank Mr. M. K. Pickard, General Manager and the President and Board of Directors of Giant Yellowknife Mines Limited for permission to present this paper. Grateful acknowledgment is made of the assistance in preparation by Mr. G. S. Dundas, Mine Superintendent, and Mr. D. G. Adam, Chief Engineer. The preparation of the drawings by Mr. G. H. Espley and Mr. B. Kerns is also appreciated.

Table V

Tramming Performance

	1957	1958	1959	1960
Tons Trammed Per Man Shift........	64.1	88.3	120.0	131.4

Table VI

Underground Operating Cost/Ton Milled

	1956	1957	1958	1959	1960
Underground Crushing..............	—	0.03	0.05	0.07	0.05
Development......................	1.91	1.26	1.05	0.97	1.08
Stope Preparation..................	1.07	1.17	1.05	0.64	0.65
Mining............................	4.36	4.12	3.73	3.24	3.36
TOTAL..................	7.34	6.58	5.88	4.92	5.14

References

(1) McDonald, D. C., Mining at Giant Yellowknife; C.I.M. Trans., Vol. LVI, 1953, pp. 77—87.

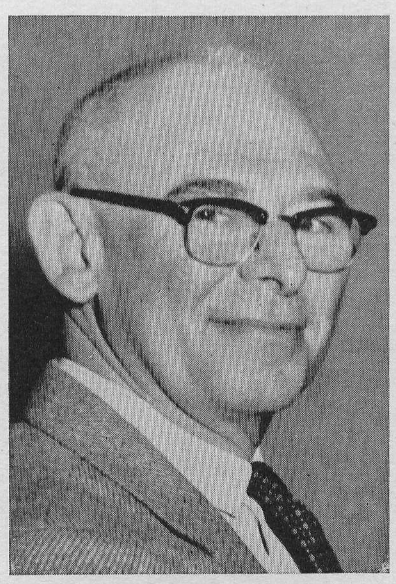
(2) Brown, C. E. G., Dadson, A. S. and Wrigglesworth, L. A., On the Ore-Bearing Structures of the Giant Yellowknife Gold Mine; C.I.M. Trans., Vol. LXII, 1959, pp. 107—116.

Quebec Metal Mining Association and Quebec Metal Mines Accident Prevention Association

Annual Meetings, Seigniory Club, Que., June 5-7

THE REGULAR Annual General Meetings of the Quebec Metal Mining Association and the Quebec Metal Mines Accident Prevention Association were held again this year at the Log Chateau of the Seigniory Club, Quebec, on Monday, Tuesday, and Wednesday, June 5, 6, and 7. The programme was planned in a manner similar to previous years and all sessions were well attended.

The Q.M.M.A. held a meeting of the Board of Directors on Monday morning. On Tuesday morning the Annual General Meeting of the members was held in the Conference Hall, followed by an open meeting for guests as well as members. Mr. D. E. G. Schmitt, President 1960-61, presided at each meeting and conducted a most interesting session for the open meeting. The programme included an address by Dr. Paul E. Auger, Deputy Minister of Natural Resources, Province of Quebec, entitled *A New Mining Camp in Quebec: Mattagami.* The session was concluded by the showing of a film entitled *Accomplishments in Northern Quebec,* presented by Mr. L. J. Patterson, Vice-President of Operations, Quebec Cartier Mining Company.



F. J. O'Connell
Pres. Q.M.M.A.

The Q.M.M.A.P.A. held a meeting of the Board of Directors on Monday morning. On Wednesday morning the Annual General Meeting of the members was held in the Conference Hall, and included a report by Mr. N. H. George, Director of Safety. This meeting was followed by an open meeting for guests as well as members. In the absence of the 1960-61 President, Mr. W. G. Brissenden, the meetings were conducted by Vice-President F. G. Cooke. The open session included a report on *Ventilation* by Mr. Gerard Grassmuck, a report on *Mine*

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