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# A study of the ventilation of the Hiawatha Mine, Menominee iron range, Michigan

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A STUDY OF THE

VENTILATION OF THE HIAWATHA MINE

MENOMINEE IRON RANGE, MICHIGAN

BY

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A

THESIS

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Approved by J. D. Forrester
Professor of Mining Engineering

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## INTRODUCTION

Mine operators in the Menominee iron range have become aware of the need for adequate ventilation to reduce the amount of dangerous dust in the air. Dust-excluding respirators and water sprays are used but it is apparent that the prevalence of diseases caused by dust will be appreciably reduced only if an adequate amount of fresh air reaches the working places.

During the past five years the necessity for direct control of the underground working environment has become particularly urgent. The increase in production rate, with two-shift and three-shift operation replacing the single shift, has complicated the problem of dealing with excessive concentrations of blasting fumes and gases, heat, and oxygen deficiency.

The objects of the study described below were:

To determine the quantity and quality of air reaching the working places.

To determine the amount of heat flowing into the mine air and to ascertain the importance of each source of heat.

To ascertain the most economic method of increasing the amount of fresh air reaching the working places.

The field work was carried out between December 22, 1947 and January 1, 1948.

## DESCRIPTION OF THE OREBODY

The Hiawatha orebody lies along the steeply dipping north limb of a syncline which closes around on the east end to form a bath-tub shaped structure. The orebody dips vertically and strikes east throughout most of its length of 3600 feet although near the east end the strike changes to a northerly direction. The average thickness of the orebody is 40 feet. The footwall is a black graphitic slate which is liable to spontaneous combustion on exposure to air; the hanging wall is a cherty, fairly hard iron formation which stands well in open stopes.

## DESCRIPTION OF THE MINE

Because of the size and shape of the orebody, sub-level stoping has been an efficient method of extracting the ore. Formerly stopes were not filled but in recent years stope filling has been necessary because of the outbreak of fires in the slate and the steady crushing of the pillars. Stopes are filled with glacial drift through raises and 30 inch diameter churn drill holes.

The upcast shaft, 2100 feet deep, is used to hoist men and ore in two counterbalanced units, each of which is a combination of a skip and a cage. All drifts are 8 feet by 8 feet in cross-section. About 380 gallons of water are pumped from the mine each minute.

## VENTILATION SYSTEM

The mine is ventilated by two intake raises from the surface to the third level where the air splits into the main east side current and the main west side current. The east side current is drawn from the third level by a fan located at the ninth level and forced down two vertical raises to the No. 17 level, where it is distributed to the working places.

The west side current is drawn through an open stope by two fans located on the ninth level; from the ninth level the air is forced down raises and open stopes to the No. 17 level where it is distributed to the working places. The distribution of the air is shown in Figure 1.

In summer a 30 in. diameter drill hole and a raise on the extreme east side of the orebody increase the amount of intake air; in winter these airways become blocked by the formation of ice.

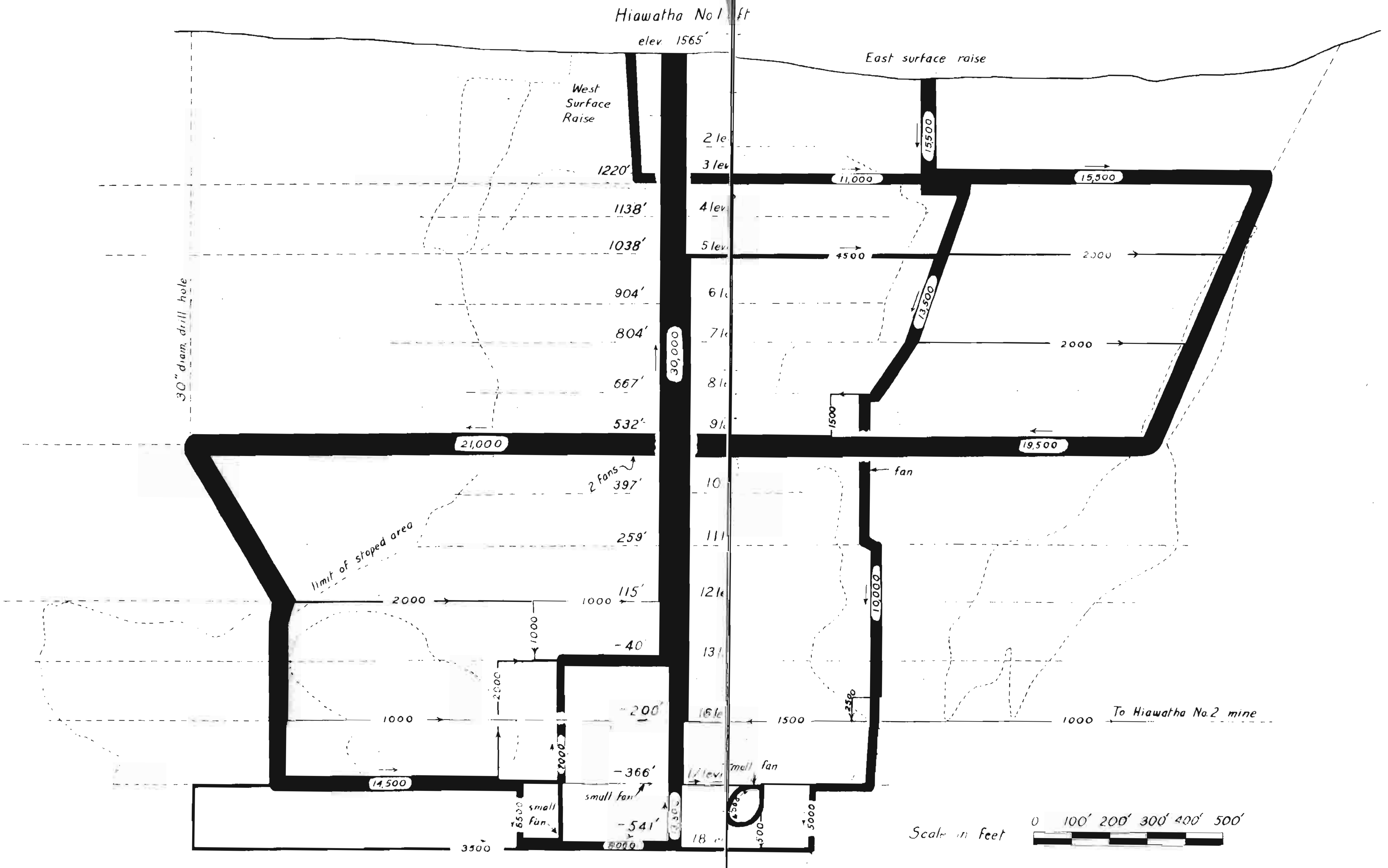
## METHOD OF CONDUCTING THE SURVEY

### Measurement of air currents

The quantity of air was calculated from the velocity of the air and the cross-sectional area at the place of measurement. Each measurement of air velocity was made in the following manner:

The anemometer, mounted on the end of a 4 foot steel rod, was held upstream in the upper right corner of an airway and allowed to run freely before setting the gear train in motion.

Fig. I. AIR DISTBUTION



As soon as the gear train was set in motion, the anemometer was traversed slowly across the airway in vertical movements as shown in Figure 2. The time elapsing between the start and finish of the traverse was recorded on a stopwatch.

The average velocity of air was obtained from the following formula:

$$\text{Av. velocity} = \frac{\text{anemometer reading}}{\text{time in minutes}} - \text{correction in fpm}$$

Measurements in drifts were taken as far from other airways as possible to obtain uniform air flow over the whole cross-section. Where the drift was timbered the measurement was made in the vertical plane of a timber set.

A low velocity anemometer was used in the same manner for all velocities below 200 fpm. In very sluggish air currents the velocity was determined by timing a cloud of smoke over a measured distance. Although this method was inaccurate, accuracy was not necessary because of the small quantities of air involved; moreover, the method was useful in determining the direction of the movement of air and in detecting leakage.

The cross-sectional area was measured by taking offsets at one foot intervals along a rod held vertically in the center of the airway. In timbered drifts the height and width at mid-height were measured.

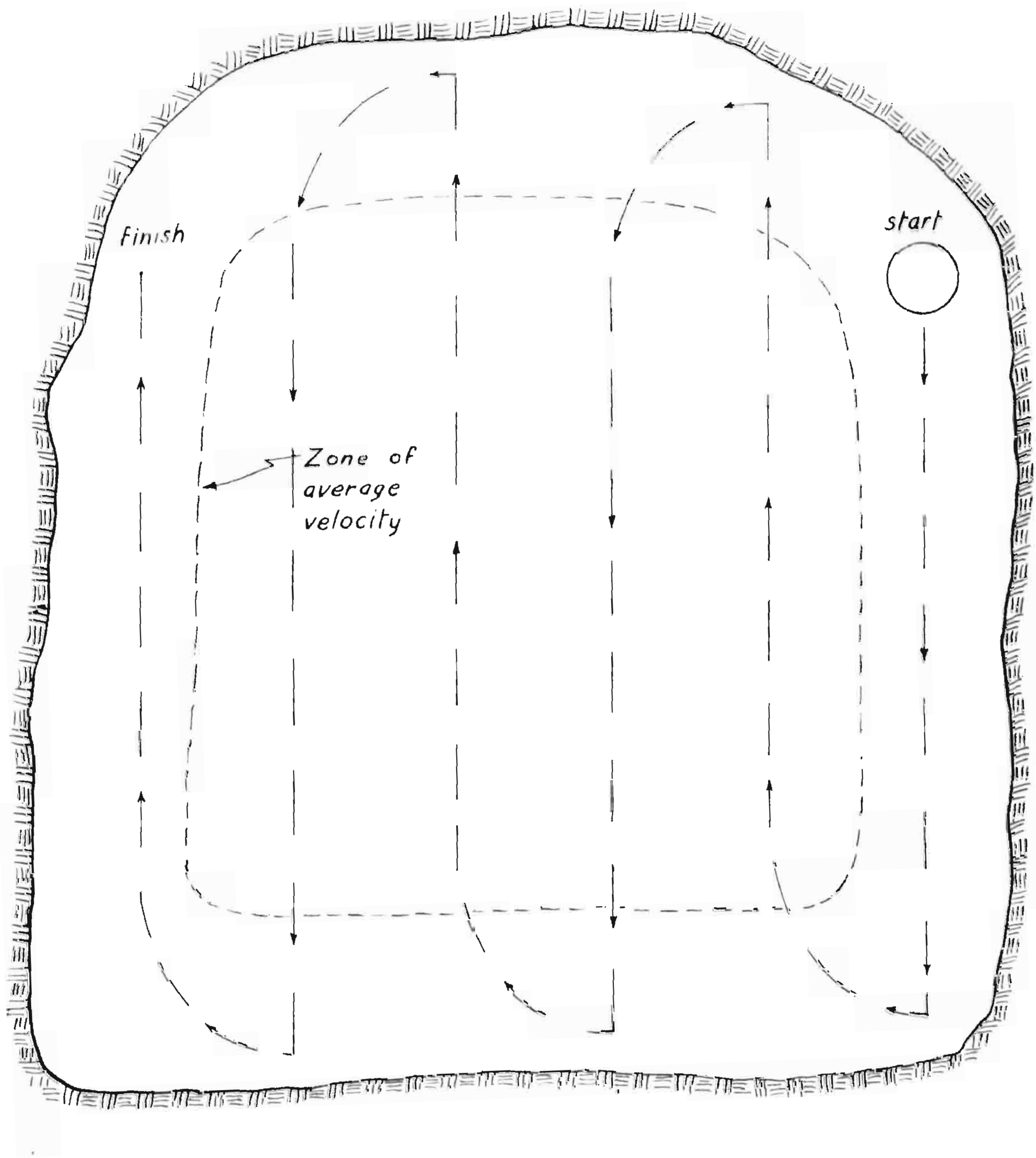



Fig. 2. ANEMOMETER TRAVERSE

The velocity in the discharge tubing of a fan was measured using a pitot tube connected to a water gage in a manner similar to that described by Weeks.(1)

(1) Weeks, W. S., Ventilation of mines, p. 26, New York, McGraw-Hill Book Company, 1926.

The anemometer could not be used for this measurement because the high air velocities in fan tubing would damage the bearings of the instrument.

The quantity of air passing in the drift was calculated immediately after the velocity was measured and was compared with the quantities measured in the airways that connected with the drift. In this way discrepancies between the air quantities were detected immediately and, if necessary, further measurements were made until agreement was reached.

The survey was carried out in sequence starting at the intake airways and working downward to the No. 18 level. It was not possible to follow the course of the air currents all the way down because some of the airways were inaccessible. By keeping check of the air quantities, however, deviation of the air currents from their expected course was detected immediately. Upon detection of any leakage all possible exits of the air were entered and measurements were taken to ascertain where the air was leaking. From the results of these measurements the distribution of the air to the various parts of the mine was mapped. Figure 1 shows the distribution of air by full lines, the thickness of which is proportional to the quantity of air.

Mine temperatures

Air temperatures were measured with a mercury thermometer graduated to 0.1°C. To avoid erroneous readings of air temperature resulting from heating of the air by the body, the thermometer was held upstream at arms length and swung slowly in the air current. The wet bulb and dry bulb temperatures were measured with a whirling hygrometer which was swung slowly back and forth for several minutes before a reading was taken. Three readings were taken at each station and the lowest wet bulb reading and the average of the dry bulb readings were taken as the wet bulb temperature and the dry bulb temperature respectively.

Wet bulb temperatures and dry bulb temperatures in the shaft were measured by holding the hygrometer at arms length in the center of one of the shaft compartments and swinging it slowly until the mercury column was constant.

Temperatures were measured at the surface each day at 8 a.m. and 2 p.m. It was found that at a dry bulb temperature of several degrees below freezing point, a wet bulb reading could be obtained by swinging the hygrometer for a prolonged period of time.

The relative humidity, moisture content, and vapor pressure of the air at various points in the mine is shown in Table 1. These quantities were obtained from tables.(2)

(2) Jeppe, C. W. B., Psychrometric Tables: Jour. Chem. Met. Min. Society of South Africa, January 1931.

Figure 4 shows the change in temperature of the air in its passage through the mine.

Pressure measurements

Barometric pressures in the mine airways were measured with a small aneroid barometer which had been calibrated against a Fortin mercury barometer in the Mining Department of the Missouri School of Mines. Thirty-seven duplicate readings of the Fortin barometer and of the aneroid barometer were taken over a period of two months before, and one month after the survey. The Fortin reading when corrected for temperature and capillary errors, averaged 0.07 in. of mercury lower than the aneroid reading.

Although the readings were taken at various temperatures and pressures, the difference between the readings of the barometers did not exceed 0.085 in., and was not less than 0.060 in. The difference did not appear to vary with temperature.

The aneroid readings in the mine airways, when reduced by 0.07 in., were considered to be within 0.01 in. of absolute accuracy. Because the error, if any, in two consecutive measurements of pressure would be the same, it was clear that the error in measuring difference of pressure was dependent only on the error of observation. With this in mind, care was exercised in reading the aneroid to obtain maximum accuracy.

Because of the accuracy with which the barometric pressure could be measured, the static pressure produced by a fan could be determined from the difference in barometric pressures on the intake and return sides of the fan. As the static pressure obtained in this way is in terms of inches of mercury, it may be expressed in inches of water by multiplying the figure obtained by the specific gravity of mercury (13.6).

Barometric measurement of static pressure provided a check on the more accurate water gage method which is descibed below.

The static pressure across fans and bulkheads was measured with a water gage. The method of measurement was as follows:

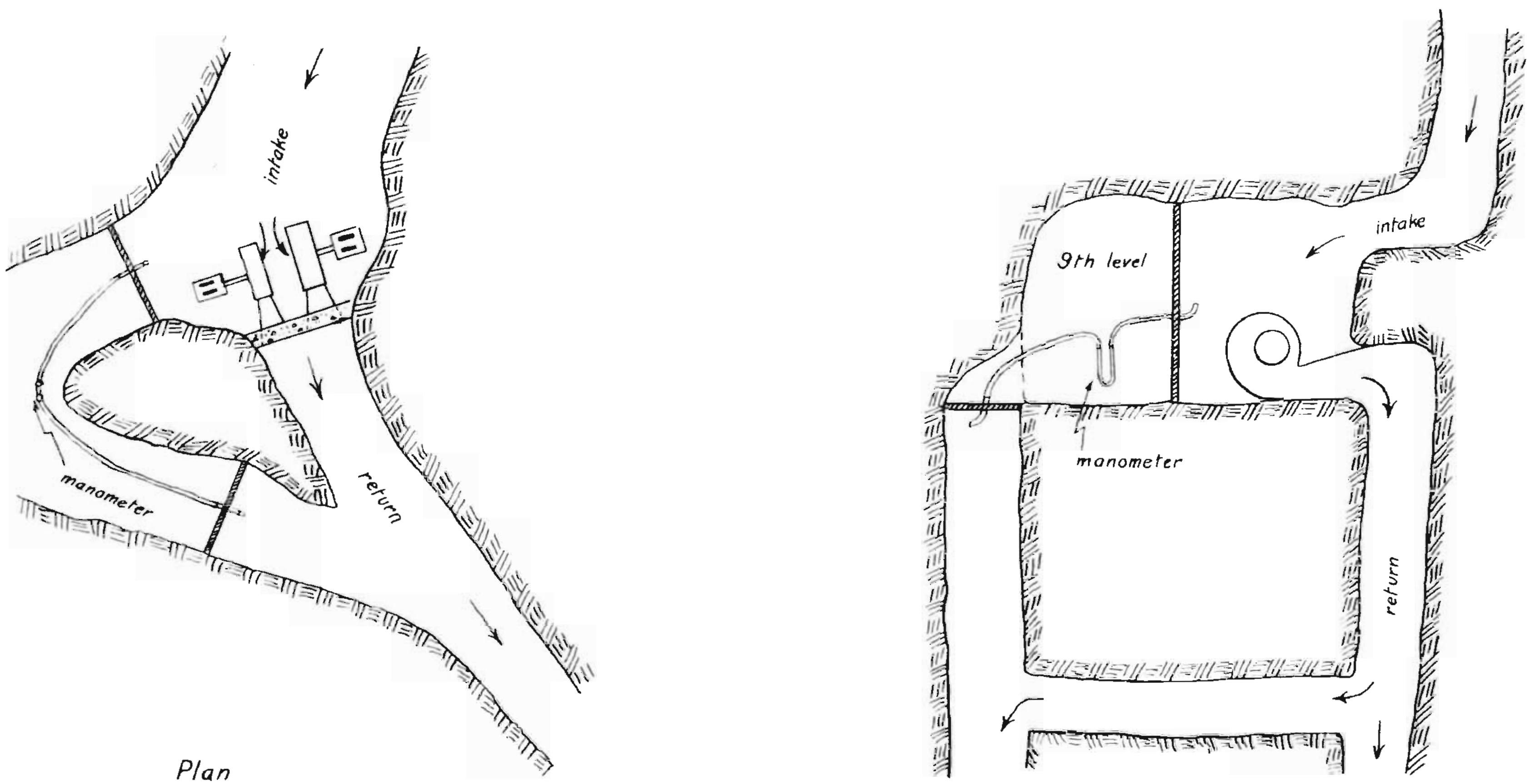
A hole was drilled in the bulkhead and a piece of hollow metal tubing was inserted in the hole. The metal tubing was connected by flexible rubber tubing to one arm of a water gage the other arm of which was open to the surrounding air. The static pressure of the two main fan units was measured across the intake and return doors as shown in Figure 3.

## DISTRIBUTION OF THE AIR

After making an extensive survey of air conditions in the Michigan iron mines, Urban (3) recommended that a minimum

(3) Urban, E. C. J., Health hazards associated with iron mining operations in the Lake Superior region: Am. Inst. Min. Met. Eng. Tech. Pub. 1620, 1943.

# Fig. 3. LOCATION OF FANS




Plan
West Side

Section
East Side

of 800 cfm of fresh air be supplied for each man underground. On this basis, the total quantity of air for the average shift of 40 men should be at least 32,000 cfm of fresh air. The survey showed that the total quantity of fresh air reaching the working places was less than this.

Recirculation occurs at the fifth level, where 4500 cfm of air from the upcast shaft mixes with the fresh intake air. Because of this recirculation 15 percent of the air reaching the working places is foul upcast air. It is apparent from Figure 1 that the total quantity of air reaching the working places is 25,500 cfm and of this about 85 percent is fresh air. Thus the total quantity of fresh air reaching the working places is 22,000 cfm which is equivalent to 550 cfm for each man underground.

In one of the stopes between the No. 17 level and the No. 18 level, 4000 cfm is being recirculated by a forcing fan on the No. 17 level. It is apparent that the practice of ventilating stopes by forcing air down from the upper level tends to induce recirculation because the heated exhaust air tends to rise and return to the fan.

Natural ventilation

Natural ventilation results from the difference in weight of the intake air and of the upcast air. The natural ventilation pressure in lbs. per square foot is equal to the difference in weight of two vertical columns of air, one of which is in the upcast shaft and the other is in the intake airways.

The calculation of the weight of each column of air and of the natural ventilation pressure is shown in Table II. Each column was considered to extend from the No. 18 level, 541 feet below sealevel, to an elevation of 1650 feet which is the elevation of the top of the shaft headframe. Because the top of the shaft headframe is higher than the collar of each intake raise, it was necessary to include in the intake air column the air above the collar of the intake raise and beloww an elevation of 1650 feet. Each air column was divided into portions and the average weight of air in each portion was calculated from the measured temperatures and pressures using Jeppe's tables.[4]

(4) Jeppe, C. W. B., op. cit.

The computation shows that the difference in weight of the two columns is 3.83 lbs. This is the natural ventilation pressure acting on one square foot; in terms of inches water gage this pressure is equal to 0.73 in. water gage.

Because the intake air column is heavier than the upcast air column, natural ventilation pressure is positive; that is, it acts in the same direction as the pressure produced by the fans.

In the summer months, when the surface air temperature is usually above 60°F, the weight of the intake air would be less than the weight of the upcast air, consequently natural ventilation pressure would become negative and would act against the pressure produced by the fans.

## PRESSURE LOSSES

Pressure losses in the mine airways because of air friction were calculated by using the following formula[5]

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(5) McElroy, G. E., Engineering factors in the ventilation of metal mines: U.S. Bur. Mines Bull. 385, p. 41, 1935.

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$$Hf = \frac{k\ P\ L\ q^2\ d}{5.2\ A^3 0.075}$$

Hf is the pressure loss in inches water gage
k is the friction factor obtained from tables
P is the perimeter in feet
L is the length of the airway in feet
q is the quantity of air in cfm
d is the density of air in lbs. per cubic foot
A is the area of the airway in square feet

The value assumed for k in the calculations was governed by the condition of the airway. For straight, relatively unobstructed raises, k was assumed to be $100 \times 10^{-10}$; for raises congested by ladders and pipes, k was assumed to be $110 \times 10^{-10}$. A value of $165 \times 10^{-10}$ was given k for all drifts except the ninth level, which was given a friction factor of $175 \times 10^{-10}$ because of its greater sinuosity and its congested condition.

The calculated friction losses theoretically should equal the pressure potential produced by the fans and by natural ventilation. As the west side fans produce a pressure of 2.2 in. water gage, and the east side fan produces a pressure of 1.8 in. water gage, and natural ventilation pressure is equal to 0.73 in. water gage, the total pressure produced is 4.7 in. water gage. The sum of friction losses in all the important airways, as given in Table III, is 3.2 in. water gage. Thus there is 1.5 in. water gage of pressure potential that is not accounted for by friction losses. There are other minor airways which have not been considered but as these carry a small quantity of air friction loss in them would be negligible. Nor does it seem likely that the friction factors are too low. Hence it appears that the major portion of the pressure potential of 1.5 in. water gage that has not been accounted for is due to shock losses. Inasmuch as all the intake airways have numerous constrictions and sudden changes in size and direction, it is likely that shock loss is considerable.

Because the pressure loss due to shock doesnot vary exactly as the square of quantity, the pressure loss in a mine airway due to shock can not be calculated exactly using arbitrary coefficients.

For practical purposes the friction factor may be increased to include intermittent shock losses and the pressure loss due to friction and shock may be treated as friction loss, which varies as the square of quantity.

To compare the resistance of an airway or a system of airways with that of another airway it is necessary to know the quantity of air that will be passed through each airway or system of airways at a standard pressure potential. Many expressions have been proposed to make a direct comparison of airway resistance.

McElroy[6] gives two formulae which are commonly used

(6) McElroy, G. E., Engineering factors in the ventilation of metal mines: U.S. Bur. Mines Bull. 385, p. 73, 1935.

in comparing mine resistances.

$$\text{Equivalent orifice} = \frac{0.0004\ q}{\sqrt{Hf}}$$

$$\text{Resistance factor} = \frac{Hf \times 10^{10}}{q^2}$$

The equivalent orifice as determined by the above formula is 4.6 square feet. The resistance factor of the mine is found to be 72.

## MECHANICAL VENTILATION

The performance of a fan is measured by the pressure produced by the fan and the quantity of air passing through it. The pressure produced by a fan is the sum of the static pressure and the velocity pressure; the velocity pressure is generally small enough to be neglected in computing the performance of the fan. The performance of a fan is usually stated in terms of air horsepower, which is the rate at which useful work is being done. If the input horsepower to the motor driving the fan and the air horsepower produced by the fan are known, the combined efficiency of the fan and the motor may be computed.

McElroy[7] gives the following formula for air horse-

(7) McElroy, G. E., op. cit. p. 75.

power:

$$\text{Air hp} = \frac{5.2 \times q \times (Hs + Hv)}{33{,}000}$$

q is the quantity of air in cfm
Hs is the static pressure in in. water gage
Hv is the velocity pressure in in. water gage

It is usually inconvenient to determine velocity pressure directly and it is computed from the air velocity using the following formula[8]:

(8) McElroy, G. E., op.cit. p. 15.

$$\text{Air velocity} = 4008\sqrt{Hv}$$

Likewise, if the velocity pressure is known, the air velocity may be determined from the above formula.

Tests were made on each fan to determine the quantity of air produced by the fan and the static pressure across the fan.

The location of the east side fan is shown in Figure 3; this fan is a double inlet Jeffrey fan driven by a 15 hp electric motor which is working at 10 KW input. The door on the intake side of the fan is not absolutely leakproof but because there is no measurable pressure difference across it, leakage from the west side air current is very small. Although the pressure difference across the return door is considerable, this door is well constructed and the amount of air leaking through it is negligible.

Result of test on east side fan:

| | | |
|---|---|---|
| Quantity of air passing | = | 10,200 cfm |
| Static pressure | | |
| across both doors | = | 1.8 in. water gage |
| across intake door | = | Negligible |
| across return door | = | 1.8 in. water gage |

As the velocity on the discharge side of the fan is only 500 fpm, velocity pressure may be neglected so that the total pressure produced by the fan is equal to the static pressure which is equal to 1.8 in. water gage. The air horsepower as determined from the formula given above is 2.9. This is the useful work resulting from a power input of 10 KW which is equivalent to 13.4 hp. Thus the efficiency of the fan and motor combination is equal to:

$$\frac{2.9 \times 100}{13.4} \text{ which is } 21.6\ \%$$

The main ventilating unit for the east side air current consists of two fans working in parallel. The larger of these two fans is an American Blower, size 5½, single inlet fan. The smaller is a Coppus TM 8 fan. Each fan is driven by a 15 hp electric motor working at 10 KW input. The velocity pressure in the discharge duct of each fan was measured by means of a pitot tube.

The static pressure across both fans was measured with a water gage as shown in Figure 3.

Result of test on west side fans:

<u>Coppus TM 8 fan</u>

Area of discharge duct = 2.1 sq. feet
Velocity pressure = 0.85 in. water gage

Velocity may be computed from the velocity pressure using the formula given on p. 17.

Velocity (computed) = 3650 fpm
Quantity of air = 3650 X 2.1 cfm
= 7700 cfm

<u>American Blower fan</u>

Area of discharge duct = 6.0 sq. feet
Velocity pressure = 0.40 in. w.g.
Computed velocity = 2520 fpm
Quantity of air = 2520 X 6.0 cfm
= 15100 cfm

The static pressure as measured applies to both fans.

<u>Static pressure</u>

across both doors = 2.2 in. w.g.
across return door = 0.8 in. w.g.
across intake door = 1.4 in. w.g.

As the velocity of air in the drift on the return side of the fans is 900 fpm, the velocity pressure may be computed from the formula given on p. 17.

velocity pressure = 0.05 in.w.g.
total pressure = 0.05 + 2.2 in. w.g.
= 2.25 in. w.g.

The air horsepower produced by each fan may be computed by using the formula given on p.17.

Coppus TM 8 fan
air horsepower = 2.7 hp
efficiency = $\frac{2.7 \times 100}{13.4}$ = 20 %

American Blower fan

$$\text{Air horsepower} = 5.35 \text{ hp}$$

$$\text{efficiency} = \frac{5.35 \times 100}{13.4} = 40\ \%$$

The test shows that all the fans are working at an efficiency which is lower than that which should be attained in main fans. The fan efficiency should be about 65 percent and if the efficiency of the driving motor is 85 percent, the combined efficiency should be about 55 percent.

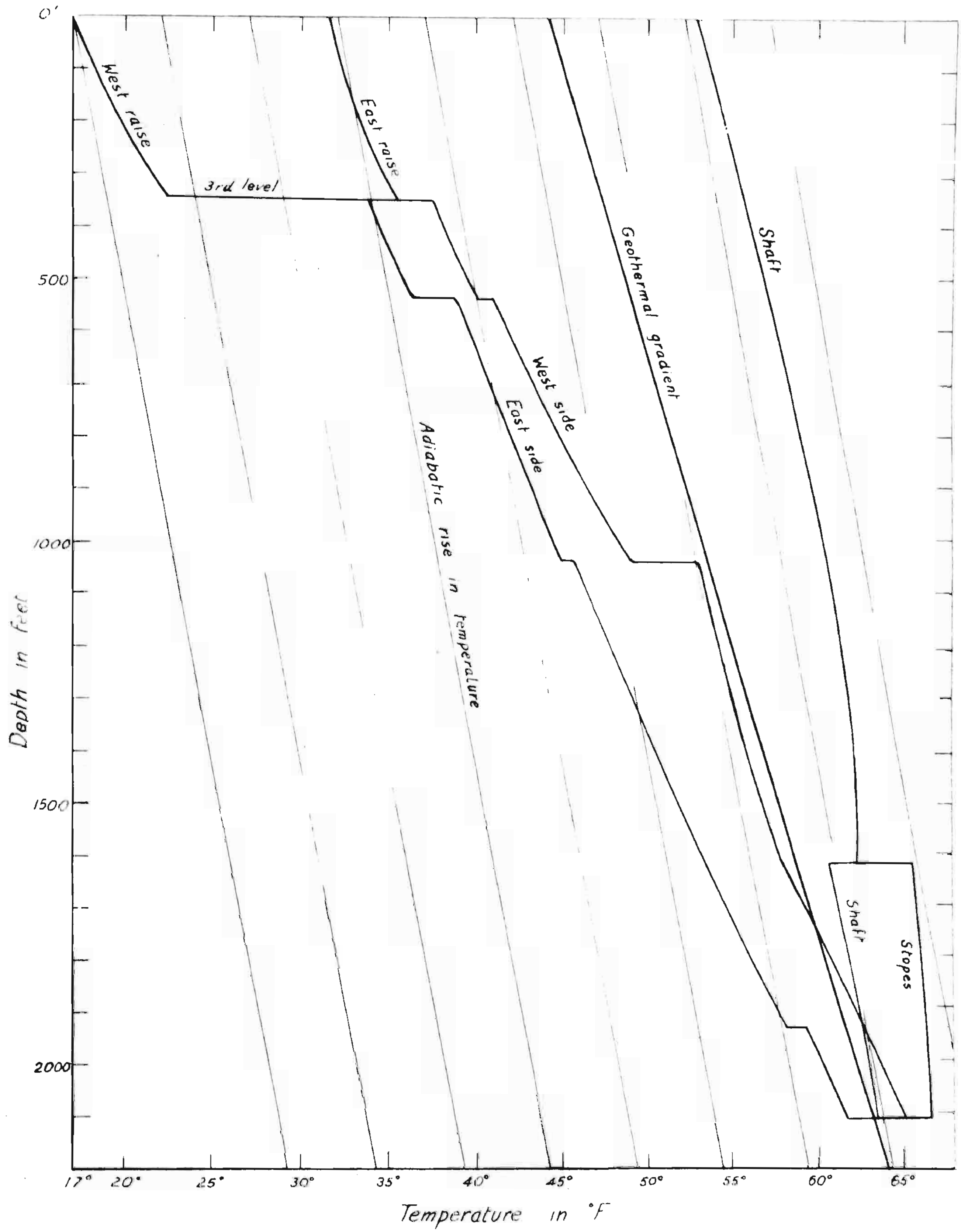
TEMPERATURE AND HUMIDITY

The dry bulb temperature and the wet bulb temperature were measured at various points in the mine and the dry bulb temperature was plotted against depth as shown in Figure 4. The wet bulb temperature and the moisture content of the air at various points in the mine is shown in Table 1.

The heat produced by the operation of the fans has an appreciable effect on the air temperature. This was shown on the ninth level where it was observed that the temperature of the air on the return side of the west side fans was 1.9°F higher than the temperature of the air on the intake side of the fans.

The difference between the dry bulb temperature and the wet bulb temperature was always less than 2°F, except above the third level. In the upcast shaft the wet bulb

# Fig. 4. MINE TEMPERATURES



temperature was the same as the dry bulb temperature at all points above the No. 18 level.

It was noticed that the amplitude of the variation of surface temperature was greatly reduced in the intake airways. At the ninth level, 1033 feet below the surface, the effect of a variation of surface temperature of $10^{\circ}F$ was imperceptible.

In the intake airways, the rate of change of temperature with depth was greater than the rate of change of temperature due to adiabatic compression; in the upcast shaft the rate of temperature change with depth was less than the rate of change of temperature due to adiabatic expansion. It is apparent that there are causes of change of temperature other than that due to adiabatic compression and expansion.

The rock surrounding the intake airways did not affect greatly the temperature of the air except where the velocity was low.

The intake air rapidly absorbed moisture; the relative humidity of the air supplied to the working places was about 95 percent.

## GEOTHERMAL GRADIENT

To measure the geothermal gradient of a mine, it is necessary to obtain either by measurement or calculation the rock temperature at various depths and at the surface. The rock temperature at the surface varies seasonally, the amount of variation decreasing with depth.

Direct measurement of the mean rock temperature at the surface is difficult because of the depth at which temperatures must be taken to reduce seasonal variation to an insignificant amount. Van Orstrand has[9] pointed out

(9) Van Orstrand, C. E., Jour. Washington Acad. Science, pp. 529-539, 1932.

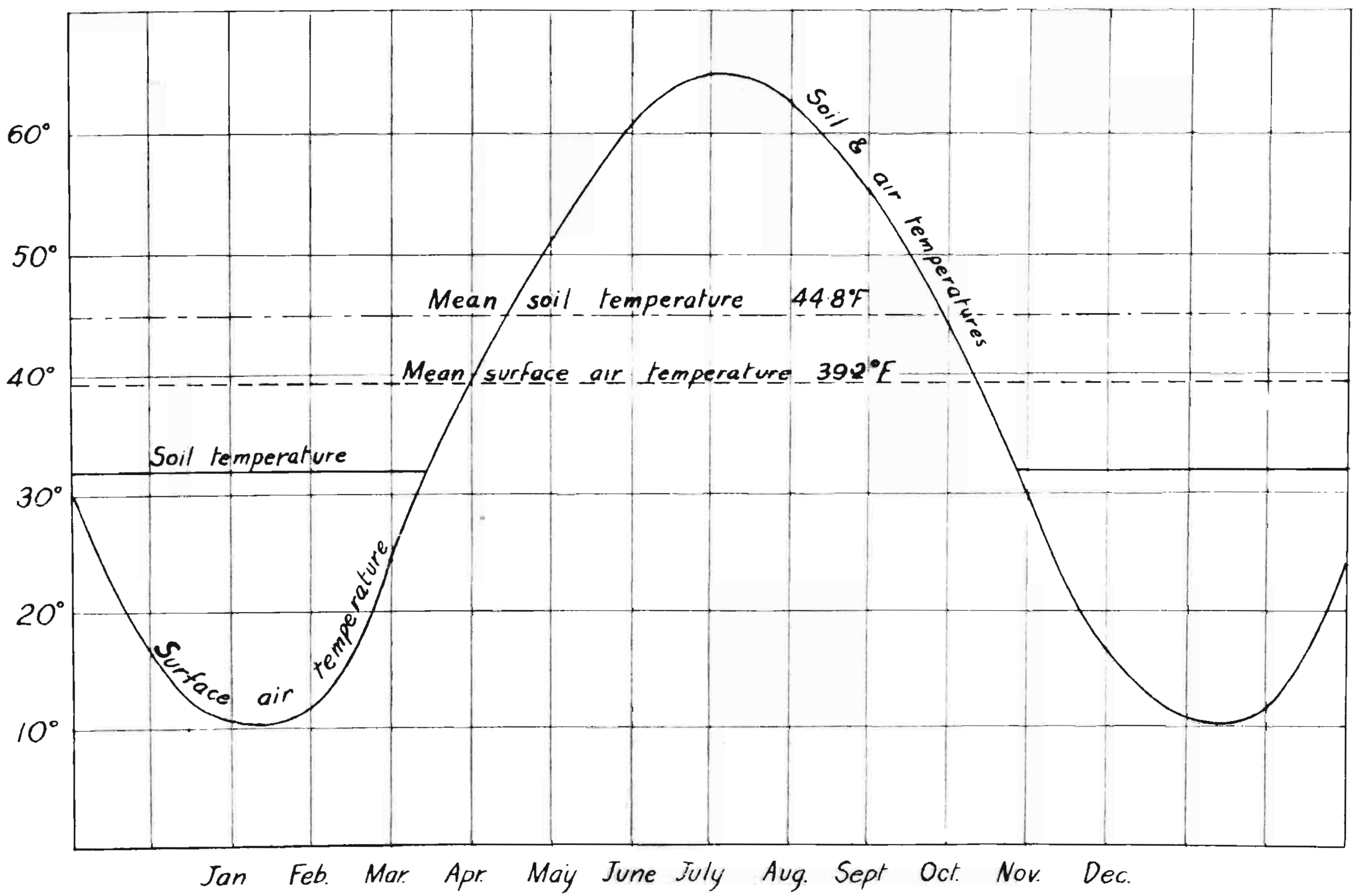
that the mean rock temperature at the surface is not the same as the mean air temperature, but differs from it by several degrees. This difference, in colder climates, is due mainly to the blanketing effect of snow, which shields the underlying rock from the colder surface air. Lane[10]

(10) Lane, A. C., Am. Inst. Min. Met. Eng. Trans. vol. 110, p. 538, 1934.

has proposed a method of calculating the mean rock temperature from the mean monthly temperatures as given by the U.S. Weather Bureau. In Lane's method of calculation the rock temperature is assumed to be the same as the surface air temperature for each month except the winter months when the air temperature falls below 32°F; during the winter months the rock temperature is assumed to remain at 32°F. Table IV shows this method applied to the computation of mean rock temperature at the Hiawatha mine.

Rock temperatures were measured directly in the No. 17 level and in the No. 18 level by inserting a thermometer in a freshly drilled hole in an advancing drift. The thermometer was placed in a grooved rod and packed with glass wool

Fig. 5 ANNUAL RANGE OF TEMPERATURE



to prevent damage through shock. An air pocket was formed around the bulb of the thermometer by inserting it into a bubble of air that had been blown from the liquid plastic known as "Plastic-loons". This air pocket appreciably slowed down the response of the thermometer to sudden changes of temperature such as would occur when the thermometer was withdrawn from the hole.

The thermometer was left in the hole for one hour after which it was quickly withdrawn and the temperature read. Measurements were made in two holes in the No. 17 level and in two holes in the No. 18 level.

The rock temperatures at the ninth and the third levels was measured by taking the temperature of the water seeping from the rock.

Results:

No. 18 level

No. 1 hole = 63.2°F
No. 2 hole = 63.8°F

No. 17 level

No. 1 hole = 61.1°F
No. 2 hole = 62.2°F

Water seepage

No. 9 level = 52.7°F
No. 3 level = 48.6°F

Mean rock temp. at surface (calculated) = 44.8°F

These temperatures, when plotted against depth, lay on a line, the gradient of which was 1°F for each 110 feet.

It is not known whether other measurements of geothermal gradient have been made in the Iron River district, but in the Michigan copper district, 95 miles north east of Iron River, the geothermal gradient has been found to be 1°F for each 129 feet[11].

(11) Fisher, James, Ingersoll, L. R., and Vivian, Harry, Recent geothermal measurements in the Michigan copper district: Am. Inst. Min. Met. Eng. Trans. vol. 110, p.532, 1934.

## REMOVAL OF HEAT

To determine the rate at which heat is being removed from the mine it is necessary to know the content of heat in the intake air and in the return air. The heat content is known as the total enthalpy and the heat content of each 1 lb. of air is known as the unit enthalpy.

The unit enthalpy of an air-vapor mixture may be defined as the amount of heat required to raise, under constant pressure conditions, 1 lb. of dry air from an assumed base temperature to the dry bulb temperature, plus the amount of heat necessary to change a weight of water at an assumed base temperature to vapor under the conditions existing in the atmosphere. The base temperature for the air was assumed to be 0°F and the base temperature for water was assumed to be 32°F. The following formula for unit enthalpy is given by Weeks[12].

(12) Weeks, W. S., Heat removed from a mine by ventilating current: Eng. and Min. Journal, June 1936.

$$E = C_p T + 0.622 \left(\frac{e}{p - e}\right) E_1$$

E is the unit enthalpy
$E_1$ is the unit enthalpy of 1 lb. of saturated vapor at the dry bulb temperature T
Cp is the specific heat of air at constant pressure
T is the dry bulb temperature in °F
p is the barometric pressure
e is the vapor pressure at the dewpoint

The unit enthalpy of air at various points in the mine is given in Table V. The unit enthalpy is plotted against depth in Figure 6. The general form of the enthalpy curve is very similar to the temperature curve as shown in Figure 4.

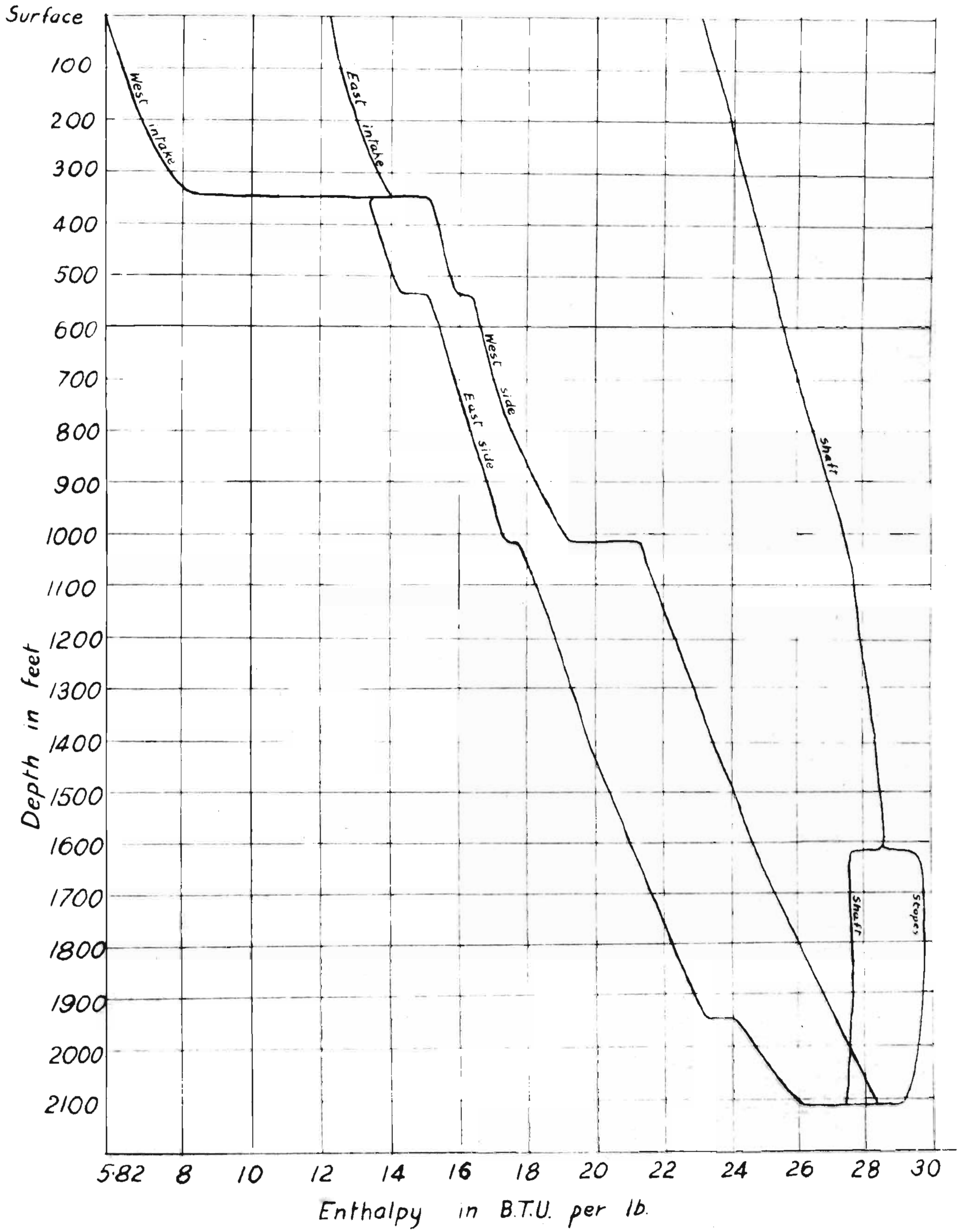
If, at any given point in the mine, the unit enthalpy, the weight of one pound of air, and the quantity of air in cfm, are known, the total enthalpy of the air may be computed. If the total enthalpy of the air entering the mine, and the total enthalpy of the air leaving the mine are known, the amount of heat being removed from the mine may be computed from the difference in total enthalpies.

The calculation of the heat being removed from the mine is given in Appendix B which shows that the heat is being removed from the mine at the rate of 35,000 B.T.U. each minute.

## SOURCES OF HEAT

There are several causes of heating of the mine air which are considered below.

# Fig. 6. UNIT ENTHALPY



Heat evolved by underground workers

McElroy[13] states that one man working at average

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(13) McElroy, G. E., Mine ventilation, Section 14, p.56: R. Peele, Mining Engineers' Handbook, 3d. ed., Wiley, 1941.

---

speed evolves heat at the rate of 17 B.T.U. per minute. As there are about 40 men working underground in each shift at the Hiawatha mine, the heat evolved by them would be about 700 B.T.U. per minute.

Heat evolved by the operation of the fans

Part of the power input to a fan is converted into useful work as air horsepower; the remainder is converted immediately into heat which is removed in the air current. But the air horsepower is expended on pressure losses, and in this way it reappears as heat which is removed by the air current.

McElroy[14] states that 1 hp is converted into heat

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(14) McElroy, G. E., op. cit. ( Mine ventilation) p.56.

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at the rate of 42.4 B.T.U. per minute. As the total power input to the fans is 30 KW which is 40 hp, the heat resulting from the operation of the fans is 1700 B.T.U. per minute.

Heat evolved by underground machinery and explosives

It was not possible to estimate the amount of heat produced by underground machinery or explosives but it is thought that the heat produced by these agencies would not be large.

Heat produced by fires and chemical reactions

The amount of heat produced could be approximately estimated if the type of reaction and the composition of the upcast air were known. In the absence of this information the heat due to these causes cannot be calculated. Because of the combustible nature of the wall rocks, the heat resulting from fires and spontaneous combustion may be considerable.

It is apparent from Figure 4 that the air from the west intake raise increases 16°F in passing from the surface to the third level. Of this rise in temperature, 1.9°F may be attributed tó adiabatic compression; the remaining 14.1°F must be due to rock heating to rock heating and chemical reactions. From Figure 4 it is apparent that rock heating does not cause more than 2°F rise in temperature in the east intake raise. In the west intake raise where the temperature difference between air and rock is about twice that of the east intake raise, rock heating would be expected to cause about twice as much rise in temperature, that is, about 4°F rise.

Thus a total of 6°F rise in temperature may be attributed to adiabatic heating and rock heating. The remaining 10°F rise in temperature results from the only other known cause of heating which is a fire at the bottom of the raise.

This illustration shows that in some parts of the mine fires and oxidation are important sources of heat.

Heat resulting from adiabatic compression and expansion

Adiabatic compression of descending air causes a rise in air temperature of about $5\frac{1}{2}$°F for each 1000 feet of descent. The sloping lines in Figure 4 are drawn to show the rate at which this rise in temperature takes place. At the same time, the exhaust air on rising up the shaft is cooled by adiabatic expansion at the same rate of $5\frac{1}{2}$°F for each 1000 feet of ascent.

Inasmuch as the collar of the intake raises is at the same elevation as the collar of the shaft, the heating due to adiabatic compression will be balanced by cooling due to adiabatic expansion, and none of the heat removed from the mine is attributable to adiabatic effects.

Heating of air due to rock

As soon as cooler air passes through a freshly exposed airway, the air is heated and the rock is cooled. The cooled zone of rock extends inward from the rock surface until a temperature gradient has been established after which the amount of heat passing from the rock to the air is practically constant.

Formulae, which are based on mathematical reasoning, have been developed for the purpose of calculating the amount of heat flowing from the rock into the air but in order to apply these formulae, the density, conductivity, specific heat, and diffusivity of the mine rock must be known. Because of the heterogeneous nature of most mine rocks, values for these factors can not be more than approximately estimated, consequently the application of these formulae to conditions in a mine is subject to a large degree of approximation.

The application of these formulae is, therefore, restricted to comparison of the amounts of heat flowing into airways in the same type of rock and where even an approximate result is useful.

In Appendix C, the time taken to cool the rock in one of the mine airways is calculated by using formulae that have been derived by mathematical reasoning. The calculated time of 11½ hours may be several times, or it may be only a fraction of the time actually taken to cool the rock in the airway; nevertheless the result shows that equilibrium conditions in a mine airway are reached much more rapidly than is commonly believed.

Because all the mine airways have been open for several years, it is considered that, if the surface air temperature is constant, the amount of heat flowing from the rock will be constant.

It is clearly impossible to evaluate accurately the amount of heat flowing from the rock; rock heating would be expected to contribute the portion of the total heat removed that is not attributable to other causes of heat.

## CONCLUSIONS

The survey showed that the total quantity of air reaching the working places is 10,000 cfm less than the quantity that is considered necessary to establish good ventilation throughout the mine. Improvement of the ventilation of the mine can be achieved by reducing leakage in the intake airways, by preventing recirculation, and by increasing the quantity of intake air.

Because of the indirect passage of the intake air to the working places, some leakage of air from the intake airways is inevitable. Although no single source of leakage accounts for more than 2000 cfm, the total quantity of air lost through leakage is considerable. This leakage could be reduced by:

1. Sealing the bulkhead on the west side of the No. 12 level through which 2000 cfm leaks from the west side airways.

2. Repairing the air doors on the No. 16 level east, through which 1000 cfm pass to the Hiawatha No. 2 mine.

3. Sealing off the main east side raise from the No. 15 level.

The introduction of 4500 cfm of foul upcast air into the fresh air current at the fifth level should be prevented. The consumption of power by the main fans would be reduced and the entire ventilation system improved by placing air doors on the fifth level to prevent this air from reaching the intake airway.

Recirculation occurs in one of the stopes between the No. 17 and the No. 18 levels. It is possible that a certain amount of recirculation occurs in other stopes between the No. 17 and the No. 18 levels because the practice of ventilating stopes by forcing air down from the No. 17 level tends to induce recirculation.

Better stope ventilation would be attained if the east side air course could be taken directly to the No. 18 level from which all stope air could be drawn.

It is advisable to select auxiliary fans in accordance with their probable duty if their initial cost and the cost of power are to be kept at a minimum. The fans that are used for auxiliary ventilation are Coppus TM 6 fans which are essentially high pressure fans and these are not suitable for the low pressure requirements of most stope ventilation. For the ventilation of stopes where the total length of tubing is not more than 350 feet it would be better to use low pressure fans such as the Buffalo vaneaxial type B, size 18 or the Joy model I-9. Both of these fans are capable of supplying 4000 cfm through 150 feet of tubing at a power input of 1.8 hp.

The main east side fan is inadequate to deliver the required amount of air, but as it is probable that stoping will become concentrated on the west side, it does not seem economic to make any major alterations to the east side circuit.

In the west side circuit the demand for air will no doubt increase when stoping becomes concentrated on this side and the ventilation system should be planned to anticipate this increased demand.

It is desirable to remove the Coppus TM 8 fan from the ninth level because it is unsuitable for parallel operation with the American Blower fan. The American Blower fan, when operating alone should be capable of supplying 18000 cfm at a power input of not more than 10 KW. Although the removal of the Coppus TM 8 fan would reduce the power input by 10 KW, the quantity of air supplied to the west side circuit would be reduced by 3000 cfm. Therefore this step should be taken only after the quantity of air in the west side circuit has been increased by placing a fan at the top of the 30 in. diameter drill hole to force air down to the ninth level.

The fan at the top of the drill hole should have a capacity of 10,000 cfm at a static pressure of 4½ in. water gage. A fan suitable for this purpose would be the Buffalo " Limit Load", single width, size 6½A. This is a centrifugal fan which will deliver the required rating at 1200 rpm and 10 hp input. These two steps would result in a net increase of 7000 cfm of fresh air supplied to the west side circuit at a saving in power input of 3½ hp.

The data obtained during the survey showed that the cooling power of the air is adequate in all parts of the mine.

Natural ventilation pressure, which is 0.7 in water gage in winter, would be negligible in summer; consequently the total quantity of air passing through the mine in the summer months would be 9000 cfm less than in winter.

Although their separate evaluation is difficult, it is probable that fires, oxidation, and rock heating are the sources of most of the heat removed from the mine.

Adiabatic effects are the chief causes of change of air temperature withim the mine, but they do not contribute to the amount of heat that is removed from the mine by the air current.

## SUMMARY

During a field study of the ventilation of the Hiawatha mine, Iron River Michigan, data were assembled showing the distribution of air to each part of the mine and the changes in temperature and moisture content of the air in its passage through the mine. Consideration of the distribution of the air indicated that ventilation could be improved by eliminating leakage and by preventing recirculation.

The static pressure produced by each fan was measured and the natural ventilation pressure was computed from the air temperatures and barometric pressures.

Friction losses in each airway were calculated and tabulated to facilitate the computation of the pressure potential necessary to increase the amount of air by a given amount.

The temperature of the rock at various depths was measured and plotted against depth to obtain a value for the geothermal gradient of the mine.

The wet bulb temperatures and the dry bulb temperatures were interpreted as far as possible to ascertain the causes of transfer of heat to the ventilation current and the evaluate the importance of each cause.

# APPENDIX A

Table I : Moisture content of Mine air

| Location | Rel. Hum. % | Dry Bulb in$^{o}$F | Wet Bulb in$^{o}$F | Dew-point in$^{o}$F | Grns. water p.c.f. | Vap. press. in in. |
|---|---|---|---|---|---|---|
| surface | 75 | 17 | 15.4 | 11 | .810 | .066 |
| | bottom of intake raises(3rd level) | | | | | |
| west raise | 80 | 33 | 31 | 28 | 1.755 | .148 |
| east raise | 80 | 36 | 34 | 31 | 1.94 | .166 |
| | east side airways | | | | | |
| 5 level | 92 | 39 | 38 | 37 | 2.84 | .218 |
| 9 level | 93 | 50.5 | 49.5 | 48.5 | 3.807 | .341 |
| 17 level | 94 | 58 | 57.5 | 56 | 5.106 | .448 |
| 18 level | 94 | 62 | 60 | 60 | 5.703 | .517 |
| | west side airways | | | | | |
| 9 level east | 93 | 50 | 49 | 48 | 3.79 | .334 |
| 9 level west | 91 | 53 | 51.5 | 50 | 4.175 | .361 |
| 17 level | 94 | 62.5 | 61.5 | 60.5 | 5.751 | .523 |
| 18 level | 94 | 65 | 64 | 63 | 6.452 | .575 |
| | stope air | | | | | |
| 13 level | 97 | 66.5 | 66 | 65.5 | 6.931 | .627 |
| | upcast shaft | | | | | |
| 18 level | 95 | 63.5 | 62.5 | 61.5 | 6.102 | .545 |
| 17 level | 100 | 63 | saturated | | 6.349 | .575 |
| 16 level | 100 | 62 | saturated | | 6.142 | .555 |
| 13 level | 100 | 63 | saturated | | 6.246 | .565 |
| 9 level | 100 | 60.5 | saturated | | 5.746 | .522 |
| collar | 100 | 53 | saturated | | 4.526 | .402 |

Table II: Computation of natural ventilation pressure

## Intake airways

| East side: | Length in feet | Weight in lbs./cu.ft. | Total wt. in lbs. |
|---|---|---|---|
| above intake(west) | 85 | .07923 | 6.72 |
| east raise (3rd-9th) | 688 | .07626 | 52.4 |
| east raise (9th-17th) | 898 | .07662 | 68.8 |
| east raise(17th-18th) | 175 | .07723 | 13.5 |
| west intake raise | 345 | .07757 | 26.8 |
| | | TOTAL | 168.22 |

| West side: | | | |
|---|---|---|---|
| above east intake | 110 | .07923 | 8.7 |
| east intake | 320 | .07694 | 24.6 |
| stope(3rd-9th) | 688 | .07621 | 52.5 |
| west raise(9th-12th) | 417 | .07603 | 31.7 |
| west raise(12th-17th) | 481 | .07652 | 36.8 |
| west raise(17th-18th) | 175 | .07705 | 13.5 |
| | | TOTAL | 167.8 |

Because about twice as much air passes through the west side airways as through the east side airways, a weighted average of the two columns of air must be taken.

$$\text{Weighted average } \frac{(168.2 \times 1)(167.8 \times 2)}{3}$$

$$= 167.93 \text{ lbs.}$$

| Upcast shaft: | | | |
|---|---|---|---|
| 18th-17th | 175 | .07715 | 13.5 |
| 17th-13th | 326 | .07651 | 24.9 |
| 13th-9th | 572 | .07528 | 43.1 |
| 9th-5th | 506 | .07431 | 37.6 |
| 5th to top headframe | 612 | .07363 | 45.1 |
| | | TOTAL | 164.1 |

$$\text{Natural ventilation pressure} = (167.93 - 164.1) \text{ lbs/sq.ft}$$

$$= 3.83 \text{ lbs. per sq. ft.}$$

$$= .73 \text{ in. water gage}$$

Table III Calculated friction loss

| Airway | | Length | k X$10^{10}$ | Friction loss in in. w.g. |
|---|---|---|---|---|
| West intake raise | | 345 | 100 | .038 |
| East intake raise | | 320 | 100 | .25 |
| 3rd level | | 2400 | 165 | .21 |
| 7th level | | 800 | 165 | .002 |
| 5th level | | 1900 | 165 | .034 |
| West raise(9th-12th) | | 480 | 110 | .18 |
| West raise (12th-17th) | | 420 | 100 | .05 |
| 17th level | | 900 | 165 | .008 |
| 18th level | | 1100 | 165 | .01 |
| East raise (3rd-9th) | | 800 | 110 | .29 |
| East raise (9th-11th) | | 280 | 110 | .18 |
| East raise (11th-17th) | | 640 | 110 | .39 |
| 9th level | | 3200 | 175 | .93 |
| Upcast shaft | dimensions | | | |
| 18th-17th | 5'10" x 18½' | 175 | 110 | .009 |
| 17th-13th | 5'10" x 13' | 325 | 110 | .036 |
| 13th-6th | 5'10" x 13' | 944 | 110 | .265 |
| 6th-collar | 5'10" x 9' | 661 | 110 | .320 |
| | | Total friction loss | | 3.202" w.g |

Dimensions of airways: All levels are 8 feet by 8 feet. All raises except the ones noted below are 4 feet by 5 feet. East intake raise is 4 feet by 5½ feet. West intake raise is 6 feet by 7 feet. West side airway between the ninth and the 12th levels consists of two raises each 4½ feet by 5½ feet.

Table IV Computation of mean soil temperature

| Month | Mean air temp. in $^{o}F$ | Mean soil temp. in $^{o}F$ |
|---|---|---|
| January | 10.6 | 32 |
| February | 11.7 | 32 |
| March | 24.2 | 32 |
| April | 39.1 | 39.1 |
| May | 51.2 | 51.2 |
| June | 60.6 | 60.6 |
| July | 65.1 | 65.1 |
| August | 62.5 | 62.5 |
| September | 55.3 | 55.3 |
| October | 44.0 | 44.0 |
| November | 30.0 | 32 |
| December | 16.7 | 32 |
| TOTALS | 471.0. | 537.8 |

Mean air temperature is $\frac{471.0}{12}$ or $39.2^{o}F$

Mean soil temperature is $\frac{537.8}{12}$ or $44.8^{o}F$

Table V Unit enthalpy of 1 lb. of air

| Location | Bar. press. | Temp.°F | Enthalpy in B.T.U |
|---|---|---|---|
| surface (average) | 28.46 in. | 17 | 5.82 |
| **Bottom of intake raises(3rd. level)** | | | |
| west raise | 28.73in. | 33 | 13.13 |
| east raise | 28.73in. | 36 | 14.06 |
| **East side airways** | | | |
| 5th level | 28.99in. | 39 | 14.98 |
| 9th level | 29.37 in. | 50.5 | 17.63 |
| 17th level | 30.50 in. | 58 | 23.87 |
| 18th level | 30.70 in. | 62 | 26.60 |
| **West side airways** | | | |
| 9th level east | 29.37 in. | 50 | 19.46 |
| 9th level west | 29.46 in. | 53 | 20.97 |
| 17th level | 30.49 in. | 62.5 | 26.32 |
| 18th level | 30.70 in. | 65 | 28.31 |
| **Stope air** | | | |
| 13th level | 30.18 in. | 66.5 | 29.48 |
| **Upcast shaft** | | | |
| 18th level | 30.70 in. | 63.5 | 27.43 |
| 17th level | 30.48 in. | 63 | 27.74 |
| 16th level | 30.34 in. | 62 | 27.69 |
| 13th level | 30.18 in. | 62.5 | 28.61 |
| 9th level | 29.43 in. | 60.5 | 27.29 |
| collar | 28.46 in. | 53 | 23.09 |

## APPENDIX B

Calculation of the amount of heat removed from the mine

A total quantity of 26,500 cfm enters the mine at the surface through two intake raises and 25,500 cfm leaves the mine through the upcast shaft and the remaining 1000cfm passes to the Hiawatha No. 2 mine at the No. 16 level.

By calculating the unit enthalpy, the specific weight, and the quantity of air entering the mine, we may obtain the total enthalpy, or the total heat content, of the air entering the mine. In the same way, the total heat content of the air leaving the mine may be calculated. If the total heat content of the air entering the mine is subtracted from the total heat content of the air leaving the mine, the amount of heat being removed from the mine is obtained.

The unit enthalpy was calculated from the formula given on p.27.

$$E = Cp\ T + 0.622\left(\frac{e}{p - e}\right)E_1$$

$E_1$ and Cp were obtained from Goodman's tables[15]

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(15) Goodman, William, Air conditioning analyses, Macmillan, 1943.

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Dewpoint, relative humidity, vapor pressure, and specific weight of air were obtained from Jeppe's tables (ref. p.8) Wet bulb temperatures, dry bulb temperatures, barometric pressures and quantities of air were measured directly.

At the intake

Data:

| | |
|---|---|
| Average dry bulb temperature | $= 17^{o}F$ |
| Average wet bulb temperature | $= 15.4^{o}F$ |
| Barometric pressure | = 28.46 in. |
| Quantity of air | = 26,500 cfm |

From tables:

| | |
|---|---|
| Dewpoint | $= 11^{o}F$ |
| Relative humidity | = 75% |
| Vapor pressure | = 0.0665 in. |
| Specific weight of air | $= 0.07923$ lbs/foot$^3$ |
| $E_1$ | = 1069 |

Unit enthalpy in B.T.U. per lb.

$$= (0.241)(17) + 0.622\left(\frac{(0.0665)(1069)}{28.46 - .0665}\right)$$

$$= 5.82$$

Total enthalpy

= 5.82 X 26500 X 0.07923 B.T.U./minute

= 12,200 B.T.U. per minute

At the upcast collar

Data:

| | |
|---|---|
| Dry bulb temperature | $= 54^{o}F$ |
| Wet bulb temperature | $= 54^{o}F$ |
| Dewpoint | $= 54^{o}F$ |
| Relative humidity | = 100% |
| Barometric pressure | = 28.46 in. |
| Quantity of air | = 25,500 cfm |

From tables:

| | |
|---|---|
| Vapor pressure | = 0.417 in. |
| Specific weight of air | $= 0.0733$ lbs/ foot$^3$ |
| $E_1$ | = 1085 |

Unit enthalpy in B.T.U. per lb.

$$= (0.241)(54) + 0.622\left(\frac{(0.417)(1085)}{28.46 - .417}\right)$$

$$= 23.09$$

Total enthalpy

= 23.09 X 25,500 X 0.0733 B.T.U./ minute

= 43,200 B.T.U. per minute

<u>At the No. 16 level</u> (where air passes to the Hiawathaa No.2 mine)

Data:

dry bulb temperature = 58°F
wet bulb temperature = 57°F
barometric pressure = 30.33 in.
quantity of air = 1000 cfm

From tables:

dewpoint = 56°F
relative humidity = 94%
vapor pressure = 0.465 in.
specific weight of air = 0.0766 lbs/ foot$^3$

Unit enthalpy in B.T.U. per lb.

$$= (0.240)(58) + 0.622\left(\frac{(0.465)(1086)}{30.33 - 0.465}\right)$$

$$= 25.89 \text{ B.T.U. per lb. of air}$$

Total enthalpy

= 25.89 X 1000 X 0.0766 B.T.U./minute
= 1980 B.T.U. per minute

The total enthalpy of the air entering the mine is

12,200 B.T.U. per minute.

The total enthalpy of the air leaving the mine is

43,200 + 1980 B.T.U. per minute

or 45,180 B.T.U. per minute

Therefore the heat removed from the mine by the air current

is 45,180 − 12,200 B.T.U. per minute

or 32,980 B.T.U. per minute

## APPENDIX C

Cooling time for freshly exposed rock

If the amount of heat flowing from the rock to the air, and the difference between the air temperature and the temperature of the uncooled rock are known, the time taken to establish equilibrium conditions may be calculated.

The data taken in the survey for the east side airway between the ninth level and the No. 17 level are used in calculating the time taken to establish equilibrium conditions in the airway after it was frehly exposed to the flow of air. It is assumed that equilibrium conditions had been established at the time of the survey.

The temperature of the rock when it was frehly exposed would be the same as the rock temperature at that depth, which may be obtained from the geothermal gradient curve in Figure 4.

To simplify the calculations the figures are converted into metric units.

The followingvalues are taken as fair averages for sedimentary rock:

Density = 2.75 grams per cc.
Conductivity = 0.005 gm. cals. per sec. per sq. cm. per $^{o}$C diff. in temperature per cm. thickness
Specific heat = 0.22 gm. cals. per gm. per $^{o}$C.
Diffusivity = 0.0083

Length of airway = 898 feet
Perimeter = 36 feet
Exposed surface = 36 X 898 sq. feet
= $3.00 \times 10^{7}$ sq. cm.

Average air circulation $= 10{,}000$ cfm
Specific weight of air $= 0.076$ lbs/foot$^3$
Weight of air circulated $= 10{,}000 \times 0.076$ lbs/min.

$= 5.8 \times 10^3$ gms per sec.

Air quality at the ninth level
dry bulb temperature $= 45.5^{\circ}F$
wet bulb temperature $= 44.5^{\circ}F$
moisture content $= 3.24$ grains per cub. foot

Air quality at the No. 17 level
dry bulb temperature $= 57.5^{\circ}F$
wet bulb temperature $= 56.5^{\circ}F$
moisture content $= 5.02$ grains per cub. foot

The temperature of the air increases by $12^{\circ}F$ between the ninth level and the No. 17 level; of this increase, $6.1^{\circ}F$ is due to adiabatic compression, therefore $5.9^{\circ}F$ or $3.3^{\circ}C$ is increase due to rock heating which is the only other known cause of heating.

Therefore gain in sensible heat in passing from the ninth level to the No. 17 level is equal to

$$5.8 \times 10^3 \times 0.241 \times 3.3 \text{ gm. cals. per sec.}$$

which if distributed over the rock surface will set up a heat current equal to

$$\frac{5.8 \times 10^3 \times 0.241 \times 3.3}{3.0 \times 10^7} \text{ gm. cals. per sec./sq.cm}$$

$$\text{or } 1.54 \times 10^{-4} \text{ gm. cals. per sec. per sq. cm.}$$

The moisture content of the air increases by 1.78 grains per cubic foot of air between the ninth level and the No. 17 level.

Total gain in moisture content per second is equal to

$\dfrac{1.78 \times 10{,}000}{60}$ grains which is converted into latent heat at the rate of 10,330 gm. cals. per second.

If this latent heat is distributed over the rock surface the heat current set up will be $3.44 \times 10^{-4}$

The total heat current will be the sum of the heat currents set up by latent heat and by sensible heat.

Total heat current $= (3.44 - 1.54)\, 10^{-4} = 4.98 \times 10^{-4}$

From Figure 4, it will be apparent that the average difference in temperature btween air and uncooled rock is 5.9°F or 3.3°C.

Considering the surface exposed as afalat surface boundary of an infinite mass of rock, the temperature of which differs from the air temperature by T°C and with the heat being absorbed by the air current as soon as it is released from the rock, it will be seen that for equilibrium conditions to become established, the temperature of the rock surface will approximate that of the air.

The rock temperature ~~in~~ from the surface will rise in accordance with the following equation which may be obtained from any standard textbook on the conduction of heat.

$$R = \frac{T}{\sqrt{\pi d t}}$$

R is the rate of temperature rise with distance from the rock surface
d is the thermal diffusivity of the rock
t is the number of seconds that have elapsed since cooling began

It has been computed that the cooling effect of the air on the rock is equal to

$4.98 \times 10^{-4}$ gm. cals. per sq. cm per sec.

If this cooling effect is divided by the conductivity, the quotient obtained is equal to R the rate of increase of rock temperature with depth.

$$R = \frac{4.98 \times 10^{-4}}{0.005} = 0.1^{\circ}C \text{ per cm. depth}$$

Equating this result to the temperature gradient as given by the above formula

$$0.1 = \frac{3.3}{\sqrt{\pi\, 0.0083t}}$$

Therefore $t = 41{,}840$ seconds

$= 11\frac{1}{2}$ hours.

Although this calculation is subject to a large degree of approximation, the result shows that equilibrium is reached in a matter of hours.

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# VITA

Thomas Alan O'Hara was born at Richmond, Nelson, New Zealand in 1923. He was educated at Brightwater School and Nelson College at which he passed the University Entrance Examination in 1939. He remained at Nelson College until 1941 when he secured a Credit Pass in the National University Scholarships examination after which he entered the University of New Zealand to study Science.

After 1943, he studied Mining Engineering concurrently with Science until 1945 when he completed the B.Sc. degree, advancing in Geology and Applied Chemistry. He completed the B.E. (Mining) in November, 1946 after which he joined the engineering staff of The Consolidated Mining and Smelting Company of Canada at their Sullivan mine, Kimberley, British Columbia.

He left Kimberley in August, 1947 to take up a Research Fellowship in Mining Engineering at Missouri School of Mines and Metallurgy, Rolla, Missouri.