THE DEVELOPMENT, MINING AND CAVING OF THE EAST OREBODY—BLUEBERRY MINE
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DEVELOPMENT

History—
A short time after the North Range Mining Company took over the operation of the Blueberry mine seven miles West of Ishpeming, it became apparent that the orebody on the East end of the mine showed indications of extending to a greater height than had been expected from the general trend of the top of the orebody in that area.

The development work had reached a point as shown in Figure 1 on January 1st, 1934, and it seemed then, that the top of the ore had been reached. It was later to be discovered that this orebody extended nearly to the sand and gravel. This article deals with the developing, mining and subsequent caving to surface of this area.

Occurrence—
The orebody occurs as a thin, lenticular body, approximately 50 feet in average width and 300 feet in average length. On the 346 sub-level, however, the lateral extent of the orebody is actually a little over 400 feet, the East end of which thins out to a narrow width of ore of four or five feet.

The walls consist of a soft ore jasper. On the East end, a large diabase dike cuts the iron formation at a small angle, striking Northeast and Southwest. The ore is found concentrated immediately on the dike in this vicinity. The footwall proper is the Siamo slate series. The ore is not found directly on the slate, however, being separated from it by a layer of soft ore jasper.

Concentration has not been effected by the dike forming at impervious strata, but appears to have been a result of leaching and enrichment along the area of a cross fault, which allowed waters to seep and penetrate the formation. This occurrence in the vicinity of cross faults is typical of the entire Blueberry ore-body.

The East orebody, so-called, is really not a separate, distinct orebody, but rather an extension, nearly to surface, of the main deposit on the East end of the mine.

Plan of Development—
Because of the great distance from the East Orebody near surface to the main tramming level 700 feet below the collar of the shaft, it was decided to stope the upper portion of this new ore-body and transfer the broken ore through long raises a distance of 350 feet to the 700 level.
The main scraping or transfer sub-level was chosen at the 346 elevation and the development of the stope area above this elevation was begun as soon as the main raises reached this point. The transfer sub-level was driven from East to West in a straight line, a distance of about 400 feet and encountered jasper on both ends of the drift.

Two diamond drill holes put down by the North Range Mining Company in 1934 to determine the vertical extent of this new ore, gave the information which was used to determine the final plan of stoping.

**Raises and Sub-Levels—**

A small dog-raise was put up about 200 feet high to the jasper capping, directly in the middle of the orebody. This was used for a travelway to the stopes on the East and West side of this raise.

In order to open up each stope area, a cutting-in raise was put up about 200 feet, to the 146 sub-level, for the end travel ways to the stopes. These raises were about 200 feet away from the central ladderway.

In order to open up each stope area, a cutting-in raise was put up in the middle of each stoping block. This was done so as to retreat, in mining, from the middle to the travelways on each end the stoping block, thereby permitting miners to work on both sides of the stope.

The sub-levels were driven at 20 feet vertical intervals throughout the length of the East orebody. Ten sub-levels were driven above the main scraping sub on the 346 elevation. The height of the stope from the top sub, to the top of the mills on the first sub, was thus 180 feet.

Mill raises, 15 feet apart, were put up from the North side of the scraping sub to the next 20-foot sub above, commonly called the mill sub. It was not necessary, because of the narrow width of the orebody, to put mill raises on both sides of the scraping drift. It was found by testing, also, that the scraping sub was quite close to the hanging wall of the orebody and that mill raises on that side of the drift would not be practical.

**Ventilation Shaft—**

Early in the development of the East orebody it became apparent that some outlet would have to be provided in that area for handling tools, materials, and improving ventilation. The time required to hoist materials from the 700 level to the 346 scraping sub, and then up an additional 200 feet to the working places in the stopes; the long climb for the men; the long time required for smoke to dissipate itself after blasting, made a second outlet a necessary objective.

A crosscut was driven North 135 feet from the West end of the 346 scraping sub, through jasper and footwall slate. A small ventilation shaft, 6½ feet by 12 feet, with ladderway and cage compartments, was raised to connect with a drainage shaft of the same size, put down previously in that area.

The drainage shaft was down 65 feet and small drifts driven from it just below ledge to catch the water making over the East orebody.

The ventilation shaft, when completed, improved conditions materially and provided easy access to the East orebody, as is shown in Figure 2, showing in section the fully developed area preliminary to stoping operations.

**MINING**

**Method of Stoping—**

The method of mining employed was sub-level stoping, such as is used in some mines in the Marquette district and in the Iron River district.

Benches were 20 feet apart and the working stope face was kept nearly vertical. No overhanging benches were permitted due to the nature of the ore.

In starting a bench, a six-foot slice about eight feet high was driven along the stope face to the footwall and the hanging wall. The back of the bench was then taken down with upper holes and worked back in this fashion to the dog drift opening to the stope. Each bench was approximately 50 feet long, which corresponded to the width of the orebody. The sub-level drift leading to the stope was approximately in the center of the orebody resulting in a bench about 25 feet long on each side of the sun drift and at right angles to it.

As has been stated before, each stope block was 200 feet high, with ten sub-drifts or working benches 20 feet apart. In mining, a slice was completed on the top bench, then the miners dropped to the next bench and so on until they had reached the bottom bench. They then moved up again to the top sub and went through the same procedure. In this way, a nearly vertical face for the stope was obtained. All broken ore sliced away on the benches fell free to the mills below and dropped through these openings, which were funnelled around the top, to the scraping sub, where it was transferred by scraper and trigger mechanism to the main cribbed raises leading to the 700 level.

**Scraping and Breaking—**

The broken ore transferred on the 346 scraping sub was handled by standard hoe-type scrapers, 42 inches wide, and pulled by 15 horsepower, electric, double-drum triggers mounted at the main raises.

Two such raises, double-compartment, 120 feet apart, handled all the ore to the 700 level. Branches to the East and West from these main raises, without timber, were put up to shorten the haul from the extremities of the orebody. No scraping distance was more than 150 feet.

The Blueberry ore is a semi-hard hematite and large chunks resulted in breaking ore in the stopes. These chunks frequently had to be blasted from the throat of the mills, from below by the scrapermen, and sometimes...
drilled and blasted clue to their large size. In addition to the larger chunks coming from the stopes, smaller fragments were too large to pass through the grizzly rails, set one foot apart, on the top of each main transfer raise. Breaking these smaller chunks by hand with sledge hammers was slow and tedious, so the older, worn out auger drilling machines, without rotation, and fitted with moils, were introduced with great success, and incidentally reduced the danger of flying fragments of ore when sledges were used.

Raise Lining—
A question which arose was how to maintain the cribbed transfer raises under continual ore passing. These raises, as has been said, were double-compartment cribbed raises, one compartment for a ladderway, the other for the ore pass.

Again, these raises extended from the 700 level to the 346 sub, a distance of 354 feet. The abrasion of the ore on the chute compartment was very severe due to the hardness of the ore. Two-inch hardwood planks were used to cover the four sides of the ore mass. The repair work on this planking necessitated maintaining weekend repair crews continually, and in some cases severe breakdowns interrupted production for a few days.

To remedy this situation, 20-lb. and 30-lb. rails were drilled in the bottom flange, or sole, and nailed side by side with six-inch spikes on the hardwood planking, and in some places directly on the cribbing, on all four sides of the ore pass for the full length of the raise. This was a costly and slow job to complete, being done on the week-ends only. However, after completing this work in both main raises, troubles were over. Occasionally a rail would get loose, but the repair work did not interrupt production and was reduced to a minimum.

Handling Ore At Chutes—
After the broken ore was transferred to the 700 level, the material had to be taken out of the chutes and trammed to the main hoisting shaft.

A considerable amount of water worked its way into the main raises when the ore was scraped on the 346 sub and from seepage throughout the length of the raises.

It was discovered after many trials that ordinary three-inch hardwood stopper boards on the chute openings would bulge and buckle like so much cardboard, sometimes breaking entirely and allowing the wet ore to flow onto the floor of the level in the so-called “jackpot” manner.

Sometimes the main level drift near these long raises was blocked for 60 feet or more on each side of the chutes with the run dirt, which had to be laboriously shovelled into cars by hand.

To relieve this situation, one-quarter pan discs, made of three-quarter-inch plate, were installed on each chute and, to facilitate the opening and closing of the discs, air operated cylinders, eight inches in diameter, were mounted horizontally over the trolley wire on platforms and connected to these discs by a jointed toggle arm. The cylinder and disc unit was operated by a lever which was moved a short distance away from the chute for safety. The “jackpot” condition was thereby solved and has given no further trouble.

CAVING

Conditions of Backs—
As can be seen by an examination of Figure 3, which show the stope areas on each side of the central pillar mined out, mining had progressed until the backs of the stopes were quite close to the overlying sand and gravel.

It was necessary to commence stoping operations below these old workings, but the fact that they remained open to such a height, was a potential hazard.

The backs of the stopes were a hard jasper and lean ore. Nothing fell from these backs, but in time, pieces were bound to slough off the walls. It was feared that a large fall of ground would probably crush the new workings below. There remained, too, the central pillar to mine out above the 346 sub, and a fall of ground of large size would undoubtedly, from that height, crush the 346 scraping sub-drift and render it useless for scraping.

Filling Bottoms—
In order to prevent as much as possible any heavy fall of ground from endangering the scraping sub, lean material was allowed to accumulate in a layer on the stope bottom when it fell from the walls. This layer of broken material attained a thickness of 60 feet or more.

Inducing Caving By Blasting—
An attempt was then made to induce caving the back of the East stope to surface.

A Waugh Turbo, Model 34, long hole drill was set up on the 146 sub and inclined holes drilled up into the jasper back, as are shown in Figure 3.

Four long holes were thus drilled to an average depth of 70 feet. These holes were loaded full with 60 per cent gelatin powder.

Besides this, an 800-lb. charge of 60 per cent gelatin powder was placed in the extreme end of the drainage drift on the 65-foot level. (See Figure 3.) Both places were then shot. The resulting blast had no apparent effect on the back of the East stope. No fall of ground occurred.

A churn drill was then placed on surface above the stope area and seven churn drill holes, six inches in diameter, put down. The first hole was drilled through into the back of the stope at 123 feet, the first 30 feet of which was sand and gravel, leaving 93 feet of iron formation capping over the open stope. This hole Bras drilled through in order to know exactly how deep to drill the remaining six holes to leave a 20-foot burden on the bottom of each hole.
The six holes were drilled 15 feet apart on the circumference of a circle 30 feet in diameter and one hole was drilled in the center.

The total explosive charge for the seven holes was 6,200 lbs. of 40 and 60 per cent gelatin powder. The shot was fired with Cordeau Bickford fuse.

After the blast, which occurred on Saturday night, May 30th, 1936, no evidence of a break through to surface was evident. The casing pipes through surface in the drill holes had jumped up a little, but there was no yawning cavity in the ground.

Underground conditions had not changed appreciably. Some ground had fallen from the back but the miners were able to resume work in the area and no additional material fell.

**Caving In**—

After waiting a few days, it was decided that the ground was too solid and too thick to break through to surface, and that tip central pillar could be mined without filling the stope on the East side of it.

Consequently the top subs of the pillar were sliced and robbed away, the broken ore mostly falling by direction into the West stope. Four subs in the central pillar were thus mined out, slicing the pillar as thin as possible, drilling a number of holes in what ore remained, and shooting the entire group of holes together.

Exactly one month after the churn drill holes had been blasted, on June 30th, 1936, the back of the East stope began to work slowly and started caving in, starting at about 5:30 p. m. and finishing at about 12:00 p. m. Men underground on the 346 scraping sub at the time of the cave-in were hardly aware of it. On surface, however, vibration after vibration was felt as large fragments kept tumbling into the cave. The cushion of broken lean ore in the stope bottoms caught the first, sudden blow and the rest was a gradual crumbling away of the edges which allowed the surface sand and gravel to run in.

A cavity on surface, 120 feet long by 60 feet wide was made. The East stope underground was completely filled up. The West stope was not filled quite to the top due to the flat slope of the jasper back, as seen in Figure 3.

had to be blasted from the threat of the mills, from below by the

The ground caved in the exact location of the circle of churn drill holes; no damage was done underground by wind; no timber was crushed or openings squeezed in by the blow of the cavern.

The cave-in occurred between the day and night shift work, and the latter shift was able to go underground for their regular work an hour and a half later than regularly accustomed.

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**THE TESTING OF METALLURGICAL FURNACES AND COMBUSTION CONTROL**

**BY C. T. EDDY** and **R. J. MARCOTTE**

**ABSTRACT**

Methods of collecting the necessary data incident to the calculation of material and energy balances and the use of these data in a study of the operation of metallurgical furnaces are presented. Factors influencing the adaptability of such material to studies of this kind and the accuracy of the observations required are also presented. The subject of furnace control is briefly taken up from the standpoint of combustion, and the use of the thermal data in determining furnace efficiencies and discovering possible lines of improvement is discussed. To illustrate the principles involved, a specific example of a material and energy balance for a complete operating cycle of a 120,000-pound reverberatory; copper smelting and refining furnace is given, and the various efficiencies discussed are calculated for the test.

This paper has developed as the result of questions and discussions arising in connection with the use of thermal and material balances in the study of the operation of reverberatory furnaces. These discussions and the data collected for the calculation of the balances have often revealed the possibility of desirable changes in both the metallurgical and economic phases of metallurgical operations. The purpose of this paper is to present a discussion of several methods of collecting data primarily intended for studies of this kind and to indicate lines along which such studies might be directed.

The calculations of energy and material balances and of theoretical flame temperatures and their correlation with other operating data, together with the application of more precise methods of furnace control are becoming increasingly important as the capacities of modern furnaces are becoming greater and more interest is being taken in operating costs. Metallurgists are realizing more and more that such calculations lead to a detailed understanding of the operation and the functioning of metallurgical processes. This subject involves the application of the principles of combustion, thermo-physics, and thermo-chemistry to the testing and operation of metallurgical furnaces; it will be discussed in this paper with particular reference to the reverberatory furnace. A discussion of furnace control from the combustion standpoint is included because it is a closely allied problem and is almost wholly dependent upon the same principles.

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In recent years much excellent work has been published dealing with the application of physico-chemical principles to the operation and testing of various processes in industrial chemistry and metallurgy. (See bibliography) Most of these works are general treatments of the subject and contain a wealth of
material essential to the engineer proposing to use the methods discussed in this paper in studying the economics of any process. An application of these principles and fundamental data to the process under investigation becomes a separate problem which, to a large extent, is dependent upon a thorough understanding of the operation of the process in question from the standpoint both of the chemistry and of the mechanics involved.

The development of thermal and material balances is based on the laws of conservation of matter and energy, which for the most part at hand, are valid. If adequate observations and measurements are taken during the operation of the process, and if accurate information is obtained concerning the materials introduced into the process and produced during it, a balance of input to output can be obtained for each element involved, and, likewise the heat input can be balanced against the heat output.

Professor Richards, a pioneer in this field, expressed very adequately in few words the importance of the work. He said: "The making of calculations respecting the quantitative workings of any metallurgical process, furnace, or piece of apparatus used in metallurgical operations is of the greatest importance for estimating the real efficiency of the process, for determining avenues of waste and possible lines of improvement, and for obtaining the best possible comprehension of the real principles of operation involved. From a correlation of all data, not only the specific efficiency of the furnace or process is determined but also the efficiency of any part further, the manner in which the energy required in the process distributes itself is revealed and often indicates ways in which the operation could be improved and the efficiency increased. In addition, the methods used serve as excellent means of comparing the operation and relative efficiencies of two or more similar furnaces or units operating under similar conditions. In the latter case, either simultaneous observations on all of the units in question must be made or observations must be taken under similar conditions, the comparability depending upon the judgment of the metallurgist and his comprehension of operating principles. The value and importance of these balances are thus dependent upon the collecting of the necessary quantity of data by direct measurement, by observation, and by chemical analysis, and also upon the careful handling and interpreting of these data in making the computations. Much relatively delicate apparatus is used which requires careful manipulation by observers trained not only in the operation but in the interpretation of the results. Because of the difficulties encountered in securing these data and in making the necessary calculations, investigations of this kind are infrequent. The difficulty with which the required data is obtained is increased by the large number of variables entering into the operation not only of various types of metallurgical furnaces but also even of furnaces of the same type operating under practically identical conditions. Changes in the character and the composition of the charge and the fuel, conditions under which the fuel is burned, the extent to which the system is sealed to prevent infiltration of air, and many other variables, all contribute to the difficulties encountered in a study of this kind and obviously make it impossible for a definite routine to be outlined for the testing of metallurgical furnaces.

This paper describes in a very general way some of the observations and calculations involved; it does not attempt a complete treatment of the subject, since that would be too formidable a task for a discussion of this kind. Furthermore, each furnace or process presents an independent problem which must be understood thoroughly before an attempt is made to carry out a thermal investigation. As a specific example of the procedure and as an illustration of the foregoing discussion, the average data and calculations for a 120,000-pound grate-fired copper smelting and refining reverberatory furnace and its auxiliary boiler are given.

**OBSERVATIONS AND CALCULATIONS**

The most important factor governing the accuracy of the furnace test is adequate and dependable data; computations and calculations, no matter how involved, tell little or nothing concerning the operating process if they are based on erroneous observations. Further, the measurements should not be restricted to those absolutely essential to the computations, since any added information which can be secured with but little effort serves as a desirable check on previous work. The observations should be made at definite time intervals which should be sufficiently close together to insure results representative of average conditions. The interval is dependent upon the process and the rapidity with which variations in analyses, temperature, etc. occur. The data necessary for a study of the operating process may be divided into two general classes: first, that pertaining to the material balance; and second, that pertaining to the thermal balance.

**Material Balance**

All materials entering or leaving the cycle of operation should be carefully metered or weighed, and a complete chemical analysis should be obtained on a truly representative sample of the material in question. The methods of metering and weighing the ores, fluxes, slag, metal, fuel (solid, liquid, or gaseous), the cooling water used, etc. are all standard and need not be discussed here. A discussion of the necessary analytical data will also be omitted. As no two plants have exactly the same layout, it is unwise to attempt to specify any definite location for the instruments required for making the above tests. These must be arranged to suit the particular conditions of the plant.

In obtaining much of the other necessary data, however certain definite requirements must be fulfilled if accurate results are to be obtained. The analyzing of the exit gases and the measurement of the gas volume present problems the solution of which demands extreme accuracy. Their importance is not quite so pronounced...
in the balance of materials as it is in the thermal balance, inasmuch as the heat in the gases is the major item in the latter case. Notwithstanding their greater value in the thermal balance, it is well to discuss at this point the apparatus and methods used for the determination of gas analyses and volumes, since these measurements are an integral part of the material balance.

Gas Analysis—The gas analysis proper may be accomplished with the aid of the Orsat apparatus. It must be remembered, however, that analyses obtained by this method represent the composition of the dry gases and that the moisture in the gases must be obtained through the material balance. Standard methods are used for determining the moisture in the materials charged, while the sling psychrometer can be used for obtaining the moisture content of the air. This phase of the problem should present but little difficulty. The obtaining of a truly representative sample of gas, however, is a question well worth considering. Some of the factors which contribute to erratic gas analyses are infiltration of air, the use of a sampling device constructed of a material which reacts with the gas at the sampling temperature, and the improper location of the sampling tube. Other factors affecting the accuracy are solubility of the gases in the liquids used, the effects of temperature and pressure changes during analysis, etc. With the proper handling of apparatus these latter factors are negligible if the sampling is done intermittently and the gas samples are analyzed as soon as they are collected. The infiltration of air merely results in dilution, the effect of which on the analysis of the gas is obvious. In loose flues, where infiltration is taking place, it is practically impossible to secure a representative sample, inasmuch as the analysis varies widely across any section of the flue. Although a loosely constructed and leaking flue does not materially affect the specific furnace efficiency, it greatly reduces the gross net efficiency of a system in which the heat in the gases is utilized. The wide variation in gas analysis obtained in a given cross-section of flue and often attributed to the stratification of the gases is, in the opinion of the authors, largely due to infiltration. If stratification were the sole cause, there would be CO₂ rich stratum, the CO₂ content of which would be greatly in excess of the theoretically possible value. This condition has never been reported. Experiments relative to the subject of flue gas stratification are now in progress, and the results will be published in a forthcoming bulletin of the Michigan College of Mining and Technology.

From the standpoint of furnace control it is imperative that the fine gas analysis actually represent the gases as they leave the furnace. If dilution has occurred by infiltration through the flue walls between the sampling point and the furnace, the analyses are of little value as a means of controlling combustion. However, if it is desired merely to obtain the heat in the gases, these analyses will suffice, provided of course that they represent the average composition, inasmuch as the total heat in the gases is not affected by infiltration at this point. Since, as was previously stated, it is practically impossible to obtain a truly representative gas sample from a flue which is subject to infiltration, and since the importance of the gas data is paramount in furnace testing, it is necessary that every precaution be taken to prevent infiltration. Operators interested in investigations of this kind are of necessity interested in furnace efficiency, since the furnace test would be futile if it were not being conducted for the purpose of studying the operation of the furnace and of determining possible means of improving conditions and increasing efficiency. Regardless of whether or not the heat in the gases is utilized in a waste heat boiler or other auxiliary device, the profound effect of leakage on the draft necessarily warrants the careful sealing of the flue.

The use of sampling tubes which react chemically with the gases at the temperature at which sampling is done comprises the second major item affecting the accuracy of the gas analysis. Iron, for instance, cannot be used at temperatures much higher than 400°C, since iron oxide is reduced by carbon at temperatures above 304°C and metallic iron is oxidized by carbon dioxide at temperatures in excess of 400°C. Various alloys are used which resist the corrosive action of the gases at temperatures somewhat in excess of those mentioned above; however, care must be exercised in selecting a tube which would prove safe under such conditions. For higher temperatures silica, porcelain, or clay tubes may be used, or the metal tube may be jacketed to permit water cooling.

The location of the tube in the cross-section of the furnace flue is relatively unimportant in straight flues practically free from infiltration. However, in flues in which a small amount of infiltration takes place it is obvious that the tube should not be placed too near the flue wall. Where a large amount of infiltration occurs, analysis of the gas sample is almost useless and may lead to erroneous conclusions regarding the operation of the process.

4 It is realized that infiltration into flues is sometimes advantageous; for instance, in flues in which the temperature of the gases is excessively high, it is often necessary in order to prevent undue loss of refractories.

Gas Volume Measurement—Perhaps the greatest difficulty encountered during the running of a furnace test is the measurement of gas volumes. The gas volume measurements must be made at some straight section of the flue in order to minimize the errors due to eddy currents which are bound to occur at any bend in the gas stream. The gas volume is not actually measured but rather is computed from the velocity of the gas and the area of cross-section, of the flue. The velocity of the gas is determined by means of the pitot tube, and the differential manometer which measures the difference between the velocity pressure and the static pressure of the steam. The vertical displacement...
of the liquid column in the manometer is, then, a measure of the pressure causing the gas flow, a pressure which is proportional to the velocity. The location of the pitot tube in the flue should be such that an average velocity of the gas is obtained. It is an experimental fact that the velocity of the gases flowing through a conductor is less near the walls than at the center because of the frictional resistance offered by the walls. The position in the flue at which the gases are traveling at average velocity can be computed from a series of readings taken at equal space intervals along both the horizontal and vertical axes of a cross section of the flue. Because of the many variables which may influence the reading of the differential manometer, the authors recommend the use of two complete instruments simultaneously recording the velocity in any particular section of the flue, one to serve as a check on the other. As soon as a difference is observed in the reading of the two instruments, a thorough inspection should be made of both, and the error accounted for and corrected.

With the above gas data and with sufficient quantitative and analytical data for all of the materials entering and leaving the system, a complete balance may be obtained for each element involved.

(5) An approximation of the gas volume may be calculated from the material balance with the ultimate analysis of the fuel and analysis of the gas being available.


Thermal Balance.

The primary object of the furnace test is, very often, to determine the various thermal efficiencies of the process under investigation. The reliability of the results and the justification of conclusions drawn depend to a great extent upon the accuracy of the temperature measurements made. It therefore becomes necessary to exercise the utmost care in making these measurements. The selection of pyrometric equipment and its use in obtaining temperature determinations will be discussed under the various items of the thermal balance for which temperatures are required.

As in the material balance, so also in the thermal balance, the items involved can be divided into two general classes: heat input and heat output.

HEAT INPUT.

The items which must be considered on the heat input side of the balance are:

1. Heat in the fuel.
2. Heat in reducing agents (if used).
3. Heat developed by exothermic reactions.
4. Sensible heat in the materials charged.

Heat in the Fuel—Except in converter and roasting processes in which the exothermic reactions supply a sufficient quantity of heat for the operations, the heat developed by the combustion of the fuel is the major item of heat input. In order to obtain this value the weight of fuel and its net calorific value must be known. The calorific power may be determined experimentally or may be calculated from the ultimate analysis of the fuel. The use of either method gives satisfactory results if proper precaution is taken in interpreting the data and in disposing of the question of the latent heat in moisture due to the combined water and the water resulting from the combustion of hydrogen in the fuel. Since the conditions under which the experimental calorific power is determined cannot be duplicated in the furnace, this value should not be used without applying corrections. The calorific power as determined by the calorimeter represents the higher heating capacity of the fuel. It may be determined either on the dry fuel (dried at 100-105°C) or on the fuel as fired. If it is determined on the dry fuel, the heat input due to the combustion can be found by computing the weight of dry fuel used from the weight of fuel as fired and its per cent moisture. Dry fuel contains hydrogen and also water of combination. Since this water, together with that formed by the combustion of the hydrogen cannot be eliminated by ordinary drying methods, and since its latent heat is not available in the furnace, the latent heat must be subtracted from the calorimeter value. Any free moisture in the fuel which can be eliminated by evaporation should be treated as water in the charge. The presence of this water in the fuel reduces the heating capacity per pound of fuel as fired, but the latent heat of this free water should be charged against the system. In high moisture fuels this item is of considerable importance. However, in a fuel such as an average bituminous coal ordinarily used in reverberatory furnaces, the correction is insignificant and usually lies within the limits of the accuracy of the calorimetric determination. For example, in an average bituminous coal containing 5 per cent hydrogen and 3 per cent moisture, 0.03 pounds of water exist in the uncombined state, while fifteen times as much, or 0.45 pounds, are produced by the combustion of the hydrogen. The correction necessary for the 0.03 pounds of water equals 0.03 x 586° = 17.5 Calb, a value which, under normal procedure, should not be deducted from the heating value of the fuel but should be considered with the water in the charge as an item of heat output. However, should this correction (deduction of latent heat of uncombined water) be applied with the consequent lowering of the calorific power of the fuel as fired, and the incident water not considered as an item of heat output, the resulting error would distort the furnace efficiencies by less than 0.3 per cent of their values. The following is given as an illustration of methods used in determining the heating capacity of the fuel. The fuel cited in the illustration is the actual fuel which was used in the furnace test discussed in this paper.
Heat in the Reducing Agents—In many metallurgical processes, coke, coal, gas, oil, and wood have important functions in addition to their use as fuel, one of the outstanding being their use as reducing agents. In this capacity, however, their calorific power is developed in addition to their use as fuel, one of the outstanding being their use as reducing agents. In this case the bath immediately surrounding the pole is cooled considerably and furnace efficiency is hampered because oxidation at the combustion surface of the pole is limited chiefly to the monoxide, and the full heating value of the carbon is not developed at this surface; the heat required to decompose the cuprous oxide and to evaporate the combined water in the pole is taken almost wholly from the surrounding copper bath; combustion proceeds at an extremely slow rate because of the limited oxygen supply from the bath and the atmosphere above it; and the partial distillation of the wood and the delayed combustion of the products of distillation cause heat loss at the combustion surface. Although the total calorific power of the wood is ultimately developed in the furnace, the rate at which it is made available is offset by radiation loss and the efficiency of the process is thus decreased. The method of determining the heat input due to the reducing agents is outlined in the section on heat in the fuel.

**Heat Due to Exothermic Reactions**—The chemical reactions involved naturally vary with the process, the conditions of operation, etc., and the heat due to these reactions can be determined from thermo-chemical and thermo-dynamical data, a thorough understanding of the chemistry of the process being necessary, however.

In the problem discussed in this paper the energy of the reactions was considered solely from the thermo-chemical standpoint, the available data being inadequate to permit the treatment of entire problem from the thermo-dynamical standpoint.

**Sensible Heat in the Charge**—Sensible heat in the charge is considered only when some of the materials charged have been pre-heated. If no pre-heating is done, all thermal calculations for sensible heat in materials can be referred to room temperature the heat in the charge disregarded. In this procedure a slight error is introduced which is due to the difference in specific heats of the materials input and output, but the error is very small.

**HEAT OUTPUT.**

The major items to be considered are:

1. Heat in the gases.
2. Heat in the metal.
3. Heat in the slag.
4. Heat required to vaporize the water.
5. Heat required for endothermic reactions.
6. Calorific power of ash.
7. Calorific power of waste gases.
8. Heat in the cooling water.
9. Heat lost by radiation, conduction, convection, etc.

Heat in the Gases—In order to determine the sensible heat in the gases it is necessary to know their composition, temperature, and weight or volume at standard conditions. With this data and the mean specific heats of the independent constituents between room temperature and the temperature of the gases, the sensible heat may be determined from the following general equation:

\[
Q = (W_{t1} S_{m1} + W_{t2} S_{m2} + W_{t3} S_{m3} + \ldots) (T - Tr)
\]

Where subscripts 1, 2, 3, etc. refer to the particular independent constituents 1, 2, 3, etc. (i.e. \(Co_2, N_2, O_2\), etc.)

\(T\) = Temperature of the gases.

\(Tr\) = Room temperature.

\(Q\) = Total sensible heat in the gases between temperatures \(T\) and \(Tr\).

\(W_t\) = The weight of the constituent.
The analyses, volume, and temperature are thus obviously equally important in obtaining the true value for Q. The analyses and the determination of the volume of the waste gases have been discussed under “Material Balance.” The actual methods used in making the temperature measurements and the factors influencing their accuracy will necessarily vary with the approximate temperature of the gases, and the range of temperature through which they may vary fluctuates during a complete cycle of operation. At temperatures above 800°C and in cases where the temperature of the gas does not undergo rapid changes, the optical pyrometer may be used by employing a refractory or alloy tube closed at one end and inserted in the stream of the gases. The optical pyrometer can then be sighted onto the back of the tube under black body conditions, and relatively accurate results obtained, some errors, of course, being introduced because of the lag between the temperature of the tube and that of the gases as the temperature of the gases changes. The method, however, offers the additional advantage that a bare thermocouple may also be placed into the same tube for making actual measurements or for purposes of checking. In measuring much lower gas temperatures such as at points beyond waste heat boilers, etc., the base metal couple may be used in conjunction with either indicating or recording thermometers. The use of recording instruments at any point is, of course, advantageous because a continuous record is made available. The gas temperature measurements should be taken at the same point as the analyses and volume measurements, and too much stress cannot be laid upon the necessity of the selection of the proper location.

(12) These items may vary somewhat with the nature of the process and the type of furnace used.

Heat in the Metal—In the case of pure metals heat in the metal is easily obtained if the weight of the metal, its specific heat in both the liquid and the solid states, its latent heat of solidification, and the temperature be known. The temperature can usually be obtained by the use of the optical pyrometer with the application of the proper radiation constants or by the use of a bare base metal thermocouple which can be inserted directly into the bath. If the composition of the bath is not known or is not constant, the total heat, per pound of metal at the temperature of the bath can be determined by means of the calorimeter. A relatively crude form of calorimeter in which the metal is poured directly into water and the rise in temperature of the water is measured, usually suffices.

Heat in the Slag—The heat in the slag may be obtained by means of the calorimeter as mentioned above, or an approximate value may be calculated from the analysis and the specific and latent heats of the individual constituents.

Heat Required to Vaporize the Water—The heat required to vaporize all of the free water entering the system comprises this item. The heat required to evaporate one pound of water will be taken as 80 + 539 — 38 = 581 Calb, where 80 Calb is the heat necessary to raise the temperature of one pound of liquid H₂O from room temperature, or 20°C, to 100°C, 539 Calb is the latent heat of vaporization of water from and at 100°C, and 38 Calb is the heat in water vapor between 200 and 100°C. (This amount is deducted because it is included in the sensible heat in the gases)

Heat Required for Endothermic Reactions—Like the exothermic reactions, endothermic reactions vary with the process, etc., and the determination of the heat required may be made with the aid of sufficient thermo-chemical and thermo-dynamical data.

Calorific Power of the Unburned Fuel—In grate-fired furnaces using lump coal, much unburned fuel falls through the grate into the ash pit. The ash is then a product of the furnace, and the heat potential in the unburned fuel becomes an item of heat output. The total heat lost may be computed from the weight of the ash and its calorific power as determined in the calorimeter. In furnaces firing powdered coal, the ash, except that which enters the flue dust, becomes a part of the slag and here, as in the case of gas and oil, the unburned fuel can be accounted for in the exit gases.

Calorific Power of the Waste Gases—When a pound of carbon burns to CO₂, 8,080 Calb are liberated. If that same pound of carbon in the presence of an insufficient amount of air burns to CO, only 2,430 Calb are liberated; in other words, the heating potential of one pound of carbon existing in the form of carbon monoxide is 5,650 Calb. Expressed in more useful terms, calorific power of one pound of carbon monoxide is 2,420 Calb. This value multiplied by the weight of CO in the waste gases will then equal the total calorific power. It is thus apparent that any undue amount of CO resulting from incomplete combustion has decidedly deleterious effect on the efficiency.

Heat in the Cooling Water—The initial and final temperatures of the cooling water are easily obtained by means of indicating or recording thermometers, and the heat carried away through this channel may be readily computed from the weight of the water and the temperature difference, the specific heat of water (liquid) being assumed to be equal to one.

Heat Lost Due to Radiation, Conduction, Convection, Etc.—Because of the many variables involved which are likely to result in error in the determination of actual radiation, convection, and conduction losses, it is deemed advisable to follow the usual practice and to determine the gross value by difference.

Efficiencies.

In general the term efficiency may be defined as the ratio of input to output. Its particular meaning and its degree are entirely dependent on the choice of the values that...
are to be considered in the input and output. From the data available in a furnace test of the kind discussed in this paper, a number of efficiencies may be obtained; those most valuable to the operator in judging the excellence of operation are:

Specific, the ratio of the heat in the metal to the total heat input.

Actual, the ratio of the heat utilized in the furnace to the heat available for use above the highest temperature required in the furnace.

Specific Actual, the ratio of the heat in the metal to the heat available for use above the highest temperature required in the furnace.

Gross Net Furnace, the ratio of the heat utilized in the furnace to the total heat input.

Gross Net System, the ratio of the heat utilized in the furnace and auxiliaries to the heat input.

Others, such as the efficiencies of auxiliaries (waste heat boilers, regenerators, recuperators, etc.) may be obtained if sufficient thermal and steam data are available.

The methods of arriving at the above efficiencies are, with one exception, explained in detail under the specific example of the furnace test given. The value of the gross net system efficiency may be secured when the efficiency of the auxiliaries is known. In the example no attempt was made to determine the efficiency of the waste heat boiler. Although the heat absorbed in the boiler was measured, the water evaporated and the quality of the steam and other necessary data were not collected.

Furnace Control.

Before entering the discussion of the actual furnace test, mention should be made of the possibility of utilizing some of the observations discussed in connection with heat and material balances as a means of furnace control.

It is well known that fuel is the most important raw material consumed by metallurgical furnaces in the production of most metals. Variation in the nature and extent of the combustion of fuel in furnaces is the most important factor contributing to variations in furnace efficiencies. For this reason, observations and calculations dealing with combustion, flame temperatures, gas measurements, load, and related subjects are among the most important in the study of the possibilities of furnace control. The control of combustion thus offers a possible means of controlling the efficiency. It is, of course, understood that many metallurgical processes operate on cycles during which the nature of the combustion necessarily varies. In these processes it is obvious that an absolute automatic control cannot be applied; however, any means of automatically or even manually obtaining data concerning combustion will serve both as an indicator at all times and as a means of guiding possible or required changes in the nature of the combustion at any point in the cycle.

The gas analysis is an indicator of the efficiency of combustion. Any variation in the amount of air supplied from that required for perfect combustion results in a decrease of the available heat. The presence of CO in the gas indicates incomplete combustion and a consequent loss due to the heat potential of the CO. Oxygen in the gas indicates an over-supply of air and a loss due to the sensible heat in the excess of air as it leaves the furnace at the temperature of the gases; further, the temperature of the gas rises somewhat with slight increases in the excess air.
where a reducing atmosphere is not required and where heat efficiency is the prime consideration, attempts should be made to maintain the air supply slightly in excess of the theoretical amount. In many furnaces this can be attained to some extent by observing the character of the gases as they enter flue, since a smoky gas indicates a deficiency of air while a clean gas is indicative of an excess. However, the more precise method of arriving at the required air supply would be to obtain actual analyses with the Orsat. As already shown, CO₂ analysis also indicates these conditions and offers the advantage of simplicity in obtaining a continuous record. The use of many of the automatic CO₂ indicators or recorders has been widely adopted in boiler control and to a rapidly increasing extent in the control of combustion in metallurgical furnaces. However, these instruments, if used, should be frequently checked against the Orsat, since they give no indication of the CO content and since the tendency for the presence of CO in the gas increases as the amount of excess air decreases to the point where the air supplied just approaches the amount necessary for complete combustion. The losses resulting from carbon monoxide in the gases are illustrated in Figure 1.

**FURNACE TEST.**

The furnace upon which this test was made is a 120,000 pound grate fired reverberatory furnace which, for the particular cycle under study, was smelting and refining a charge of native copper mineral.13 The layout of the furnace system and relative location of some of the instruments are shown in Figure 4. The furnace is 14’ x 30’ and is equipped with a 125-hp. waste heat boiler. In most furnaces the firing of coal on grates has been replaced by the use of oil, gas, or powdered coal; however, such a change does not modify the underlying principles of furnace testing, and the grate-fired furnace was used in this example because its operation is short and comparatively simple, and, though not illustrating optimum operating conditions or unusual efficiencies, shows the principles and methods involved as adequately as does a larger oil, gas, or powdered-coal fired furnace.

(13) A local term for the milling concentrates.

---

**Required Data.**

**Heats of Formation**

- Ca₃O₄: 50,810 Cal per pound mol
- CaCO₃ (from CaO and CO₂): 25,000 Cal per pound mol
- MgCO₃ (from MgO and CO₂): 27,800 Cal per pound mol
- Of slag made: 240 Cal per pound of basic oxide
- Heat content of solid copper at absolute temperature T:
  \[ C_v = 4.91 + 0.00122 T - 0.00000054 T^3 \]
- From this
  \[ S_{m} \left( t_{1} + t_{2} \right) = 0.00017 + 0.00022 t_{2} \]
  \[ 0.000000028 \left( t_{1} + t_{2} \right)^{2} \]
- Cal per pound of metal per degree between temperatures \( t_{1} \) and \( t_{2} \) degrees centigrade.
- Latent heat of copper 10.49 Cal per pound
- Sm of copper liquid (1083° — 1300°) 0.1218 Cal per pound degree.
- Mean specific heats of gases in Calb per pound of gas per degree centigrade:

---
Summary of Observations for Furnace Test.

Weight of mineral charged .......................... 157,995 pounds
Weight of limestone charged ................. 16,700 pounds
Weight of coal charged as adding agent ....... 6,760 pounds
Weight of wood used as adding agent .......... 6,750 pounds
Volume of cooling water used ............... 19,900 gallons
Volume of metal in finished shapes ......... 95,400 pounds
Weight of copper scrap ........................... 2,172 pounds
Weight of rich slag made ......................... 8,075 pounds
Weight of retreating slag made ............... 6,750 pounds
Weight of waste slag made ....................... 44,340 pounds
Weight of ash made ............................... 3,250 pounds
Calorific power of the fuel (calorimeter value) ............................ 7,817 Calb per lb.
Calorific power of the wood (calorimeter value) on dry wood ............................ 4,771 Calb per lb.
Calorific power of the ash (calorimeter value) ............................ 3,055 Calb per lb.
Barometer (average for run) ....................... 755 mm.
Room temperature, average for run .......... 20°C.
Average psychrometer reading: wet bulb ......... 68°F.
Dry bulb ............................................. 68°F.
Temperature (average) at which metal was removed ............................ 1,154°C.
Temperature (average) at which slag was removed ............................ 1,200°C.
Average calorimetric determination of total heat in the slag ...................... 375 Calb per lbs.
Initial temperature of cooling water ........... 15°C.
Final temperature of cooling water .......... 30°C.
Temperature of the gases, average over run ...... 1,266°C.
Gas analysis, dry: .................................. CO₂ 15.6%
O₂ 2.7%
CO 1.2%
N₂ 80.5%

THERMAL BALANCE.

**HEAT INPUT.**

1. Heat in the Fuel.14
   Practical calorific power of the fuel as fired is 7,531 Calb
   Weight of fuel fired ......................... 35,793 pounds
   Heat input due to fuel ................. 259,790,333 Calb

2. Heat Due to Coal Used as Reducing Agent in the Charge.
   Weight of coal charged ........................ 6,760 pounds
   Heat developed ........................... 46,514,160 Calb

3. Heat Due to Combustion of Wood.
   Practical calorific power of the wood as fired ........ 2,646 Calb
   Weight of wood used ........................... 6,750 pounds
   Heat developed ........................... 17,806,500 Calb

4. Heat Due to Exothermic Reactions.
   a. Oxidation of copper to Cu₂O during oxidizing period.
   b. Weight of Cu₂O formed ........................... 8,620 Calb
   c. Weight of Cu₂O formed ........................... 8,620 Calb
   d. Heat of formation of slag
   e. Weight of the basic oxides entering slag ....... 3,357 pounds
   f. Heat of formation of slag 240 Calb per pound of basic oxide
   g. Heat developed ........................... 240 x 3,357 = 805,680 Calb

**HEAT OUTPUT.**

1. Heat in the Gases.
   Average temperature of the gases ................ 1,266°C
   Average gas analysis, dry: CO₂ 15.6%
   O₂ 2.7%
   CO 0.9%
   N₂ 80.5%
   Cross sectional area of the flue .......... 4.51 square feet
   Slope of the flue tube .......................... 4.51 inches
   Average retort head flue gas analysis ........ 11.6%
   O₂ 7.4%
   CO 0.9%
   N₂ 80.1%
   Slope of piston tube manometers ................ 1/20
   Cross sectional area of boiler flue ........ 4.51 square feet

NOTE—In the following problem, the calculations are carried beyond the degree of accuracy warranted by the data, merely for the purposes of checking.
SUMMARY OF FURNACE TEST.

HEAT INPUT.

Source        | Calor | Percent  
--------------|------|----------
Fuel          | 269,593,533 | 79.62   
Reducing Agents  
  a. Coal | 46,514,160 | 13.74   
  b. Wood | 17,800,590 | 5.27    
Exothermic Reactions  
  a. Formation of CaO | 3,829,610 | 1.13   
  b. Formation of slag | 805,680 | 0.24    
Total               | 338,603,393 | 100.00  

HEAT OUTPUT.

Source        | Calor | Percent  
--------------|------|----------
Heat in gases | 193,232,379 | 57.96   
Heat in the metal | 17,806,638 | 5.11    
Heat in slag | 22,300,250 | 6.50    
Heat required to vaporize water | 5,690,314 | 1.68    
Heat Required for Endothermic Reactions  
  a. Calcine lime | 2,371,236 | 0.70    
  b. Reduce CaO | 2,958,725 | 0.87    
  c. Caloric power of ash | 992,573 | 0.29    
Caloric power of gas | 13,952,550 | 4.12    
Heat in the cooling water | 5,690,615 | 1.50    
Heat lost by radiation conduction, convection, etc. (by difference) | 63,827,366 | 18.63    
Total               | 338,603,393 | 100.00  

**Additional Items Used in the Determination of Efficiencies.**

**Heat Absorbed in Boiler.**

Heat in boiler flue gases:  
Average temperature of gases .......... 399°C.  
Average gas analysis, dry .......... CO₂ 11.5%  
                              | CO 0.9%  
                              | O₂ 7.4%  
                              | N₂ 80.1%  
Average pitot tube reading .......... 1.53 inches.
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Basis—1 Pound of Coal as Fired.

Maximum temperature, \( t \), to which the product of combustion and excess air can be raised by the available caloric power may be determined as follows:

**HEAT AVAILABLE IN THE FURNACE.**

Only the heat existing in the products of combustion between the maximum temperature required in the furnace and the maximum theoretical temperature of the products is available for use in the furnace. The degree to which this heat is utilized is the most important measure of efficient operation. It is obvious that if the maximum temperature attainable in the furnace is below that required for the process, the actual efficiency is zero. In this connection, it is important to note that many reactions, although exothermic from and at room temperature, actually require heat if the products of the reaction are removed at higher temperatures. The heat developed by exothermic reactions is considered on the heat input side of the balance, and if they become endothermic in the process, the heat available in the furnace is decreased and the actual efficiency thus lowered.

**Heat Available Above 1200°C and 2009°C from the Combustion of the Coal—**

<table>
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<th></th>
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</thead>
<tbody>
<tr>
<td>Excess Air</td>
<td>12.5</td>
<td>10.5</td>
<td>8.5</td>
<td>6.5</td>
<td>4.5</td>
<td>2.5</td>
</tr>
<tr>
<td>( N_2 )</td>
<td>0.034</td>
<td>0.034</td>
<td>0.034</td>
<td>0.034</td>
<td>0.034</td>
<td>0.034</td>
</tr>
<tr>
<td>( O_2 )</td>
<td>0.014</td>
<td>0.014</td>
<td>0.014</td>
<td>0.014</td>
<td>0.014</td>
<td>0.014</td>
</tr>
<tr>
<td>( H_2 )</td>
<td>0.004</td>
<td>0.004</td>
<td>0.004</td>
<td>0.004</td>
<td>0.004</td>
<td>0.004</td>
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<tr>
<td>( CO )</td>
<td>0.003</td>
<td>0.003</td>
<td>0.003</td>
<td>0.003</td>
<td>0.003</td>
<td>0.003</td>
</tr>
<tr>
<td>( CH_4 )</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
</tr>
<tr>
<td>Total Excess Air</td>
<td>0.101</td>
<td>0.101</td>
<td>0.101</td>
<td>0.101</td>
<td>0.101</td>
<td>0.101</td>
</tr>
</tbody>
</table>

**Heat Available from Exothermic Reactions—**

- Available heat due to oxidation of copper, \( 4 Cu + O_2 \rightarrow 2 Cu_2O \).

<table>
<thead>
<tr>
<th>Heat of Formation of ( Cu_2O ) from copper and oxygen from and at 25°C, 3,829,610 Calb.</th>
<th>Heat of Formation of ( Cu_2O ) from copper and oxygen from and at 25°C, 3,829,610 Calb.</th>
<th>Heat of Formation of ( Cu_2O ) from copper and oxygen from and at 25°C, 3,829,610 Calb.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Weight of coal used, 6,750 pounds.</td>
<td>Weight of coal used, 6,750 pounds.</td>
<td>Weight of coal used, 6,750 pounds.</td>
</tr>
<tr>
<td>Total heat available from coal above 1200°C, 6,750 x 799 = 5,393,250 Calb.</td>
<td>Total heat available from coal above 1200°C, 6,750 x 799 = 5,393,250 Calb.</td>
<td>Total heat available from coal above 1200°C, 6,750 x 799 = 5,393,250 Calb.</td>
</tr>
</tbody>
</table>
**ACKNOWLEDGEMENTS.**

The writers gratefully acknowledge the cooperation received from the technical staffs of the Lake District Smelting Works in obtaining data and in conducting furnace tests during the past several years. The assistance given by the metallurgical staff of the Michigan College of Mining and Technology in obtaining the data for the test reported in this paper is also acknowledged with appreciation.

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**SAFETY AT PICKANDS, MATHER AND COMPANY IRON MINES**

BY A. H. TRESTRAIL*

(This Paper First Appeared in The Mining Congress Journal)

Where there is vision followed by consistent effort, there is achievement. Twenty-five years ago mining was acknowledged one of the most hazardous industries. The natural forces that man encountered in uncovering buried treasure were believed to be uncontrollable and not amenable to human effort. Some held that safety might be applied to shop or factory but when mining with safety was contemplated, the majority concerned were skeptical if not ready to ridicule. As every new movement has its pioneers, there were some mine operators who were convinced that safety could be applied to mining and were ready to take the initiative. Pickands, Mather and Company stood among these pioneers of safety who were possessed with the vision and the desire to carry out their convictions. Now, after a quarter century of consistently keeping the objective before them, they have every reason to believe their vision has been clear and their efforts toward safe operation have brought desirable results. It is true that experiences have been encountered which have been discouraging, yet results attained are such that we know safety pays in tangible assets while the misery which has unquestionably been avoided over this period of time will never be revealed by the balancing of any ledgers.

*Range Safety Instructor, Pickands, Mather & Co.

The conviction that safety is right first found expression in a sincere effort to guard all mechanical and conceivable hazardous physical conditions. This, while fundamental and elementary in any safety program, must be subscribed to by all who expect to have the cooperation of the entire organization. In order to bring mechanical equipment into conformity with the best conceptions of safety, much had to be changed and some equipment discarded. Mechanical guards were not available at first and “homemade” guards were used. Today, however, practically all machinery comes from the manufacturers carrying the proper protection. Such hazards as were apparent were promptly cared for while those less apparent were remedied after experience had proven a hazard existed. As an illustration, when the first tugger hoists appeared, the revolving drums and winding cables looked quite innocent. When an employe lost the tip of his thumb trying to control the cable while operating the machine, it was realized that the cable must be guided and the drum enclosed. Later, it also became apparent that broken cables can do considerable damage to the operator of the tugger hoist and a guard was devised to prevent a broken slusher cable from flying back into the operator’s face. These examples of developing protection at points of danger...
are typical of safety growth. While foresight is highly desirable in the prevention of injuries, no opportunity has been lost to profit promptly by such experiences.

A tour of inspection would reveal, from the time one left the change house until he returned, many devices designed to make employment safe and to keep safety uppermost in the workmen’s minds.

Hoisting equipment for handling men into and out of mines is provided with the best known automatic controls. Thus, if for any unforeseen reason, the hoistman should become incapacitated, the hoist would come to a stop when it approached its destination. Rules provide that the hoistman must return all signals as received before he can move the hoist. This prevents hoisting accidents due to misunderstanding of signals.

Upon opening the self-closing gate which guards the entrance to the shaft, we step onto the cage. Our conductor closes the cage door and we are protected overhead and on all sides. We descend, possibly two thousand feet and step from the cage onto a well-lighted level. While one carries a carbide lamp, he could in many of our mines dispense with it entirely due to electric lighting throughout the mine. All mines have levels and main travel ways electrically illuminated. Good light is a strong factor in the prevention of accidents.

As we proceed, we have little fear of a mis-step for all openings are guarded and ladderways are kept in good condition. Passageways are kept clear from obstructions as cleanliness is recognized as having a definite bearing on safety. “Clothes may not make the man” nor good housekeeping the mine but good housekeeping does prevent stumbling, slipping and other injuries. It tones the employees mental attitude which is the major factor in any worthwhile effort toward the reduction of injuries.

One would complete such a tour convinced that no efforts were being spared to guard every point where danger might exist. A similar survey of the open pits operated by the company will also reveal that no effort is spared to make their physical conditions conform to the best safe practices.

There is considerable difference from a standpoint of comfort in the first hard hat or hard toe boot and those that are in the first hard hat or hard toe boot and those that are now used. Their effectiveness in the reduction of injuries was soon realized. They both met with opposition when first introduced but it did not take long to turn opposition into approval and today the hard hat is worn by all underground employes and hard toe boots are generally worn. Goggles are furnished for all work where danger exists from flying objects. Underground the screen type of goggle is most commonly used. Safety belts for employes have justified their use in more than one instance.

No safety program is complete after the guarding of hazards has been completed. The education of the employe and the cultivation of the proper attitude towards safely performing his work is a continuous process. To be sure the employe does not desire an accident but he has to be taught to take care of himself. Here the foreman and safety inspector find ample opportunity to make their contributions to the well-being of the individual employe. This is not a concise series of “don’ts” delivered in one short lecture but must continue daily by precept and example. The foreman is held responsible for the safety of his employes and therefore it is he who must observe carefully the habits of his men and correct any unsafe practices.
Bulletin boards are maintained at all properties for the display of safety bulletins. The National Safety Council's bulletins are used, augmented by home made bulletins.

The general safety program is directed from the Cleveland office. On each of the ranges where the company operates is a range safety inspector. His chief responsibility is to interpret the company's policy in safety and he reports directly to the local operating official. Recommendations of the safety department are presented to the general superintendent and if approved by him must be carried out by the operating department. In other words, the actual execution of safety measures are a part of the mining program. A monthly summary of the department's activities is forwarded to the Cleveland office, the general manager and his immediate assistants.

Much has been written about the necessity of management's active interest and forceful participation in prosecuting safety work. A tree's value is best determined by the fruit it bears; management's sincerity can best be recognized in the results obtained from the safety program. If Pickands, Mather and Company felt that life and limb were not being saved, that there was not reason for more contentment in families, that both the earnings of employees and the company were not enhanced by this safety work, they would not have the conviction nor the courage to prosecute it with such vigilance as has been characteristic of their efforts.

Recognizing safety as an integral part of efficient operation and its ability to serve the highest interest of employee and employer, they strive toward the goal which is the elimination of all accidents.

Interest has been maintained by many and varied forms of contests. Mine has been pitted against mine and mines have competed with their own records. Awards have been made to winners in district contests with each employee at the winning mine receiving a suitable prize. One of the most successful methods of increasing interest in safety was to reward employees for each month, the mine went without a lost time accident. Some very fine records were made. Several mines operating with large crews of men were able to complete 12 months of continuous operation without a lost time accident and others made records of several years without a lost-time accident.

First-aid and mine-rescue training was introduced very early as a part of the safety plan.

Classes in first-aid are held regularly at all properties. At many of these, 100 per cent of employees have received this training and at all properties enough trained first-aid men are maintained to care for any emergency injury that might arise. First-aid primarily is a strong factor in cutting down disability which may follow an injury. It also has an influence in making the employee safety minded. Instruction in the treatment of imaginary injuries and discussion of the conditions under which they occur impresses on workmen the results of carelessness.

First-aid contests between teams representing various mines have increased interest in first-aid work and developed some exceptionally competent first-aid men. The company has not only encouraged these contests on their operating ranges but have, on one occasion, sponsored an interrange company contest with teams representing the Mesaba, Gogebic and Menominee Ranges competing. The winners of this contest went to the national meet where they were awarded third place.

Trained crews in mine rescue are maintained at each mine and several complete rescue stations are maintained on each of the ranges where the company operates.
The Bureau of Mines has been decidedly helpful both in training first-aid and mine rescue men and helping to arrange contests. They have also contributed valuable aid in the solving of other safety problems.

BRIEF HISTORY OF SOME OF THE ABANDONED MINES OF THE MARQUETTE IRON RANGE

BY HARRY T. HULST, ISHPENING, MICH.*

The mines of Marquette County may be grouped in several districts or locations, some of which are contiguous while others are separated by several miles. For convenience I shall designate these as the Michigamme, Republic, Champion, Humboldt, Winthrop, Ishpeming, Negaunee, Cascade and Swanzey districts. This is merely an arbitrary geographical division and has no connection with any geological or economic condition. Each has passed through a cycle of activity (or is still on the way) and today no one can say but that each and every one of these localities may see activity again.

Rather than carry you through a tiresome set of annual production figures, which may be obtained from the reports of the State Mineral Statisticians, I have selected one or two of the earliest operations in each of these districts for brief mention of performance.

Conditions in the early days, and I mean the sixties and seventies, can in no way be compared with those of the present. The local press of earlier times made news of the discharge of a few kegs of powder and the shipment of 2,700 tons of ore from one mine in a week. However, it is not the purpose of this paper to draw comparisons.

Manual and animal labor were the chief sources of power in those remote days and the shallow depths of operations required little mechanical assistance for either hoisting or pumping. In fact, some of the earlier operators discontinued work at the water level rather than go to the expense of installing pumping equipment.

Nearly everyone who has made even a superficial study of the early history of mining in the Marquette Range knows that the first actual operations were started by the Jackson Iron Company. While Mr. Burt really recognized the first indications of the existence of iron ore by the unusual behavior of his compass needle, Mr. Everett's party were the real discoverers of "Jackson Mountain" as it was then called, being led to the actual outcroppings by an Indian.

Three years later, on June 6th, 1848, the Jackson Mining Company was incorporated and in 1849 the name was changed to the Jackson Iron Company. Statistical records, which do not always agree, credit the Jackson Iron Company with shipments aggregating 25,000 tons of ore prior to 1856. Eleven years later the season's shipments totaled 127,491 tons which was the first time the hundred thousand mark had been reached by any single mine on the Marquette Range. Annual shipments in excess of one hundred thousand tons were maintained until 1873.

In the Ishpeming district the Cleveland Iron Mining Company was the pioneer, starting the operation of open pits east of the Village of Ishpeming and known as the "Incline" and "Schoolhouse" mines. The first shipment was in 1852, amounting to approximately 1,000 tons. By 1868, the annual shipments had passed the hundred thousand mark.

For the purpose of recording a hitherto unpublished story of the early history of the Lake Superior district, the following facts given me by C. H. Moss, of Ishpeming, a relative of the man in question, may not be amiss.

Charles H. Hall, a machinist by trade, foreman of a shop in Hartford, Conn., at the age of 40 contracted a severe case of measles. Fearful of tuberculosis following, he came to Marquette in 1867 to visit a friend, Edward C. Hungerford. A civil engineer, Mr. Hungerford had been here ten years and, in partnership with E. B. Ward, had erected the Pioneer Furnace in Negaunee and the Deer Lake Furnace. Mr. Hall's restoration to health was so rapid that he decided to dispose of a residence property in Chester, Conn., and invest the proceeds in the Deer Lake Company, purchasing Mr. Hungerford's interest. In the Spring of 1868 his wife and son, a boy of 14, came to Marquette where they resided until a home was built for them at Deer Lake.

Beaver had built a dam on the rocks near the site of the furnace and this furnished sufficient power for an air compressor (they are called "blowers" in modern smelting) also the operation of a small sawmill. With the beginning of his management of the furnace, Mr. Hall had the dam strengthened and increased in height to give greater storage. In the panic of 1873 the Deer Lake Company was the only operation in this part of the country which paid its labor in real cash. The reason for this was that the company's treasurer resided in Norwich, Conn., and was able to honor drafts to meet payrolls. The success of the company was noted by the owners of the Lake Superior Iron Company's mine, Cleveland and Boston interests, and they asked Mr. Hall to undertake its management. He demurred on the plea of not being a miner, only a Connecticut Yankee mechanic, but they were convinced that anyone who...
could operate any business as successfully as he had through such a difficult period could also run an iron mine with little previous knowledge in that line.

In 1875 he assumed management, relinquishing his connections at Deer Lake which were taken up by his own son, Edward R Hah, and William H. Rood. Thanks to his unusual mechanical ability, Mr. Hall was able to introduce innovations previously unheard of in the mining and shipping of iron ore. His first step in an economic direction was the installation of a central hoisting unit from which four separate shafts could be operated simultaneously. Later this was increased to six. The Lake Superior Iron Company was a million and three-quarters in debt when Mr. Hall took over the management. That the owners were wise in their selection of managers is evidenced by the fact that the debt was wiped out very soon and handsome profits paid to stockholders.

To Mr. Hall must also be credited the first use of a steam shovel for loading iron ore into railroad cars. Up to this time power shovels had been used in gravel and sand banks only, and never after the ground became frozen. Fear was expressed that mechanical shovels could not be built strong enough to handle the mine stockpiles with long slabs of the hard varieties reaching back into the piles and the soft hematites freezing. The steam shovel “Iron King” was ultimately built and when any part failed or broke under the heavy duty, it was replaced with another of sufficient strength not to break in the same place again.

The Winthrop, or as it has been later known, Section 21 Mine location, some three miles south of Ishpeming, was a thriving community in the seventies and eighties. The first mine opened here was the New England in 1866. This was followed by the Winthrop in 1870, the Albion in 1872 and the Goodrich in 1873, although these last two are not within the Winthrop district, being over a mile west.

As you drive west on US Highway 41 past the railway station at Humboldt, how many of you know that a sizeable town was situated on the very land traversed by the concrete? Just before reaching the little hill and rock cut where a road sign warns of the approach of a trunk line junction, on the right hand side, two dwellings are visible. These are all that remain of several blocks of lots laid out with a regularity that would do justice to any modern city. A few scars from partially filled cellar holes may still be seen, and if one cares to search carefully enough the locations of the streets themselves may be found. Back toward the station and on the hill south of it was another collection of dwellings which housed the miners’ families.

The mines in this district were the Edwards, Humboldt, Argyle, Sampson, Baron, Washington, Hunger ford and Foxdale. The Washington was the first to start operations which was in 1865. Several changes of ownership and management resulted in considerable confusion of names, that of Washington, Humboldt and Baron being applied to the same operation by different operators.

To anyone at all interested in “modern” archeology I would recommend a trip to Humboldt and a visit to a few of the openings which may be found by climbing over the old rock dumps south of the highway across the level ground just before reaching the railway station. Here you will see several of the old stopes dipping to the north, excavated to the water level, with the hanging walls intact and as solid as the day the mine was opened.

In passing it may be of interest to record the fact that in 1896 Thomas A. Edison spent some time and money at Humboldt in the erection and operation of an experimental mill for the magnetic separation of local ores. A fire destroyed the plant before any definite results were obtained and it was never rebuilt.

The Michigamme Company was organized in 1870 and two years later mining was started. A shipment of 141 tons is credited to the season-of 1872. The mine operated until 1895 and a total of 875,292 tons of ore were shipped.

The present highway passes directly over one of the old shafts which was filled with rock taken from the cut just before entering the village. The Michigamme district also includes the Spurr and Steward mines which have long been abandoned.

The Webster and Wetmore mines were operated as separate properties in 1882 and today are combined under the name Imperial, at one time operated by the Cleveland-Cliffs Iron Company and later by the Ford Motor Company. Two miles west of the Imperial are the Beaufort and Titan mines which were wrought in the eighties.

The Taylor mine, seven miles from L’Anse, can hardly be considered as belonging to the Marquette Range, yet it could not be assigned to any other. Opened in 1880, shipments reached a total of 33,970 tons by 1882 when the mine was abandoned.

Almost exactly six miles east of the Village of Michigamme is the Champion mine. Just why or for what purpose no one seems to be able to explain, but the Champion mine is located in the Village of Beacon while the East Champion and Keystone mines are in the Village of Champion. None of the older inhabitants seem to know where the dividing line comes although it is a generally accepted fact that the school building forms part of it.

Charles E. Wright, state mineral statistician, in the 1879 report states that the Champion Iron Company was organized on August 26th, 1869, but later in the report the mine is credited with a shipment of 6,255 tons of ore the year previous. This was apparently the beginning of mining in this vicinity.

Several outstanding features appear at the Champion mine and one of them is its great depth. Not that two
thousand feet is considered notable today, but up to the past few years the Champion and Republic claimed the distinction of being the deepest iron mines in the entire Lake Superior district.

Another most unusual feature worthy of note is the "shape" of the Champion's deepest shaft, if one may be allowed to use that word in connection with a hole in the ground. Starting from the surface at a rather steep angle to the north, No. 7 shaft changes to vertical at the nth level. As mining was continued below this depth it was found that the ore body gained to the west so that each succeeding level required longer rock drifting to connect shaft and ore body. To remedy this condition which made the development of each level more expensive, the shaft was given a slight twist counter clockwise with each set of shaft timber. This was continued until the 23rd level was reached when the long dimension of the opening had been turned gradually through almost a right angle. At this depth another change was made in the vertical plane and the shaft was started on a dip of 60 degrees to the west to follow the gain the ore was making. Should anyone care to find a sample of magnetic ore, a small pile at Champion No. 5 shaft will supply it.

The Republic mine was perhaps as well known in distant localities as any iron mine in the entire Northwest. When I was a pretty small boy our family spent part of a summer on Cape Ann, Massachusetts. In the little hotel in the Village of Magnolia I remember one of the hotel employees was displaying a highly crystalline bluish rock which he was telling all who asked that it was iron ore from the Republic mine in Michigan. The Republic Iron Company was organized on October 20, 1870, and shipments commenced two years later.

From an historical point of view there is little further to be said about the Republic except to note a record of continuous annual shipments equalled by no other mine on the range up to the time of its abandonment.

The Cascade Range, as it is known by some, is the Palmer area where several pits are operating today. Opened in 1871, the Cascade mine was later called the Pittsburgh and Lake Superior and finally the Volunteer. The latter has no connection with the present mine of the same name. The Star West is the only other mine in the Cascade district which was opened prior to my dead-line date of 1890. Starting shipments in 1878, a total of nearly 200,000 tons are credited to the mine.

No doubt some of you will wonder at my failure to mention the Gwinn district and the mines situated there, but let me remind you that Gwinn did not exist in 1890, nor in 1900 for that matter, so it is beyond the scope of this paper. However, there were mines in the Princeton area which is a part of the Gwinn district. In 1872 the Smith mine was opened. Later it became known as the Cheshire and after this it was the Swanzey when it was obtained by the Cleveland-Cliffs Iron Company who again changed the name to Princeton. The Brotherton, later called the Stegmiller, was the only other mine in the district at the time.

The list of abandoned mines seems almost limitless when one starts to pore over the old records and I know I have omitted names very well known to most of you. There is the Lake Angeline, the Queen, or Regent Group; the Dexter, the Cambria, the American-Boston, the Hartford, the Dalliba and Phoenix and many others you are probably familiar with, but this is not intended to be a complete list and I believe I have selected enough to leave a few suggestions and reminders of the earliest recorded facts and features of the iron industry of the Marquette Range.

EARLY METHODS OF TRANSPORTING IRON ORE IN THE LAKE SUPERIOR REGION

BY JOHN HEARDING, DULUTH, MINN.*

In 1844 iron ore was discovered near Negaunee, Mich., at what is now known as the Jackson mine. From that time to 1850 there is but little information as to how what ore was mined was taken to the Lake front but it is to be presumed that it was packed on men's backs on the very rough trails until such time as horses or oxen could be used for that purpose.

It was 16 miles or more to the pits where the ore was mined. The grades were very severe for a number of miles after leaving the Lake shore, so, at the beginning, the ore was at times hauled on travois, two curved sticks that could be pulled over small logs and stones, bolted at the bow and held apart in the middle by a cross log, often called the bunk, for nothing that had wheels could remain upright on the rocky trails.

In 1850, it appears that 25 double teams were employed during the winter hauling ore from the Cleveland mine to Marquette, Mich., for use in the furnace there. The roads were so bad that such hauling had to be done in winter on the snow.

In November, 1851, Heman B. Ely made an agreement with the Cleveland and Jackson Mining Companies to build a railway from Marquette to the mines, if, in return, they would give him the business of hauling the ore on the railway from both mines. However, Mr. Ely found it difficult to finance the railway and the Jackson Mining company decided to build a plank road from the mines to the lake at Marquette in 1852. This idea met with the approval of the general public as it used the materials that could be obtained in the country and gave employment to those that needed the work. Also, railway equipment was very difficult to get into the country and was also very expensive. The plank road was completed and ore was hauled over it for several years. Mr. Peter White drove my mother over this road to see the mines in 1861.

*Retired Assistant General Manager, Oliver Iron Mining Co.

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STRAP RAILROAD STARTED IN 1853

Approximately in the summer of 1853 work was commenced on the building of a strap railway from the mines to Marquette, but it was not completed in time to move any ore over it during the winter of 1854 and the ore was hauled as usual by sleighs and teams. At this time Mr. Heman B. Ely was constructing his steam railway, but, owing to a lack of supplies and funds to pay for the work, it was not completed until September, 1857.

For those unfamiliar with the construction of a strap railroad, the following description may be of interest:

After the grading was completed and settled, cross ties were laid on the roadbed but at greater intervals than in modern practice. These ties were called sleepers and on them were bolted or spiked longitudinal timbers parallel to the direction of the roadbed. Upon these timbers were bolted or spiked flat pieces or strips of iron (hence straps) with countersunk holes to receive the spikes or bolts with similar heads.

The strap railroad went into commission on November 1, 1855. Its motive power was mules and it took a team of them to haul one car containing about four tons of ore. The greatest difficulty was getting the cars backs to the mines for the grades were "frightful" on the return trip. Frequently the car would get out of control and then the mules suffered as they could not get out of its way, and were usually killed. A team of mules often cost as high as $1,000.00 or more and hay went to $50.00 per ton, making the operation very expensive. Overtures were made for a consolidation of the strap railroad with the steam railroad and in 1857 it was abandoned.

Congress passed laws to stimulate the building of railroads in 1856 and in 1857 the steam railway was completed through such aid. In 1857 Mr. H. B. Ely died and the work was completed by his brother, Mr. S. P. Ely, for whom the town of Ely, Minn., was named. Two engines were shipped to Marquette for use on the railroad in 1857 and were named the Sebastopol and C. Donckersley, as was customary at that time. They weighted 25 tons and it was estimated that they would haul, under favorable conditions, 1,200 tons of ore to the docks in a day. This "avalanche" of ore was supposed to take care of all requirements for ore for a long time to come. From this beginning, that seems so small to those that know of modern operations, there has developed the most perfect system of handling coarse fright in this country. It has been successful through the cooperation of all who have been interested in its operation.

T RAIL COMES INTO USE

The strap rail was soon discarded as it became dangerous when the traffic became greater in volume as the ends of the strap irons curved up from the constant passing of the locomotives and cars and caused many derailments. There are many living who can remember hearing of accidents caused by the strap rails curling up through the floors of the wooden passenger cars and killing or injuring the occupants.

In place of the strap rail, the T rail was used. More cross ties were introduced and the longitudinal timbers omitted. The sketch below shows the various shapes of rail adopted as early as 1848 in England, all of which were used with the “stringers” and in them the idea of the T rail predominates. At the present time rail weighing as high as 136 pounds per yard is being used in some cases.

Locomotives were also developed to haul larger loads of larger cars. In 1887 approximately, a type of locomotives called "Moguls" with six driving wheels and a four-wheel Poney truck came out and in the 70’s an ore car was developed with two trucks carrying 20 tons of ore. However the railroads serving the Marquette and Menominee ranges still continued to use the four-wheel, 10-ton car until as late as 1886.

Link and pin couplings were used up to the late 80’s when the automatic couplings were introduced and at about that time, air brakes were installed on the ore cars.

With the continued improvements, the engines have grown from the four-wheel, 25-ton engine shown in the illustration to the giant Mallet weighing with its tender 332 tons, and the cars have grown from the 10-ton, four-wheel “Jimmy” to the double truck, 75-ton steel ore car. With the improvements in grades and roadbed the paying load has increased from about 40 tons to 12,000 tons, per train.

Naturally, the docks for handling the ore at the lake ports had to grow with the increased railroad capacity.
GROWTH OF ORE DOCKS

The first docks at Marquette were only platforms built on piling and were about 12 feet above the water. On them the ore was first shoveled from the flat bottomed cars and later dumped onto them from the cars having trap doors in their bottoms.

From these platforms the ore was shoveled into the small boats or wheeled into the larger boats in wheel barrows. In this way it took from five to six days to load a 300-ton boat. Hand trimming was employed after the ore was practically loaded and this work was very severe at all times, particularly so when the men had to work under the hot decks in summer.

In 1855 the Jackson and Cleveland companies built docks on the lake front as stated above, but in 1857 the Lake Superior Iron Company decided to build a dock with pockets in it that was 25 ft. above the water with the pocket openings 12½ ft. above the water.

This was considered too high by some of the captains but arrangements were made to overcome their objections.

As such pockets provided storage for the ore when there were no boats in port the Cleveland Company in 1858, built a similar dock that was 30 ft. above the water line with the pocket opening 12½ ft. above the water line, giving them an extra five feet for ore storage. In 1864 the Railway Company built a dock at Marquette to care for boats of 1,000-ton capacity that was 35 ft. above the water line with the pocket openings 12½ ft. above the water line, as the boat capacity had increased from 300 tons to, in some instances, 1,000 tons as stated above.

Fire destroyed the Railway dock in 1869 and in the place of the old dock the company erected a new dock that was 38 ft. above the water line with pocket openings 25 ft. above the water line.

In 1883 the Railway Company erected another dock that was 47 ft. above the water line with pocket openings 27 ft. above the water line, and this dock seemed to take care of the needs of the boats for many years.

The recitation of the above data is made to show the increased accommodation made necessary in the docks by the increased ore traffic and the growth in size of the ore boats.

In 1864 and 1865 the Peninsula Railroad was built from Escanaba, Mich., to Negaunee, Mich., giving an outlet from the Marquette Range to Lake Michigan (now a branch of the Northwestern Railroad). In 1872 this railroad was connected to the C. & N. W. by building the line from Green Bay, Wis., to Escanaba, Mich.

In 1877 shipments began over a new branch line that the Northwestern Railway had built to the Menominee Range from Escanaba and the mixed equipment of 10-ton and 20-ton cars was used as late as 1886.

The docks at Escanaba were built 47 ft. above the water line with pocket openings 27 ft. above the water line.

The first docks at Two Harbors, Minn., and Ashland, Wis., were of the same dimensions, varying only in length as the harbor bottoms permitted.

With the changes of size and length in the boats, the clocks have been raised to meet the requirements of the service.

The original clock built for the D., M. & N. Railway in 1893 was 2,300 ft. long, 57.6 ft. high with the pocket openings 30 ft. above the water line.

This clock was followed by a clock 2,300 ft. long, 84.5 ft. high, with pocket openings 40.5 ft. above the water line. Other docks of similar dimensions have been built at Superior, Wis., and other ports on the Great Lakes.

This enlargement of the clock facilities has been made to meet the increasing size of the boats engaged in transporting ore. From the 300-ton schooner of the early clays the boats grew to the size of the E. C. Pope of 2,500-tons burden in 1888 and from there to the size of the H. Coulby of 14,800 tons capacity on a draft of 21 ft. 9 in.

Also the speed of loading the boats has increased from 300 tons in six clays to the record loading of the steamer D. G. Kerr. September 7, 1921, with 12,508 tons of ore in 16½ minutes. It is to be noted that the same cargo of the Kerr was unloaded at lower lake clocks in the record time of three hours and five minutes.

SUMMARIZING

The information shows that there has been a growth in loco motives from 25 tons to 332 tons; in ore cars from four tons to 75 tons; in boats from 300 tons to 14,800 tons. The height of docks has grown from 12 ft. to 84.5 ft. above the water level; pocket openings have grown from 12 ft. to 40.5 ft. above the water level. The length of docks has grown from 100 ft. to 2,300 ft. Trains have grown from 40 tons to 1,200 tons. The ore shipments have risen from a few tons to 375,000 tons daily average during large years.

These figures are of interest to all, who, during the passing years have seen to some extent the changes take place in transportation facilities and should indicate to those of a younger generation the trials and vicissitudes of their predecessors.
DEVELOPMENT OF SAFETY IN THE IRON MINES OF MICHIGAN*

BY WILLIAM CONIBEAR, ISHPEMING, MICH.**

A discussion of this subject may well be prefaced by pausing to inquire what progress has been made since 1911, when many operators, recognizing the seriousness of the accident situation decided to take steps necessary to safe-guard employes. After 25 years’ experience in dealing with this problem, we surely have gone beyond the pioneer stage, and we now should be in position to indicate what success is possible in accident prevention work. To discuss the effectiveness of safety measures in an attitude that assumes that the control of mine accidents in their final analysis has been achieved in a particular mining district, would be to follow the path of error.

The average annual fatality rate of the Michigan iron mines from 1901 to 1910, inclusive, was 4.60 per thousand men employed. For the 20 years’ period from 1910 to 1930 it was 2.75. During the past four years the trend has been downward, averaging below 1.50 per thousand 300 day workers. The non-fatal injury rates also have dropped consistently year after year. Since 1930, the average annual rate has been below 50 per thousand men, as compared with an average rate of 200 for the years from 1910 to 1915. These figures mean that today the death frequency by accidents at our mines is less than one-third as high as it was for the years prior to 1911, and the non-fatal injury rate is one, as compared to four for the earlier years.

To what extent are present-day accidents preventable is another inquiry pertinent to this discussion. A description of all accidents, with full explanation of the circumstances which have a bearing upon the underlying causes or conditions responsible for their occurrence, is essential for an intelligent answer to this question. Lacking this information I avail myself of the accident record of the mines of The Cleveland-Cliffs Iron Company, but, in doing so, it is with an appreciation of the fact that great caution is necessary at all times in the advancing of conclusions from more or less statistical material. It is quite possible that the relative degree of preventability of all mine accidents is approximately indicated by this comparison.

The accident record of this Company compares favorably with the statistics already cited. As its fatality and injury rates have been reduced, the preventable proportion of each year’s accidents increased and the non-preventable proportion decreased. In the past four years it has sustained but one fatality whose occurrence was thought to have been beyond control, and 75 per cent of all non-fatal injuries was accepted as due to failures to conform to well-defined safety standards on the part of both the supervisory force and the employees.

These preliminary statements require no comment. I think that they may be accepted to validate the conclusions that, while great advances have been made in the establishment of safety at our mines during recent years, the opportunity for greater success lies before us.

In fairness it should be said that the effectiveness of safety remedies has not been solely instrumental in the conserving of life and limb indicated by this resume of our accident record. Other factors have played an important role. A period of expansion, yielding to one of contraction in our industry, eliminated many small operators, curtailed expansion and development at the active mines, and provided a choice selection of skillful labor. The inestimable benefits accruing from these factors to safety should be acknowledged.

A summary of the many effective safety practices in vogue at the Michigan iron mines might be carried to monotonous lengths. It will suffice on this occasion to mention a few of those which have had significant application to the safety structure of the various activities closely interwoven in the mining industry.

Many years have passed since the search for mine safety in the Lake Superior district was a haphazard pursuit. Today organized effort for the promoting of better conditions is a policy that permeates shop, pit, shaft, drift, and stope at every mine. Furthermore, the widest dissemination of the most up-to-date data concerning the basic facts of accident prevention work is everywhere freely discussed. I fear no contradiction in stating that the iron mine, or copper mine, does not operate in the Upper Peninsula of Michigan where the importance of a sound safety program is not accepted on par with production. I know no operator who is unwilling to fall in line with the march of progress in combating danger and correcting unsafe practices.

In the experimental days rules and regulations and mechanical devices may have been looked upon as synonymous of safety. We know from experience that mechanical appliances will always be absolutely essential as long as the possibility exists that an accident may occur from a certain danger and there also exists the probability that it will be eliminated by the installation of a simple device. As experience showed that the causes of many accidents were traceable to unsafe ways of doing work, negligence, and indifference, standardization of mine equipment and mine operations became imperative. In the formulation of standards, the active co-operation of the foremen and bosses was of incalculable value, and it has resulted in the desired effect of reducing accidents. Incidentally, it is worthy of note to mention that the economic dislocation in the iron industry of recent years has caused no relaxation in the maintenance of rescue apparatus and the rigid enforcement of standards for the prevention and fighting of underground fires. Contrary to expectation, the standard prohibiting smoking in mines caused little

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**Assistant Superintendent, The Cleveland-Cliffs Iron Co.
dissatisfaction among workmen and its enforcement has brought less violation than has characterized other standards.

Nearly all the iron ore mined in Michigan is obtained underground, and top-slicing and sub-level caving are the principal methods of extraction. These mining methods are particularly favorable for guarding against accidents by falls from roof or back. As there is an abundance of forests in close proximity to the mines, timber is always available for securing working places. Much care is taken in culling that which is defective and under-size. In many mines steel is used in the advancing headings to re-inforce overhead timber. Better illumination has aided in reducing underground hazards. There are five mines on the western end of the Marquette Range in which safety electric lamps are used.

Our days of shallow mining, excepting by open pits, apparently have passed. The transportation of men into and out of deep mines involves considerations which must be dealt with properly by experienced mechanical engineers. That shaft accidents at our mines have been comparatively few bears testimony to the constant concern that management has paid to this occupational hazard. Hoisting cables, over-winding and slack rope devices, shaft runners and cage fixtures are systematically inspected by competent foremen, whose reports reach the higher officials. To ride in a cage, with the door open, is a serious violation against safety in most mines, and the eager detected in committing such an offense knows well that a lay-off awaits him at the end of the shift. When inspecting or repairing shaft compartments, security from falls is found by wearing belts.

We have long appreciated that the relativity of accident frequency from electrical hazards is about as dependent upon the installation of the equipment as it is upon the operating functions. So-called “hay-wire” work is no longer tolerated in our mines; it has gone to the condemnation pile. A deep mine cannot be operated efficiently these days in our district without modern electrical appliances and the service of competent electricians. Our standards, both in respect to equipment and operation, are in conformity to approved electrical codes.

The hazards incidental to the use of explosives are common to mining everywhere. We found that well-defined standards for the handling and exploding of detonators and dynamite are guides pointing the way to freedom from many of the accidents which characterized this work in the past. Of course, like all standards, their effectiveness depends upon leaving no leeway for deviation from safe practices.

The physical examination of workers, as an index by which they may be placed in positions where they do the work they are physically capable of doing, has not been avoided. The need of supplying a source of fresh air in all working places underground, both as regards health and labor economy, was pointed out to us as early as 1912 by the U. S. Bureau of Mines engineers. Ventilation, by mechanical control, was started at that time and is now being carried on very vigorously in all deep mines. During the past two years the iron industry of our district has had the service of a very capable expert to direct our effort to correct the unwholesomeness that may exist in the mine air due to dust contamination. This investigation and the corrective measures are being pushed vigorously throughout the entire field.

Our foremen and bosses, representing management and interpreting its safety policy, fully appreciate their responsibilities as they make frequent visits daily to the working places. Educational processes and monetary awards are not the only forces that urge them forward in the direction of safety, for the security of their positions is at stake if they fail to measure up to the high standards of administration which are now demanded of them.

I now turn to that phase of my subject which at times is most vexatious and arduous, namely, the art of controlling the actions of men as they pursue their various jobs with the association of occupational hazards. Men’s talents and ability are often sorely taxed in the search for perfect regulated individualism and perfect team work, without which accidents are inevitable. To merely enumerate the great number of means that have been utilized with these objects in view would require much time. All the influences that are found in advertising propaganda, awards and bonuses, discipline, and other measures, have been brought to bear upon our men in order to produce safe workers.

It has been said that “progress of an appreciable kind, in any department of social activity and achievement, takes place only and when, and in proportion, as the men who are working to produce a desired result show superior capacity and efficiency in work for that result,” and also “that every business, in fact, every industrial enterprise, succeeds or fails, not according to the amount of average labor involved in it, but according to the talents and energy by which labor is directed.” Of mine safety, combining as it does both social and industrial attributes, it likewise may be said that in the domain of this field of human endeavor the mine executive is the agent of progress by virtue of the results, being what he is, he causes to be produced by others.

I think that is the secret for the success that has come to those mines credited with unusually good accident records.
Correct blasting procedure is fundamental to efficient and economical mining. This applies not only to underground work but to open cut mining as well. On the Mesabi Range, the mining complications are so numerous that many operators are studying blasting method with minute care, realizing that successful operation depends on properly blasted ore banks.

Summarized, the following advantages are gained by correct blasting methods:

1—Properly blasted ore banks give increased tonnage per shovel, and consequently increased production.

2—Properly blasted ore banks give better breakage, thus reducing maintenance and shovel costs.

When these advantages are correctly understood and applied, only efficient management is needed to achieve the lowest possible cost per ton of ore removed.

The following blasting methods, illustrated by sketches, while developed for a particular operation, can be applied generally to most wash ore problems that may be encountered on the Mesabi Range.

*Representatives, Hercules Powder Co.

**WELL-DRILL HOLES SHOT IN SINGLE ROWS**

This method, illustrated by Sketch 1, consists of a single line of holes along and back from the face of the bank to be blasted. The spacing and burden on the holes are dependent on the height and character of ore bank. The holes are drilled three to five feet below grade of working elevation and are shot simultaneously.

**SKETCH NO. 1—A SINGLE ROW SHOT—THE BURDEN PER HOLE ON THIS SHOT IS 15x20x45 DIVIDED BY 27 OR 500 CU. YDS. THE ESTIMATED POWDER FACTOR IS ½ LB. PER CU. YD., RESULTING IN 2.50 LBS PER HOLE. RECOMMENDATIONS FOR CHARGES ARE 40% L. F. GELATINE FOR DECK LOAD OPPOSITE THIN BEDDING IN WET HOLES AND HERCOMITE BAG IN SIMILAR DRY HOLES. 60% EXTRA L. F. GELATINE OPPOSITE THICK BEDDING OF WET HOLES AND GELAMITE OPPOSITE THICK BEDDING IN DRY HOLES.**

The method of firing is either by electric blasting caps or by Cordeau-Bickford fuse, depending on the number of holes to be blasted and the method of loading. Cordeau-Bickford is usually preferred for well-drill blasting. For detonation it requires the use of a blasting cap and once set off the explosive wave travels throughout its entire length with a uniform velocity of 17,500 ft. a second. For this reason, Cordeau is well adapted for well-drill shooting, as it acts as a continuous detonator throughout the length of the explosives charge. When properly connected, any number of holes can be shot. Where it is necessary to deck load (split the charge) Cordeau simplifies the work of loading, since it passes through each section of tamping and explosive.

This method of blasting, when applied to wash ores of the Mesabi Range, is illustrated by Sketch 1. It is adapted for the 120-ton, full-revolving shovel, with a 4-cu. yr. bucket capacity. It will be noted that straight column loading is recommended and that holes are not sprung. The reason for this is to have the powder working through the ore bench, giving it a shattering effect instead of lifting it up as would be done were the holes sprung and the main charge put in the bottom of the hole. It is also emphasized that when chambering in ore of this type, the formation often caves, resulting in lost holes. To dig properly, the ground must be turned and shattered, and distribution of powder along the bore hole is essential to good breakage.

Wash ore usually consists of seams of rock and seams of ore, the rock being barren and varying in thickness from a few inches to several feet. If a proper log is taken of the drill hole it can be used when placing the powder charge. By stemming the ore seams and placing the powder charges in the rock, the rock will be well broken and the ore disturbed sufficiently for efficient shovel operation. This will permit much of the rock to be combed out.

Shooting single rows in hard ground is an efficient way of blasting, but a number of difficulties are encountered. One objection is that before shooting, the loading tracks must be removed. This amounts to considerable labor.
because shots are made at so relatively short intervals. Another objection is the difficulty of maintaining a uniform burden on toe and crest. This is difficult to do from one blast to another, as the back break may vary and hence leave an excessive toe for the next shot. This necessarily means that the toe must be removed before the next shot can be made.

**WELL-DRILL HOLES—SHOOTING SEVERAL LINES OF HOLES AT A TIME**

This method is similar to the one previously described and is illustrated by Sketch 2. An advantage of this procedure is that over a given area only one removal of the loading track is required and there is but one back break. It also permits a larger tonnage to be broken at one time and prepared in advance of shovel loading. This method can be enlarged upon to the extent of what is known as a “field shot” where several hundred holes are shot at one time.

![Sketch 3](image)

**SKETCH 3—THIS COVERS DETAILS OF TWO BLASTS—ONE IN A 60-FOOT BANK, THE OTHER IN ONE OF 80 FEET.** **THE ESTIMATED POWDER FACTOR IS 35 LBS. TO 65 LBS. PER CU. YD.** **THE BURDEN PER TUNNEL 60-FOOT BANK WAS 7,000 CU. YDS.** THE 80-FOOT BANK TUNNEL BURDEN WAS 16,600 CU. YDS. **THE LOWER BENCH REQUIRED 3,500 LBS. OF EXPLOSIVE, THE HIGHER ONE 8,300 LBS. BASED ON ½-LB. A CU. YD.**

This method would be highly desirable for the operation of a 120-ton, full-revolving shovel with the bank ranging in height from 35 ft. to 40 ft.

**TUNNEL BLASTING**

Tunnel blasting is illustrated by Sketch 3. Basically, it consists of a tunnel, or series of tunnels, driven into the bank from the working elevations. “T’s” or crosscuts in which the powder pockets are placed are driven at right angles to the main tunnel. It is essential here that the height of bank be greater than the length of the main tunnel. The ideal bank should be approximately 60 ft. for the conditions encountered on the property illustrated. In a low bank the relative cost of tunnel driving is excessive because of the small amount of material displaced.

![Sketch 4](image)

**SKETCH NO. 4—THIS PLAN DESCRIBES A TUNNEL BLAST AT A WASH ORE PROPERTY. THE BENCH HEIGHT AVERAGED 35 FEET.** **NOTICE THAT THE LINE OF TUNNEL PARALLELS THE BENCH FACE.**
The tunnels serve two purposes. They not only can be used for blasting the area, but they give the operator a better knowledge of the area to be mined. This method calls for large size equipment and in the description here it is understood that a 300-ton, full-revolving shovel (bucket capacity 8 cu. yd.) would be available for work on 60 to 80 ft. banks. Although this method is comparatively new on the Mesabi Range, it now is being used with success.

It will be noted that the pockets farthest from the face are loaded heavier than those nearest the face. The reason for this is that the nearer holes are considered as relief holes, while those farthest back to most of the work.

Several of the accompanying photographs in this article are of a recent tunnel blast at a wash ore property. This tunnel is parallel to the shovel cut instead of at right angles to it. However, this is of no consequence as it is just another variation of this type of blasting. The height of the bank is only 35 ft., which is about the minimum height that can be shot in this manner. In fact, it is the lowest bank that has been shot at this property. The blast, however, was very successful.

Well-drilling in this type of formation is not considered satisfactory when compared to tunneling and it does not compete with tunneling even at this low bank. In higher banks, the tunnel method shows to still better advantage.

Probably the most economical system is the 80 ft. to 90 ft. bank shot by the tunnel method, loaded out in two 45-ft. lifts. This method of blasting is largely dependent on size and type of shovel used. It is best adapted to the large-size, full-revolving shovel.

Experience has shown that in this method of blasting, the rock is broken to a size that can be handled easily by a shovel of this type, hence secondary blasting is virtually eliminated. Because in this formation the ore seams are between barren rock layers, large size rock serves a purpose because it can be combed out, thereby resulting in a higher grade product for loading.

**RELATIVE COSTS**

While costs can be estimated only roughly, the following comparisons are offered. Well-drilling in this formation varies from 35 cents to $1.50 a foot. One of the constant hazards in well-drilling is the caving of the softer ore in die drill column which forms a cushion between the bit and the bottom of the hole; this slows the drilling speed. As against well-drilling cost of 35 cents to $1.50, tunneling cost averages $2.00 a foot driven.

On a 60-ft. bank shot, which required 210 ft. of tunneling, the cost at $2.00 a foot was $420.00. Had the shot been well-drilled the footage would have amounted to 1,280 ft. Dividing $420.00 by 1,820 we see that the proportionate well-drilling allotment per foot would be 23 cents, which is even below the low limits of actual well-drilling costs.

On an 80-ft. bench shot is 18 cents. Even in a 40-ft. bench, the well-drilling cost would amount to about 40 cents—just over the minimum level of actual cost.

In the formations considered in this article there is very little choice between the tunnel and well-drill costs on a 40-ft. bank, although it is generally conceded that the well-drilling cost will be higher. On the 60-ft. and 80-ft. banks, however, there is no doubt that the cost is in favor of tunneling.
THE ELECTRIC FURNACE SOLVES MINING PROBLEMS

BY GRANT FITCH, ISHPEMING, MICH.*

With the introduction of the detachable jack hits to the mining industry The Cleveland-Cliffs Iron Company of Ishpeming, Michigan, made what is believed to be the first installation of an electric furnace for heat treating the new bits and drill rods. This installation was made at their Cliffs Shaft Mine in April, 1935.

Suitable quenching tanks and a removable rack for supporting the drill rods while in the furnace complete the installation.

A brief description of the special tank for quenching bits will be of interest. This tank is arranged with a screen limiting the depth of immersion to ¾-inch. A screen cover is also provided, with spaced holes to assure proper spacing of bits and hold them upright. A cold water outlet is located under each spaced hole to assure adequate circulation for each bit.

Space heaters are used to heat the drawing tank so that complete electric operation is obtained. Fig. 1 shows the complete installation.

OPERATION

The time switch is set for 5 130 a. m., the control to the desired temperature, and at 7 a. m. the furnace is automatically ready for the first charge.

The detachable bits are loaded two at a time with tongs until 100 are placed in the furnace, leaving a space near the door and at the back. The control automatically brings the temperature back to 1450 degrees F. and holds this temperature. At the end of 30 minutes the bits are removed in lots of 10 to the special quenching tank. After quenching they are transferred to the electrically heated drawing tank and drawn in boiling water. The time cycle requires approximately two hours per charge of 100 bits.

The shank ends of the drill rods are heated for one hour at 1500 degrees F. and quenched in oil.

The thread ends are heated to 1500 degrees F. for one hour, quenched for seven seconds in cold water, and transferred to the oil bath. They are next reheated for one hour at 700 degrees F. and slowly air-cooled.

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EQUIPMENT

A standard type H-36 furnace was selected which was rated at 26-k.w., 220-V., single-phase, 60-cycles. The inside dimensions are 18 inches wide by 36¼ inches deep by 12 inches high. This was equipped with a standard control panel and Leeds & Northrup Micromax recording and controlling pyrometer for automatic temperature control. The graphic chart provides the operator with a written record of time and temperature. A time switch was also included to provide automatic pre-heating.

It is interesting to note the rack arrangement used for supporting the drill rods in two layers of 12 each so that
24 rods can be heated at one time. Fig. 2 shows this method of loading in detail.

In conclusion I want to stress again the uniform results obtained with automatic temperature control. The operator is relieved of continuous manual regulation and is free to lay out his work and assume other duties. The noiseless operation of the electric furnace greatly improves working conditions, and the installation as a whole reduces fire hazard and promotes safety. The cost of operation is dependent on local power rates but in general compares favorably with oil and gas and overall operating costs will be reduced due to the factors outlined. No data can be given on maintenance expense as there has been none to date.

DETACHABLE BITS AT THE CLEVELAND-CLIFFS IRON COMPANY’S CLIFFS SHAFT MINE ON THE MARQUETTE RANGE

BY C. J. STAKEL, ISHPEMING, MICH.*

The Cliffs Shaft Mine of the Cleveland-Cliffs Iron Company is located on the Marquette Range at Ishpeming. The mine produces a hard ore, the first shipments having been made a little over 50 years ago, in the early eighties. On the property are two vertical shafts, “A” and “B,” each 1,000 ft. deep, from which 15 levels, spaced approximately 50 ft. apart, run off in various directions. The first level is located at a depth of 350 ft. below the collar of the shaft. The mine workings cover an area about one-half mile north and south by 1½ miles east and west. The equipment at the mine is capable of producing 500,000 tons a year.

The hard ore occurs in the Upper Marquette Series of the Huronian Period and is overlain by quartzite and slate, with a footwall consisting sometimes of jasper, at other times of siderite and also diorite. The formation carrying the ore is an irregular bed, badly folded and faulted, and varying in thickness from a few feet to 125 ft., with an average thickness of 30 to 40 ft. in the larger ore bodies. In general, the ore occurs in a syncline with an axis running east and west, and with a downward pitch to the east and west from the high point near the middle between “A” and “B” shafts. The ore is a hard, specular hematite; and this, together with the jasper and siderite that must be drifted through to reach the ore bodies, makes the drilling problem in the mine difficult. Because of the many folds and faults, the original ore body has been split into many separate sections, and the various lenses are scattered throughout the property on all of the 15 levels. This accounts for the large amount of rock drifting and raising that must be done each year to maintain adequate ore reserves.

In the fall of 1934 the management decided to run a series of tests with detachable bits. In certain contracts in both rock and ore, the procedure was to alternate the drilling procedure by using detachable bits one day and the ordinary mine steel the next day. In this fashion, tests were made in four contracts, two of which were in “A” shaft and two in “B.” These tests proved conclusively that increased drilling speed, with consequent increase in footage, would result from the use of detachable bits. Furthermore, the cost per detachable bit proved to be lower than that of ordinary mine drill steel. As a result of these findings, it was decided to introduce the detachable bits generally throughout the entire mine.

*Superintendent, Cleveland-Cliffs Iron Co.

When the tests were started, 60 tons of 1¼-in. round hollow and hexagonal drill steel was in use underground. It was decided that it would be better to reduce the amount of steel used underground by making replacements from mine steel to detachable bits gradually, concentrating on the introduction of the new bits in the development contracts where the transportation of the old mine steel used up an appreciable portion of the miners’ working time each day.

Because the transition from the use of ordinary mine steel to detachable bits could not be accomplished without expert supervision, this job was delegated to a shift boss and a miner who had had considerable experience with the use of detachable bits. These two men were instructed to teach the miners that were to use the detachable bits exactly how to attach the bits and detach them from the rods, and to take particular care not to deface the skirt of the bit, which is comparatively soft. They also were to caution the miners not to wear the bit down so far as to impair its cutting qualities unduly, because of the consequent reduction in cutting speed, high regrinding or resharpening cost, increased air consumption and undue wear and tear on the drilling machine. It was also the duty of the shift boss and the experienced miner to see that the miners were trained how to gauge the reground bits properly, and furthermore, to teach them to keep the threaded ends of the rods covered with a discarded bit in order to prevent thread injury.

The training crew, so-called, was assigned certain contracts, and each contract was given at least a week’s personal supervision before being left to its own devices. If it was found necessary to train the miners in the use of the new detachable bits longer than a week, the two men doing the training remained with the contract miners until they became proficient in the use of the detachable bits. The training of the miners and the transition to the new method has now reached a point where 80 per cent of the 80 contracts in the Cliffs Shaft Mine are using the new type of bit.

Each working place is furnished with a metal box 18½-in. long, 13½-in wide, and 9¼-in. deep, containing eight compartments in which the bits of each gauge are kept. A photograph of this box is shown elsewhere in this article—in Fig. 1.) On the outer edge of each compartment there is a hole which acts as a gauge for the bits to be kept in that particular compartment. The gauge changes are 1/16-in. This box is kept at all times.
in the working place near the miner when he is drilling. Each mining contract is also furnished with another set of metal boxes 6-in. wide, 14-in. long and 5-in. deep, which are provided with a cover and a handle to make transportation easy. In these boxes are placed the dull bits to be taken to the shop for redressing, and in them also the sharpened bits are returned to the working place. All these boxes are marked on the end with the particular contract number the bits are intended for, and on the top or cover with the level number and the shaft in which the contract is located. Filled with the resharpened bits ready to go underground, they are arranged by shafts on shelves, as shown in Fig. 2. Each working place or contract usually has an average of about 65 bits and 12 rods per drilling machine. The drilling machines have 3¾-in. and 3½-in. pistons, operate with 90-lb. air pressure, and are usually mounted on tripods in the stopes. In the drifts and raises, columns are used for mounting the machines. Every miner is provided with a soft or annealed steel hammer with which to tap the bits on or off the drill rods.

A shank facer has been installed which uses a very narrow grinding wheel for sectioning the bits in order to check the temper.

Included in the equipment is also a water-jacketed oil-quenching tank with a capacity of three full barrels of oil. There is also a cold-water quenching tank for rods and a hot-water tank in which bits are placed after the cold-water quench. The water in this tank is heated electrically. One of the most important pieces of equipment installed is a specially constructed tank for quenching the bits which are being retempered. (A drawing showing the details of the construction of this tank also accompanies this article. See Fig. 7.) This tank is equipped with a manifold placed near the bottom of the tank, which, because of its flexible hose connections and adjustable legs, can be raised or lowered. The manifold consists of 12 water jets turned upward, which produce geyser action in the water at 12 different points, a very important feature because sufficient turmoil in the water at the bottom of the bit is necessary to prevent the formation of steam pockets and the consequent production of soft spots in the cutting edges of the bit. When the bits have been removed from the furnace, they are placed on a movable tray which can be raised or lowered to give the submergence necessary to produce proper depth of temper. Too deep submergence produces a temper which is too hard for the skirt of the bit. This in turn would cause undue wear of the threads on the rods, with consequent damage to the threads of the detachable bits. Too shallow submergence means a soft bit. The sides of the tank surrounding the tray are equipped with overflows so that the water is led away in four different directions, to insure a plentiful supply of fast-moving cold water. After each batch of bits has been re-tempered, one bit is taken from the lot, cut in two on the sectioning machine and immersed for three minutes in sulphuric acid, which clearly etches the face of the bit so that the operator of the furnace can see at a glance whether that particular lot of bits has been properly re-tempered.
FIG. 3—VIEW SHOWING FOUR BIT DRESSING MACHINES MOUNTED ON BENCH. SHELVES FOR BIT BOXES ON RIGHT HAND SIDE.

As shown in the accompanying photographs—Figs 2 and 8—the shop also has a rack containing compartments for various lengths of threaded rods and shelves on which the boxes containing the sharpened bits are placed. These boxes are assembled in groups to simplify the handling of them in each shaft.

In addition to the equipment described above, the following equipment is necessary for forming shanks and lugs and threading the drill rods: A drill sharpener for forming the shanks and lugs, an oil furnace, a power saw for cutting the drill steel in proper lengths, and a bolt-cutting machine equipped with special dies for threading drill rods.

The crew in the detachable bit shop numbers five, four men on the four grinders and a fifth acting as foreman, who operates the electric furnace and takes care of the tempering of both the bits and the rods. The men operating the grinders dress about 140 bits per man per 8-hour day.

RODS AND BITS

Most of the drill rods are 1¼-in. in diameter; but some 1½-in. hexagon and some ¾-in. hexagon rods are treated for other mines. The detachable bits used at the Cliffs Shaft Mine run in two sizes—2-in. and 2¼-in.—both center hole, 4-point, 105-degree cutting angle, 3½-degree taper, the wing being ¾-in. wide and ¾-in. long. The earlier bits had a very slight groove between the wings, but the new bits have a very deep groove. The first experiments made in very hard jasper and siderite with the bits were based on the theory that the very hard pulverized material at the bottom of the hole actually performed some of the cutting action, as this broken material was being rolled over the face of the ground in the bottom of the hole by the wings of the bit. Later experiments disproved this theory, and for that reason the deep groove bit was introduced, which allows the cuttings to escape from the bottom of the hole as quickly as possible. The 2¼-in. bit is used for very hard ground; whereas the 2-in. is used for the softer material like the slates and dikes.

RECONDITIONING DETACHABLE BITS

The dull bits are delivered from the mine to the drill-sharpening shop each morning in iron boxes on which are marked the contract numbers from which the bits came. The bits are dressed, returned to the same box, and sent back into the mine late in the afternoon of the same day. Discarded bits are replaced with new ones. In the meantime the miner has another supply of bits underground in a spare box; and if the ground he is drilling in is unusually hard, he is given still another supply.

FIG. 4—ELECTRIC FURNACE FOR HEAT TREATING BITS, SHANKS AND THREADS. BIT QUENCHING TANK IN FOREGROUND. HOT WATER TANK ON EXTREME LEFT. PYROMETER CONTROL IN BACKGROUND.

The re-sharpening operation consists of grinding the cutting edge on the grinding wheel, which is 1½-in. x 12-in., and then gauging the same bit on the 1¾-in. x 12-in. gauging wheel. The grinding wheels have to be dressed from time to time in order to true up the angle of the face of the forming wheel and to square the face of the gauging wheels. This procedure is accomplished with a special dressing tool. Any bit which shows evidence of having been worn down close to the edge of the hardened surface is placed in a special container to be re-tempered. As soon as a considerable number have been accumulated, a batch of bits is re-tempered. To perform this operation, the electric furnace is heated to 1450 degrees F.; 75 bits are then placed on the hearth of the furnace and left there for 30 minutes to be heated. A little powdered charcoal is also placed inside the muffle of the furnace to maintain a reducing temperature and thus prevent decarbonization. The bits are then extracted two at a time until 12 have been removed; during this operation the temperature of the furnace usually drops about 15 degrees because the furnace door is open. The furnace is then closed and permitted to return to a temperature of 1450 degrees before the next 12 bits are taken out. When the bits have been removed from the furnace, they are given the cold-water
quench 34-in. up from the face of the bit in the cold-water quenching tank previously described. When the bits show a blue-black color, they are removed from the quenching tank and totally submerged in boiling water, where they are allowed to cool. They are then removed and placed in the storage bins ready to go back into the mine. From 300 to 350 bits a day are usually re-tempered during an 8-hour shift.

**DRILL ROD SHANKS AND LUGS**

Shanks and lugs are formed on drill steel in the usual way with a drill-sharpener and are then sent to the detachable-bit shop for tempering. The tempering of the shanks is done by heating the electric furnace to 1480 degrees F. Twenty-four rods are inserted at a time, as shown in illustration Fig. 2, Grant Fitch’s paper, page 196, and they are so placed as to make it possible to heat 12-in. of the shank end of the rod. This is done in order to get the overlapping heat necessary to prevent breakage. In other words, the rods are inserted into the electric furnace further, and more of the rod length is heated than is customary in the oil furnace prior to the forming of the lugs. The shanks are kept in the electric furnace for an hour at a temperature of 1480 degrees F.; are then removed from the furnace one at a time, placed in quenching oil, and allowed to cool. The usual number of rods treated in a shift of eight hours is 150.

**DRILL ROD THREADS**

After the drill steel as it comes from the manufacturer has been cut into suitable lengths, the end of the rod to be threaded is heated and placed in lime to anneal the steel. At the present time the rod is then centered in a lathe, refaced, trued to the proper diameter, and chamfered, and its end squared. Then it is taken out of the lathe and put in a special bolt-threading machine equipped with the proper dies which cut four left-hand threads to the inch, 250 pitch. In a short time, however, this procedure will be changed, as equipment is now being installed which will make it possible to take the annealed rod and place it in a machine that will automatically perform all the proper operations and turn out a completed rod ready for heat-treatment. Under the old method, from 70 to 75 rods can be turned and chamfered in a lathe in an 8-hour shift, and 150 rods can be threaded in the bolt-threading machine in an 8-hour shift. The dies used in the bolt-cutter usually cut the threads on from 650 to 670 rods before it is necessary to replace them. No attempt has been made to dress the old dies, new ones being purchased when the old show signs of wear. The bolt-cutting machine is run at a very low speed in order to produce a very smooth thread and to lessen the wear on the dies. Too much attention cannot be paid to the lathe work in connection with the threads, and special attention must also be given by the operator of the bolt-cutter because unless a perfect thread is produced on the drill rods, the miner will have untold grief underground. In fact, it seems safe to say that the successful use of detachable bits depends largely on the perfection of the threads on the drill rods. That applies both to the cutting of the threads and the tempering operation.

**CALIBRATING ELECTRIC FURNACE**

Inasmuch as the temperature control in the operations heretofore described is so important, an occasional checking of the furnace pyrometer is necessary in order to ascertain whether the furnace heats are actually as shown on the recording pyrometer. Twice a year, therefore, the furnace pyrometer is checked as follows:

First a test bar is prepared consisting of a piece of tool steel 2-in. in diameter and 9-in. long. A hole ¾-in. in diameter is drilled in one end of the test bar to a depth of 4-in. The test bar is then placed inside the electric furnace, fire brick being used to support the test bar at about the center of the back of the furnace. The thermocouple is then inserted within the steel and the test is begun. The heat is turned on, and after about one hour