AIR HEATING PLANT—DAVIS-GENEVA MINE

BY F. E. BOYD, DULUTH, MINN.*

The Davis-Geneva Mine is an underground iron mine, operated by the Oliver Iron Mining Company and located about 2½ miles east of Ironwood, Michigan. Prior to 1937, it was ventilated chiefly by gravity with a few portable blowers for the working places. In June of that year, a complete system of mechanical ventilation was placed in operation. The installation was simplified by the abandonment some time previously of the Davis Shaft as a hoisting shaft, leaving it free for installation of the ventilation plant at the collar. The flow of air is down the Davis Shaft and up the Geneva Shaft.

The plant is housed in a steel framed galvanized sheathed structure, erected on concrete foundations and built directly over the shaft collar. The attached photo engraving shows the general view of the plant. Drawing No. 13140, Sheet 3, shows the elevations of the building. The walls and roof are of "sandwich" construction, i.e., they consist of an outer and inner sheet of galvanized steel, separated by 1½-in. thick spacers, and the space is filled with 2-in. Rockwool bats. The details of this construction are shown on Drawing No. 13140, Sheet 4.

Air enters the fan house through a louver opening in the south elevation (See Drawing No. 13140, Sheet 3). It first passes through weather louvers, then through a motor controlled ball bearing; steel framed face damper and two balanced by-pass dampers, then through two banks of Aerofin Flexitube heating coils each consisting of four 18 tube wide sections 10-ft. long, thence into the fan. Aquastats and Webster 026 traps are placed on the drip lines from the coils. The fan is a Clarage Class II, double inlet, double width unit of non-overloading type and designed to deliver 70,000 c.f.m. against a resistance pressure of 5-in. water gauge. It is of the top horizontal discharge arrangement and its shaft is mounted on Fafnir ball bearings. The drive is through Texropes from a 100-h.p., 875-r.p.m. induction motor. The power requirements of the fan were but 75 h.p. under the above-named conditions, but a larger motor was used so that additional air could be provided up to 90,000 c.f.m. if desired. The fan is now operating at the increased capacity.

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The heating coils and the fans are housed in an air-light room, contracted of No. 18 U. S. gauge galvanized steel sheets supported on a structural steel frame. The discharge duct of the fan is of the same gauge of material. This duct leads horizontally to the outer wall of the north elevation of the fan house (see Drawing No. 13140, Sheet 3), the opening being provided with a hinged cover. Midway of the length of the discharge duct, a branch turns downward from the underside to
connect to the mine shaft. Two hinged gates or dampers are provided, one in the main horizontal portion of the duct to close the branch and so provide a direct discharge from the fan to the outside of the fan house. The other is in a concrete passage leading to the fan room and serves to divert the air under reversed operation into the fan room. By removing the cover from the end of the main duct, closing the branch damper and opening the damper in the concrete passage, the direction of air flow into the mine is reversed. Normally this flow is down the Davis Shaft and up the Geneva Shaft. Under the conditions of reversed flow, the fan room is under sub-atmospheric pressure so that it is necessary to seal the entrance louvers with canvas closures which are provided for the purpose. Air locks are also provided so that one can enter the fan house and also the mine shaft when the fan is in operation. For details of the reversing dampers, see Drawing No. 13179, and for details of the concrete passage from the shaft into the fan house, see Drawing No. 13139.
A test was made to determine the length of time required to reverse the flow of air into the mine and it was found that this could be done in 2½ minutes.

Steam for the heater coils is furnished by one No. 421 Kewanee firebox boiler designed for operating at 15 lbs. per sq. in. The boiler is fed by a Chicago Pump Company’s No. 1667 vertical condensation pump which receives the water from the heating coils and discharges it directly to the boiler. The pump is located in the fan room. The boiler is also equipped with a McDonnel & Miller No. 306 safety boiler feeder. The stack is of steel 33-in. in diameter x 70-ft. high, of self-supporting construction. The boiler is fired by an S. T. Johnson Company’s size 5½, Type 30-AV heavy duty rotary oil burner, having electric gas ignition and operated by a 2-h.p., 3-phase, 60-cycle, 440-volt motor. The controls are Minneapolis Honeywell. The fuel, which is of the type known as Placedo or Refugio, and which comes from South eastern Texas, is of the following average specifications:

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gravity</td>
<td>23 to 25</td>
</tr>
<tr>
<td>Flash</td>
<td>185 to 200</td>
</tr>
<tr>
<td>Fire</td>
<td>200 to 220</td>
</tr>
<tr>
<td>Viscosity at 100°F</td>
<td>50 to 60</td>
</tr>
<tr>
<td>Cold test</td>
<td>-32°F</td>
</tr>
<tr>
<td>B. T. U. per lb.</td>
<td>10,000 to 15,000</td>
</tr>
<tr>
<td>Sulphur</td>
<td>19% under 2 of 1%</td>
</tr>
<tr>
<td>Sediment</td>
<td>17% under 3 of 1%</td>
</tr>
<tr>
<td>Conradson Carbon</td>
<td>52% under 6 of 1%</td>
</tr>
<tr>
<td>Color</td>
<td>Green</td>
</tr>
</tbody>
</table>

It is delivered in tank cars and unloaded into four reservoirs, converted from old 72-in. x 18-ft. H. R. T. boilers that originally stored oil for use in shop forges. The oil is piped by gravity from these reservoirs to the supply tank in the boiler room and its flow is regulated by a float valve. For location of the oil reservoirs relative to the fan house, see Drawing No. 13696, and for the arrangement of the boiler plant, steam piping, oil supply tank and miscellaneous details, see Drawing No. 13153. Gas for the ignition is furnished by cylinders of Pyrofax, located outside the fan house convenient for connection to the oil burner. (See photographic cut of ventilating plant). The control of air temperature is effected by Barber Coleman apparatus consisting of a ductstat mounted in a dust and moistureproof case located in the mine shaft and operating reversible oil immersed damper motors having built-in relays. Current is 110 volt, 60 cycle.

This control is supplemented by a Bristol 2-pen recording thermometer, Model 340-M, which simultaneously records temperatures of cold air and heated air.

The following tabulation gives operating data of the Davis heating plant for a typical cold weather month:
The power supply to this heating plant is obtained from a 2-circuit, 22,000-volt, 3-phase, 60-cycle, 4/0 strand steel core aluminum transmission line on steel towers, which line was installed when the Davis was an operating shaft. (See photo engraving of the surface plant) A short wood pole line tap from one circuit of this transmission line feeds a 450 KVA capacity, 22,000/440-volt, 3-phase transformer substation located adjacent to the maintenance hoist house. The 22-KV side of this transformer bank is protected by an air break disconnect switch, an oxide film lightning* arrester and liquid type fuses. This substation is 33-ft. 6-in. x 21-ft. 2-in. and is enclosed on three sides by a tubular post woven wire fence, the hoist house encloses it on the fourth side.

This hoist house is a frame, galvanized sheathed structure, 17-ft. 3-in. wide x 21-ft. 6-in. long x 8-ft. join, inside height, and is located 115 ft south of the air shaft. It contains a 48-in. diameter x 36-in. face geared hoist driven by a 150-h.p. 440-volt, 3-phase wound rotor motor equipped with a 440-volt solenoid brake on the motor shaft, and the necessary control equipment. This equipment consists of a contactor panel operated by a drum controller, motor secondary resistors, a Model “D” Lilly hoist controller, together with the necessary telephone, signal and safety equipment. This hoist is used only for shaft inspection and repair, and a foot-operated switch is used to allow the hoistman to energize the solenoid brake to lower the cage slowly on the hand band brake.

The 100-h.p. squirrel cage fan motor in the fan house is controlled by a magnetic compensator, pushbutton starter, and interlocked with the no-volt surface contact aquastats mounted on the Aerofin heating coils. The arrangement is such that if the temperature of the return pipe from any of the heating coils drops below 180 degrees the fan will stop. Switches are inserted in this circuit so that the fan can be operated independently of these aquastats for summer use. The 350,000-c. m. rubber insulated, lead covered cable from the hoist house serving this fan also provides power for the 2-hp. oil burner motor, the 1½-h.p. condensation pump., and the 5-KVA lighting and control transformer. This control transformer supplies no-volt power to the 25-volt face damper motors and control through a 110/25 volt transformer.
AN UNUSUAL CONNECTION SURVEY AT THE NEWPORT-BONNIE MINES

BY L. M. SCOFIELD, LAURIUM, MICH.*

In 1930 a new vertical shaft was started on Bonnie ground, well back in the granite footwall, to replace “K” and “Woodbury” shafts as the hoisting outlet for the Newport-Bonnie Mines. By September of 1934, operations having been resumed after a long idle period, this shaft was down to the 28th or 2593-foot level and a station was being cut at that level.

In order to furnish proper alignment for the station and the required 2200-feet crosscut to the footwall of the iron formation, it was necessary to carry a survey down the new shaft. As no existing surface survey points could be relied upon because of general surface subsidence, it was further necessary to bring a survey up to surface from old underground survey points (known to be stationary in a granite crosscut on the 27th level underground) through the operating shafts, one vertical and one inclined. The total length of the surveys required, exclusive of shaft work, was approximately 9,500 feet. These surveys developed some unusual features which merit recording.

In planning the surveys it was reasoned that if the entire survey were to be carried up the vertical Woodbury shaft, the point totals would be reasonably correct but the bearings would be subject to some error due to plumbing; and that if the entire survey were to be taken up through the inclined “K” shaft, the bearings would be reasonably correct at surface but the point totals would be erroneous due to the difficulty of measuring the vertical angles and distances accurately. Accordingly, it was decided to take the bearings up the inclined shaft and the totals up the vertical shaft on a single wire, and to join the two branches of the survey at surface and carry them as one survey to the collar of the new shaft and thence underground by two-wire plumbing.

Two old established survey points in the granite crosscut from Woodbury shaft on the 27th level where no ground movement had occurred were chosen as starting points. Traverses were then made each way to the 27th level stations of both shafts. A standard, one-minute mine transit was used and all angles were turned four times, read alternately erect and inverted. Measurement of distance was accomplished with a 5/16-inch 200-foot steel tape and all distances were check-taped. In computing, the angular readings were averaged and bearings were carried to one-eighth of a minute. This portion of the survey was done twice, by two engineers, each working independently.

An angular survey was next run up the inclined “K” shaft to carry the underground bearings up to steel pins established on surface. As “K” shaft changes from a 65 degree incline between surface and the 680-foot level, to a 67 degree incline below the 680-foot mark, the shaft survey required a set-up on staging at the knuckle. For the shaft sights, electric lights behind narrow slits were used for sighting points. All angles were turned four times, but no inversion was possible because of the use of the top telescope. This portion of the survey was done independently by two engineers, but using the same instrumental set-ups.

The next step in the process was to carry the underground totals up the vertical Woodbury shaft by plumbing with a single wire. Tinned steel armature-binding wire, size 21, was used to hang a 50-pound vaned weight in a 50-gallon oil drum filled with water thickened with sawdust. As Woodbury is a wet shaft, enough water ran down along the wire to float off the sawdust very rapidly and this method of inducing viscosity was found impractical. Oil would have been useless for the same reason. It was also found that agitation of the water in the drum by drip leaking through a planked shelter and water splash against the wire up the shaft were responsible for much erratic movement of the wire. The wire never did quiet down but kept moving about in a very complex path,—the result of combination of the paths which would be traveled by pendula of many different lengths, corresponding to the distances from the weight to all the nodes marking the overtone vibrations of the wire. The principal movement was of a period indicating a theoretical suspension point about 150 feet above the weight, and swing from the actual top of the wire at surface (2,550 feet above) was of such minor amplitude that it was masked by the short-period swings which it merely modified slightly.

At any rate, the continual movement of the wire necessitated development of some accurate method of determining its central position. This was accomplish by setting a 20"=1" engineer’s scale close behind the wire, reading the extremes of 20 swings through the transit, and averaging the readings. Thereafter the 27th level survey was carried to the wire by turning to the average reading on the scale, which of course was stationary, and re-turning the angle four times as done elsewhere. Distance to the wire was determined by taping at ten different times on the cycle of swing, and averaging the figures obtained.

Tying the wire in to established survey points at surface offered no difficulty as the movement of the wire there was imperceptible. The bearings brought up “K” shaft were then taken over to Woodbury collar by a standard four-angle traverse and applied to the established points carrying the totals brought up through Woodbury shaft. The survey was next extended to New Shaft collar and left there on concreted iron pins protected by padlocked covers to await such time as there should be sufficient room cut out on the 28th level to justify plumbing the new shaft. All this instrument work was performed twice by two engineers working independently.

By February of 1935 the underground work had progressed far enough to warrant plumbing and two wires were hung in the new shaft (in similar fashion to the single wire used in Woodbury) about 15 feet apart. The lessons learned in plumbing Woodbury shaft were

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*Mining Engineer and Geologist.
applied here, and galvanized iron cones were placed over the water-filled drums holding the weights, to cut down the splash effect of the water-drip to a minimum. The cones had 6-inch openings at their peaks to give the wires ample room to swing and this opening was further cut down with boards after the range of swing of the wires became apparent. As a result no trouble was experienced from drip and the wires were undisturbed except from air currents and a slight amount of splash up above in the shaft. Some water came down along the wires but it was not felt that this tended to move the wires to the degree that splash in the drums would.

The survey was taken from the protected pins near the collar and carried to the wires by jiggling in on the wires, sighting a known point, measuring to it, quadrupling the angle involved, and obtaining the bearing of the wires by setting a backsight from the jiggle set-up on the line between that set-up and the known point and then setting up on the known point and turning an angle from another known point to the backsight established.

In order to give the wires time to quiet down they were left overnight and the survey crew then laddered down the half-mile of shaft to avoid setting up strong air currents by running the cage. On arrival at the 28th level it was found that the wires were still moving but more evenly than had the single wire hung in Woodbury shaft in the preceding September. The swings were less erratic, and though the action of the wires showed a combination of many periodic movements, still the resultant was much more uniform and showed more of the influence of the long-period swings. This was undoubtedly due to the fact that there was no through air current in the new shaft, and that there was much less water coming down the shaft and none of that was permitted to splash into the drums holding the weights.

Jiggle set-ups were made on the 28th level station and the technique developed during the work in Woodbury shaft was used to determine the jiggle points. That is, 20'—1" scales were set behind each wire and the extremes of 30 swings were averaged to obtain the midpoints and the transit head was then moved into alignment with the average points on the scales. No correction for parallax was required. On the first wire, 60 swings were averaged but it was found that the average of each successive 20 swings did not vary by enough to make a difference in the jiggling, so 30 swings were used thereafter, leaving a 10-swing margin of safety. Measurement was accomplished by tapping the distance from the jiggle to the wires at ten different times in the swing cycle and averaging the figures.

Once the transit was aligned with the wires, the angles between the wire-bearing and four different fixed points in the back of the station were measured by turning four times. The distances from the jiggle to these points were taped and back-sight points set on the lines to these points. After one jiggle set-up had been completed a second was made about 5 feet farther back from the wires and the whole process repeated. Following the jiggling, each of the four fixed points on the station was occupied and the angles between and distances to all the others determined in the standard manner, so that even if two points should be lost during work on the station, there would still be two remaining from which to extend the survey. All of this work was clone twice by two men working independently, except that on the jiggle set-ups only the head was thrown out of line after the first man was through, saving time on re-alignment of the tripod. After completion of the survey work on the station the cage was rung down and the wires were watched closely to determine the possible effect of any air currents set up by the moving cage on the movement of the wires. None was apparent.

When the 28th level station was completed a crosscut was driven 1,800 feet toward the iron formation at which point it crossed under the 27th level crosscut from Woodbury shaft where the survey was started, and 140 feet below it. It was decided to put up a ventilation raise to hole into this crosscut and, incidentally, to check the survey down the new shaft. The elevation of the 28th level had been determined by rough measurement down the new shaft when that shaft was sunk. For holing purposes, something more accurately tied in with the 27th level elevations was needed. Levels were carried on the 27th level from the starting points to Woodbury shaft and the elevation of a point on the shaft steel determined at the 27th level station. The vertical distance between this point and another on the collar-set at surface was measured with a 500-foot steel tape in the following manner:

Four men were used, two at each end of the tape. A punch mark was made in a shaft studdle a little less than 500 feet above the 27th level point. The tape was held at the 27th level and unreeled up the shaft from the ascending cage which was stopped at the upper mark. The zero-mark on the tape was held on the upper punch-mark and the men at the lower end read the tape at the lower mark. The cage was signalled down to the lower mark to pick up the two men there and bring them to the upper mark. The upper-end crew then took the cage up another 500 feet carrying the tape hanging in the shaft below the cage. As the lower end of the tape reached the lower crew, they stopped the cage with the shaft signal system, the upper crew set another punch mark, and the distance was read off by the lower crew as before. This process was repeated up to the collar and then the whole measurement was made a second time with the crews reversed, to pick up any possible errors. A dead weight of 15 pounds was used to stretch the tape, as manual stretching was found to give results which were too uncertain to be dependable. There was, and is, no assurance that the footage measured was correct, but it was reasoned that if the same method were used going up Woodbury shaft and down the new shaft, the elevations at the bottom would be on a comparative basis.

The elevation obtained for Woodbury collar was used to carry the levels over to an established surface point. Since a very accurate and carefully compensated level
circuit had just been completed, tying in all the fixed surface points, the difference in elevation between the point near the Woodbury collar and one near the new shaft collar, obtained from that circuit, was applied to obtain the working elevation at the latter, and thence, by levelling, to the new shaft collar steel. The new shaft was then measured from the 28th level up to the collar by exactly the same method used in Woodbury shaft. Levels carried out in the 28th level crosscut brought the circuit to the foot of the connection raise which was to hole through to the 27th level.

As the 27th level crosscut was being used as a main haulage outlet, and the connection raise was only desired for ventilating purposes, it was aimed for a point 25 feet west of the 27th level crosscut. When the proper elevation was attained, a drift was driven over to the crosscut to make the actual holing. This work was controlled by a standard four-angle transit traverse from the 28th level station points, carried out the 28th level crosscut to the foot of the connection raise. This traverse was made point-by-point as the crosscut advanced and was then repeated, as a check, all at one time (by the same engineer) when the connection raise was started.

After the actual holing was accomplished, a standard four-angle traverse was carried up the raise to tie the 28th level survey in to the starting points on the 27th level. As the entire survey, with the exception of the work on the 28th level crosscut, and in the connection raise, was done by two engineers working independently, the final tie-in was computed for both sets of results, using the latter work in common for both computations. It was found that one set of notes showed upon closure an angular error of one minute clockwise and an error in total of +0.58 feet in elevation, +0.72 feet in latitude, and +0.54 feet in departure. The other set showed an angular error of one and one-half minutes counter clockwise and an error in total of +0.58 feet in elevation, +1.27 feet in latitude and -0.83 feet in departure. These closures were considered to be highly successful when the length of the survey, the crudeness of the instrument, and the natural difficulties of shaft work were taken into account.

The success of the survey demonstrated the correctness of two methods of procedure which may be of value to other engineers faced with a similar problem. The plan of separating bearings and totals when coming up to surface from the starting points may be credited with a large share in the accuracy attained, and great importance must be assigned to the technique used in centering the wires when jiggling in at the bottom of the new shaft.

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**Chief Engineer of The Castile Mining Company.***

The building is of concrete and brick, construction with outside over-all dimensions of 50'8"x143'-8", the long dimension trending about east-west. It has two stories for 82' of its length and one story the remaining portion. The dry proper occupies the lower story, the floor of which is 14 feet below the grade of the county highway that passes in front of the building. The ceiling height of the dry proper is 13 feet. The second story is at grade with the highway and is used as a garage and

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**Chief Engineer of The Castile Mining Company.

The Castile Mining Company’s Eureka Mine Dry at Ramsay, Michigan, is the newest in the Lake Superior District. Initially occupied August 15, 1938, it incorporates many new ideas in dry design. It was designed by N. Albert Nelson, Architect, of Ironwood, Michigan, in collaboration with the engineering department of The Castile Mining Company. The Eureka Mine is located on the eastern end of the Gogebic Iron Range and has been a steady producer of iron ore since 1890. It is at present the deepest iron mine in the Lake Superior District, the shaft having a vertical depth of 3,362 feet. Oglebay, Norton and Company, Cleveland, Ohio, are operating and sales agents.

The design of change house in the Lake Superior Mining District has gradually improved since the beginning of mining in the area, and today the change house at the Eureka Mine stands as a modern example of an employer’s effort to make the life of the employe a clean and pleasant one. The first change houses, or “drys,” as they are termed locally, were frame sheds containing rows of steam pipes upon which the clothes were hung, and wash basins where the miners could attempt to remove the tenacious hematite dust from the exposed portions of the body. Wooden lockers were introduced. The shower bath came, but was rather slow in being accepted by the surprisingly modest miners. The wooden lockers were replaced by steel. The buildings became permanent and well-built structures. Gradually the facilities for the miners were improved.

The thought uppermost in the minds of the men interested in the design of this dry was the erection of a fireproof building that would best serve the needs of miners working in a wet, dirty mine. They wanted to eliminate the intermingling of men dressed for street and miners working in a wet, dirty mine; to dry the wet underground clothes without the use of excessive heat so often nearly unbearable in the old dry; to isolate the hematite dust so offensive to all, especially the miner’s housewife; and, to eliminate the dust created by the handling of dirty mining clothes after drying; in short, it was their desire to create a modern dry that would be readily adaptable to possible future conditions.
warehouse. The second story portion is of brick construction, harmonizing with the other buildings on the property, whereas the dry proper is entirely of concrete. The entire building is without windows. The roof of the two story portion is steel decking, and that of the single story portion is concrete, both being insulated and covered with a built-up, four-ply pitch and gravel roofing. All partitions are of hollow tile and all interior walls of the dry proper are plastered. The doors to the garage and warehouse, as well as the main entrance door, are of wood. All interior doors are of steel.

The entrance to the dry is at the east end of the building, from which an inside stairway leads down into the vestibule. This concrete stairway is covered with a plastic compound to insure against slipping. The dry is divided longitudinally into halves. The south half is termed the “clean side” or “street side” and the north the “mine side” or “dirty clothes side.”

The “street side” consists of the vestibule, dryman’s supply room, combined washroom and toilet room, large clean clothes locker room, the captain’s and foremen’s clean clothes locker room and office. In the vestibule are the time clock and bulletin board. The dryman’s room is used as a supply depot and also as a timekeeper’s office and pay office. The combined washroom and toilet room is equipped with two spray type wash troughs, four toilets and a urinal. The large locker room has 287 lockers that are used only for clean clothes. All lockers are permanently set in the concrete floor and have wooden benches attached. The captain’s and foremen’s room has 24 lockers and is used daily as a meeting place for all departmental heads.

The “mine side” of the dry consists of the large dirty clothes room, shower room, toilet and laundry, electric lamp room and small dirty clothes room used by the foremen. The large dirty clothes room is about 77’ x 24’. The underground clothes are exposed to drying and aeration by being suspended from the ceiling on specially designed hooks. The room contains 262 hooks with one hook allotted to a man. Each hook, so designed that clothes properly hung will dry thoroughly, is attached to an endless chain passing through a pulley suspended from the ceiling. The chains are equipped with rings which permit the locking of the hooks after they are pulled up; each man is supplied with two combination padlocks of the same combination, one for the clean clothes locker and one for the dirty clothes.
hook. The suspension structure for these hooks is of pipe and steel construction and is hung; from the ceiling, the hanger straps having been set in the roof slab at the time of pouring. Each hook has a small wire basket attached that is suitable for holding incidentals that the miners prefer to remove from their clothes. Between each row of hooks is a stationary bench constructed of steel, with a plank top. Adjoining the dirty clothes room is the shower room. This room is about 23' x 19', with a 9' wainscoting of glazed brick on the walls; it contains 33 shower heads, each equipped with mixing valves. All hot water supplying the dry passes through a controlled mixing valve, thus eliminating the hazard of scalding. A 2" pipe railing parallels the walls of the room about 1' above the floor and 1' out from the walls to facilitate the washing of feet. Liquid soap dispensers are distributed throughout the room; the liquid soap is piped from a mixing and storage tank located in the garage above the shower room. From the shower room a door leads into the hallway connecting the vestibule and the large locker room on the street side of the dry. The toilet and laundry room is equipped with two toilets, urinal and small lavatory, and laundry equipment consisting of an electric washing machine and stationary laundry tubs. The laundry equipment is used by the men for the washing of underground clothes. The lamp room contains the charging racks for the electric cap lamp batteries and is also used by the dryman in giving out supplies to the men after they have changed to underground clothes. The small dirty clothes room used by the foremen is similar in design to the large room and contains 18 hooks. Between this small dirty clothes room and the foremen's clean clothes room is located a small shower room and toilet room. The shower room has two heads and is finished with glazed brick, in the same manner as the larger shower room. The toilet room is equipped with one toilet and a lavatory.

The dry is connected to the shaft by a concrete tunnel 70' in length. This tunnel has two entrances, one directly opposite the lamp room and the other leading from the foremen's dirty clothes room. Because of this arrangement, all men entering or leaving the mine must pass the foremen's room giving the foremen an opportunity to contact anyone they may wish to see. The end of the tunnel is extended a few feet beyond the main entrance into the dry; here are located four short lengths of hose for use by the men in washing their boots. Along one side of the tunnel is a permanent bench where the men sit when waiting to go on shift. The tunnel eliminates the necessity of the men subjecting themselves to severe temperature changes when going on or off shift. Records show that the men lost less time due to colds this past winter than ever before, and the tunnel must have been a prominent contributing factor.

As the men enter the dry, upon coming to work, they punch the clock in the vestibule, then pass through double doors into a short hallway and into the large locker room where they remove their street clothes. From here they enter, through a one-way door, the large dirty clothes room on the "mine side" of the dry. After dressing in their underground clothes the men walk down a hallway past the shower and toilet rooms, stop at the lamp room for their lamps, and pass through the tunnel to the shaft. After the men have changed to their underground clothes they are not permitted to go into the "clean side." As they come off shift all men take a shower bath before entering the "clean side" of the dry.

A small branch tunnel connects the dry with the basement of the shop building. this tunnel permitting passage between the dry and shop building without going outside.

The interior of the street side of the dry is painted white with a grey dado that matches the grey lockers. The mine side is painted white with a dark reel dado. The floors are of concrete and are unpainted. Wet concrete floors always present a slipping hazard that has been diminished to a minimum in this dry by introduction of ground corundum into the concrete just as the final troweling was being done.

The heating and ventilating of this building is unique in dry construction. It was designed by E. Orrick of Walker Jamar Company, Duluth, Minnesota, in collaboration with the architect and the mining company engineering department. The street side is heated by unit heaters, with one unit connected to the outside by a duct that permits the introduction of fresh air whenever needed. The dampers in this duct controlling the air flow are manually controlled, and can be set to introduce only fresh air, to recirculate, or any combination of the two.
procedure. This is done for two reasons: first, to gain elevation and removed at the floor, reversing the usual

It should be noted that warm air is introduced at ceiling

ventilators.

out of dust accumulations. Air is exhausted from the

under the benches. The ducts are provided with drains,

in the floor are provided with registers and are located

through a filter bank and be recirculated. The openings

large locker room and the wash room on the street side.

from the large dirty clothes room, causing a complete

change of air every three minutes, and the remainder to

the small dirty clothes room and other outlets. The air is

introduced into the rooms from an interior duct at ceiling
elevation; this duct running the full length of the building
along the outside wall. It is 36” x 30” in cross-section at
the fan, diminishing to 24” x 12” at the far end of the
large dirty clothes room and to 20” x 12” at the end in the
small dirty clothes room. There are 28—12” x 8” grilles,
spaced uniformly in the duct, in the large room, three
grilles in the small dirty clothes room, and one each in
the laundry and lamp rooms. The duct is fitted with
splitter dampers at each grille to control the distribution
of air through the rooms. Air coming from the grilles
moves across the room above the clothes to the
opposite wall; from there it is deflected back and
downward through the clothes. The return air from the
two dirty clothes rooms is handled in concrete ducts
beneath the floor. Two ducts with fourteen 14” x 10”
openings run the length of the large dirty clothes room
and join a single duct from the small room which has
three openings in that room. From this junction a single
duct passes across the building to a point between the
large locker room and the wash room on the street side.
Here the air goes up to the ventilating unit to pass
through a filter bank and be recirculated. The openings
in the floor are provided with registers and are located
under the benches. The ducts are provided with drains,
and all pitch to the drains, permitting periodical washing
out of dust accumulations. Air is exhausted from the
shower, laundry, and lamp rooms through roof
ventilators.

Eureka Mine—Asteroid Location Left, Eureka Location Right

It should be noted that warm air is introduced at ceiling
elevation and removed at the floor, reversing the usual
procedure. This is done for two reasons: first, to gain

any possible settling advantage resulting from the high
specific gravity of hematite dust; and second (and most
important) to move the dusty air down and away from
the faces of the men dressing. The underground clothes
after being dried are very dusty; the most severe dust
condition occurs as the men are putting them on.
Consequently, an upward circulation of the air would
carry the dust past the faces of the men, whereas a
downward movement carries the dust down and allows
the men to breathe air comparatively free from dust.

The ventilating and heating unit consists of a fresh air
and recirculated air mixing chamber controlled by
interconnected dampers on both openings; a filter
section; a face and by-pass damper section controlling
the air passing through the heating coils; a heating coil
section; an inlet vane control section that controls the
amount of air supplied, and a fan section. All dampers in
this unit, as well as the dampers in two roof ventilators
from the large dirty clothes room, are electrically
operated and automatically controlled. It is possible,
however, to control the entire system manually and by
so doing to rapidly cool the rooms for the comfort of the
men changing clothes. The room temperature can be
cooled to outside temperatures, under the system as
installed. Outside summer temperatures in this region
are not high enough to warrant the introduction of
artificial cooling in this type of building. A temperature
of 85°F to 100°F is maintained for drying, dependent upon
the relative humidity of the outside air. During the winter
months the relative humidity of the outside air is so low
that the clothes can be dried with the minimum
temperature, but in the summer the higher dry bulb
temperatures are required.

The dry bulb temperature of the mine side is controlled
by a dry bulb thermostat located in the recirculated air
duct, this thermostat operating the modulating type face
and by-pass damper in the central unit. The operation of
this damper is such that as more heat is required the
face damper opens wider allowing more air to pass
through the heating coils; inversely, should less heat be
required, the face dampers will close .and the by-pass
dampers open. To prevent unduly chilled air from
entering the room a low temperature control thermostat
is located in the discharge duct and functions to prevent
the discharge air from the fan from going below a
predetermined setting.

The humidity of the room, so important in the drying of
the clothes, is controlled by a wet bulb thermostat
located in the recirculated air duct close to the dry bulb
thermostat. The wet bulb thermostat operates a
modulating type damper motor which controls the fresh
air damper and the recirculated air damper in the central
unit. Its action is such that with an increase of humidity
in the recirculated air the fresh air damper will open to
admit more fresh air and dilute the moist air to a
satisfactory humidity. In addition to the above, the wet
bulb thermostat controls two damper motors in the two
roof ventilators from the large dirty clothes room. Their
action is synchronized with the operation of the fresh air
damper so that as more fresh air is admitted the dampers in the roof ventilators are opened wider and wider, permitting the exhausting of the damp air.

A mixed air thermostat, incorporated with the fresh and recirculated air damper motor circuit, is located directly back of the filters. The function of this thermostat is to prevent too cold an air mixture being admitted to the heating coils. Also included in the fresh air damper motor control circuit is a manual and automatic selector switch and a manual ventilation switch to allow manual positioning of the fresh and recirculated air dampers, and the dampers in the roof ventilators, as stated above. These switches are located in the dryman’s room.

Through a relay, the fresh air damper motor is interlocked with the fan motor circuit in such a way that on power failure to the fan motor the fresh air damper and dampers in the roof ventilators will close automatically. In addition to this safety relay two heat exchanger thermostats are located in the condensate return of the heating coils and operate to shut down the fan and to close the fresh air damper on failure of the steam supply.

The fan for this central unit is driven by a 220-volt, 3-H.P. motor, the power being supplied through a three-phase circuit controlled by a magnetic starting switch. This switch is equipped with two push button stations, one located near the motor on the control box and the other located near the manual and automatic selector switch in the dryman’s room.

The introduction of the ventilating system in this dry marks the end of the discomforts of the old change houses. The constant, excessive heat to dry the clothes is no longer necessary. If the relative humidity of the outside air is so high that high temperatures are required for drying the clothes, the rooms can be quickly cooled for the comfort of the men as they change. The division of the dry into a clean clothes side and dirty clothes side does away with the mingling of men dressed for the street with those dressed in dirty underground clothes, and gives the men a clean place in which to keep their street clothes.
(2)—In territories that are dangerously near to dikes or horses of jasper.

(3)—In the first two or three sub-levels under an old territory which has stood long enough to permit the old covering to rot.

When confined to the above and similar uses, the consumption is considerably less as is evidenced by the reduction in average use of 0.5 lin. ft. per ton in 1931 to 0.02 ft. per ton in 1935. The period from 1935 to 1937 shows an increase from 0.02 to .04 lin. ft. per ton due to the opening up of new territories and the reopening of old previously worked territories. From a cost-standpoint, the use of the netting is not expensive considering the benefits derived. At one property in 1937, with a production of nearly 800,000 tons, 30,200 sq. ft. were used at an average cost of $.0003 per ton.

The accompanying photographs show three views of a standard slice with 9' legs and caps, double poles and wire. In this case, as an additional precaution, poles with the wire attached have been laid against the jasper hanging wall rock in the breast. When the slice on the next sub-level is advanced beyond the last set, this extra wire will serve as a basket which aids in holding back a possible rock run. This system proves very effective under certain conditions, but occasionally causes considerable trouble due to the fact that if a breakdown occurs just beyond the last set, the presence of the netting makes the driving of spiling poles much more difficult than if the wire had not extended out over the end of the slice.

Sketch No. 1 shows a sectional view of a typical topslicing operation which is progressing out under a relatively flat and uneven hanging wall. The cross-hatched area shows the two preceding sub-levels in which the usual pole covering was reinforced by wire netting in the area immediately under the rock and extending far enough back to form a safe overlap. In the first two or three slices at the top of the ore body, wire was used throughout in order to form a substantial, impervious mat as rapidly as possible. This mat must be covered by a minimum of 30' of broken rock blasted from the hanging. The rock acts as a protective cushion which is necessary to protect the working places in the event that a large mass falls from the hanging wall. As the slices progress out under the uncovered area, more rock must be blasted onto them in the same manner. When the hanging wall is relatively steep, a thick safe gob is built up more rapidly since there is a smaller proportion of each slice that extends out into uncovered territory.

A condition such as shown in the plan view, Sketch No. 2, where a small dike cuts transversely across the ore body, calls for some planning in advance. In order to complete the mining of the ore up to the mining limit with a minimum amount of dilution from the dike, it is necessary to plan the slices so as to cut it a minimum number of times. Further, in order that the dike shall not create a hazard when mining is continued on the sub-level below, it is necessary to blast it down into the slices immediately adjacent to it. Without the use of wire
netting, it is almost impossible to keep this lean material from running in when mining is continued below.

One method of handling the above situation is shown in the sketch. Slice No. 1 is turned toward the dike as sharply as possible and then driven adjacent to it up to the mining limit. The floor is then covered with double poles and wire. Slice No. 2 is treated in the same fashion. Before mining operations are continued in Slice No. 3, the dike is blasted into the first two and the timber sets in these are drilled and blasted. When handled in this fashion, it is then possible to slice up to the original location of the dike from Raise No. 2 without contamination from it.

ECONOMICS OF MINNESOTA IRON ORES*

BY O. A. SUNDNESS, CHISHOLM, MINN.**

The iron ore deposits in the northeastern part of the State of Minnesota have been renowned for many years, due to three important attributes—namely, the enormous size of the deposits, high average iron content, and easy accessibility. With a ready market to absorb production, these factors all contributed to placing an immense value on these ore fields.

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RESERVE TONNAGES

There are three separate and distinct deposits.

The Vermilion Range is the most northerly and contains a high grade of ore worked mostly by underground methods to depths of approximately 2,000 feet or more. The estimated reserve, as reported by the Minnesota Tax Commission of May 1, 1939, including ores in stockpile, is approximately 14,000,000 tons.

The Mesaba Range lies to the south of the Vermilion Range a distance of about 20 miles as the crow flies. This is the largest of the deposits, containing, according to Tax Commission records, approximately 1,156,000,000 tons of which 18,000,000 tons are reserve ores and lean ores in stockpile. It is worked largely by open pit methods, since very little of the ore reaches a depth of 600 feet.

The Cuyuna Range lies about 50 miles to the southwest of the Mesaba Range. Its ores are of a lower iron content than the other two ranges, but they carry some manganese, which makes them desirable. This deposit is worked by both open pit and underground methods, and has a reserve of approximately 60,000,000 tons. The total tonnage of iron ore remaining in northern Minnesota is therefore about 1,230,000,000 tons, and Tax Commission records indicate that 1,148,000,000 tons have been shipped.

ANALYSES AND SHIPMENTS

The Vermilion Range, which started its shipments in 1884, has the highest average iron content. Its early shipments exceeded 61 per cent natural iron, while current analyses average between 57 and 58 per cent
natural iron. The total shipment from this deposit through the 1938 season has been 65,378,566 tons.

The Mesaba Range began to ship in 1892. Prior to 1904, the analyses averaged higher than 55 per cent natural iron. Beginning with 1904, a steady lowering of the iron content of shipments took place until an average of 51.50 per cent was reached in 1909, since when the analyses have been held very close to this figure. The ability to hold this average analysis is probably accounted for by the increased beneficiation of low-grade, non-merchantable areas in the deposit. In 1906, only .6 per cent of the total ore shipped was beneficiated, while in 1910 the washed concentrates began to have an appreciable effect. Today more than 48 per cent of Minnesota's shipments are beneficiated by washing, crushing, screening, drying or sintering. The total shipment from this range through the 1938 season has been 1,044,130,938 tons.

The Cuyuna Range is the youngest of the three ranges, coming into production in 1911. The average analysis, combined iron and manganese, in the early operation was approximately 50.50 per cent natural, while the average in the last few years has dropped to 47 to 48 per cent natural. Total shipments from the Cuyuna Range through the 1938 season have been 38,792,099 tons.

**HIGH POTENTIAL VALUES**

It is apparent that deposits of such large size and high grade, with their easy accessibility, indicated enormous values in the early days. These iron ore values induced large capital expenditures by operators for shafts, mine structures, rock and earth stripping, etc. However such values do not accrue only to the operator as is commonly supposed. When set in motion by capital, they become distributive values. Only that portion of the value remaining after development expenses, operating costs, transportation charges, royalties and taxes have been paid accrues to the operator. Auxiliary industries such as rail and water transportation, power facilities and equipment supply companies draw on these iron ore values, and in turn create new and additional values, the contribution from which, together with the contribution from the main iron ore industry afford enormous income to local communities, counties, the state, and even to the nation as a whole.

Unfortunately, this conception of the far flung distribution of iron ore values is often obscured by politics, and is usually overlooked or not comprehended by the general public, who are the principal beneficiaries of it all. In northern Minnesota many fine communities, modern in every way, with excellent schools and municipal improvements, have been erected and maintained from these iron ore values. Through 1938 a total of $277,729,934 had been diverted for the purpose of local governmental expense.

In addition to maintaining the local communities, these ore values have contributed generously to the county and state governments. Through 1938 the counties had received $75,988,681, while the State of Minnesota, through its ad valorem tax, occupation tax and royalty tax, had benefited to the extent of $204,653,978. Also supplementing the tax income to the state, are the royalty payments on state owned mines. This distribution from the ore values totaled $39,924,477 through 1938. The state and its subdivisions have therefore drawn on these ore values to the extent of $498,000,000.

In addition to the foregoing, the Federal government has benefited handsomely in its collection of income tax, corporation tax, war profits tax, surplus tax, social security tax, and others, the total of which is not easily obtainable.

![Chart 1](image1)

A-Year Running Average Production Pig Iron and Steel in United States
B-Year Running Average Metallic Content of Minnesota Iron Ore Shipments
C-Percentage Relationship of Metallic Content of Minnesota Shipments to Total Steel Production

Hull-Rust Open Pit Iron Mine, Oliver Iron Mining Co., Hibbing, Minn.
VALUES DEPENDENT UPON MARKETS

These payments from the iron ore values, in addition to local payrolls, which totaled $33,000,000 in the one year, 1920, together with the income from, and the payrolls in the various auxiliary industries, have been possible because this iron ore had a value. In order to have a value, it is necessary to have a market for the ore in which it can be sold at a price which will pay for its extraction, pay for its transportation, pay its taxes, and still leave a profit for the investment of capital by the operator after he has satisfied the demands of the Federal government. It is this broad and comprehensive, but fundamentally sound definition of value that is so often lost sight of by legislatures when formulating harsh taxation policies, and by labor organizations when enforcing unreasonable labor demands.

In the final analysis, so-called “Ghost Towns” do not always come into being because of the exhaustion of the mineral itself. Competition with other minerals or materials, competition with the same mineral from other fields, harsh taxation policies, and other economic reasons are just as vital and just as effective in destroying the market for a mineral, and consequently its value, as though the deposit were exhausted. In other words, any mineral which can no longer be brought into a market and sold at a profit has no further value, can pay no more taxes, can support no more payrolls, and can sustain no more auxiliary industries.

The iron ranges of Minnesota are not without their “Ghost Towns” even though they are not spoken of as such. Competitive conditions with aluminum, with scrap iron and scrap steel, with iron ores from other fields, causing a slow but steady decline in Minnesota's position as a producer of raw material for the steel industry, have resulted in the inability to operate most of the underground mines on the Mesaba and Cuyuna Ranges. Because of low grade ores in some cases, because of high costs, high taxes, as well as a multiplicity of taxes, and high royalties, these ores can not be brought to a market and show a profit. Hence the ore is valueless, the mine is inoperative, the payrolls are destroyed, and the town without its industry is in its ghost stage.

MINNESOTA ORES LOSING THEIR MARKETS

That Minnesota ores have been losing their markets, that the State of Minnesota has steadily been losing its position in the pig iron and steel picture of the United States, is a fact that has been but little considered, and even now is grudgingly admitted by only a few people. It is difficult to conceive, and unpleasant to admit that fields such as these can be vulnerable, that ore bodies such as these could be declining in value because they are losing their markets; but a study of the records confirms this fact.

Chart 1 is presented showing three curves, all of which are constructed on a five-year running average in order to level off the widely fluctuating annual variations. Curve “A” indicates the total production of pig iron and steel in the United States in millions of tons, and is computed from the American Iron and Steel Institute reports. Curve “B” indicates the metallic content of iron ores shipped from Minnesota in millions of tons, and this is computed from the Minnesota Tax Commission reports of annual shipments, and Crowell & Murray's reports on average analyses of Mesaba Range ores. Curve “C” is a trend line developed by using points on curve “B” as a numerator, with points on curve “A” as the denominator of a fraction to arrive at a percentage figure expressing the metallic content of Minnesota shipments in terms of the total United States steel production.

Scranton Mine Pit, Pickands, Mather & Co.

A glance at the two curves, “A” and “B,” indicates at once that they are slowly but steadily diverging, until in the last few years the spread has become very marked. Curve “B,” metallic content of Minnesota ores, is not keeping pace with curve “A,” production of steel in the United States.
United States, as evidenced by the increasing length of the heavy vertical lines. In the first five-year period, 1906-1910 plotted on 1908, curve “A” shows the average steel production to have been 28.6 million tons, while curve “B” indicates that the average metallic content of Minnesota ores shipped during this period was 13.9 million tons. Studying the effects of the war-boom period, which centers on the year 1918, we find steel production at 49.1 million tons, an increase of 72 per cent over the 1908 period, while the metallic content of Minnesota ore shipments was 21.6 million tons, or an increase of only 55.4 per cent over the 1908 average. Similarly, looking at the Coolidge-boom period centering on the year 1927, we see steel production averaging 57.1 million tons, up 100 per cent, while the metallic content of Minnesota shipments was 21.0 million tons, up only 51 per cent over the 1908 period, and down 2.8 per cent from the 1918 period. The comparison of these two boom periods indicates that Minnesota ores are gradually being eased out of the picture, and this is further borne out by looking at the latest five-year average. This period includes the year 1937, Minnesota’s record high year in ore shipments, and is plotted on the year 1936. Here we see steel production averaging 41.4 million tons, up nearly 45 per cent over the 1908 period, but the metallic content contributed by Minnesota to the steel picture of the United States is only 13.8 million tons, or .7 per cent less than the amount taken from Minnesota in the 1908 period.

The argument is frequently advanced that since Minnesota is shipping as much ore as she did in the earlier periods, the industry is holding its own. This is a fallacy, because costs, transportation and taxes have increased so greatly that for some mines all of the profit is wiped out. By reference to curve “C” in Chart 1, the actual position on a metallic tonnage basis of Minnesota ores m the finished steel production of the United States may be traced. As previously explained, this is a percentage curve expressing the relationship between Minnesota ores and the total United States steel production. The points are computed for each year from the points immediately above curves “A” and “B.” In the pre-war period, 1906-1914, it will be noted that the metallic content of Minnesota ores shipped was approximately 47 per cent of the total tonnage of steel produced in the United States. A steady annual decline in Minnesota’s position in the steel picture has taken place until today she is having difficulty in maintaining a 33 per cent position, and during the depression dropped as low as 26.7 per cent. The vulnerability of these immense, high grade deposits thus becomes apparent. These fields, accustomed to supplying nearly 50 per cent of the raw material for the production of steel in the United States, must now be content with the bare one-third or even less. Minnesota has lost, and is still losing, much of her prestige in the steel world; other minerals and materials, as well as iron ores from other fields, are making their influence felt. Competition is destroying the market, and hence the value, of these ores.

INFLUENCE OF SCRAP IRON AND SCRAP STEEL

The most serious competitor of Minnesota iron ore has been scrap iron and scrap steel. From the standpoint of conservation of natural resources, the use of these materials is of a great economic benefit to the United States. However, the lack of a proper knowledge, and the lack of a sound understanding of the extent to which this scrap affects the potential values of Minnesota ores is general, and therefore detrimental to the best interests of the iron mining communities, and to the state itself.

Chart 2 indicates the relationship of scrap to the production of steel in the United States. Curve “A” is identical with curve “A” in Chart 1—a five-year running average of the production of steel and pig iron in the United States in millions of tons. Curve “D” is a five-year running average of computed scrap consumption. Authentic figures on the consumption of scrap throughout the period shown on the chart are not available. However, by using the total tonnage of ore consumed in the United States, and assuming this to have a natural iron content of 50 per cent, the total metal coming from the ores may be computed. By subtracting this metal from the total tonnage of steel and pig (other than basic and bessemer) produced annually in the United States, a balance is arrived at which approximates the annual scrap consumption. This method of calculation aims to exclude the recycled plant production scrap and to include “old” plant scrap (discarded metal products). For purposes of this study, impurities in finished pig iron and billets are assumed to offset losses of metal in slag, oxidation, tapping, etc. Curve “E” is a trend line or a percentage curve indicating the relationship that has existed between the consumption of scrap and the total production of steel in the United States. The points are computed from the corresponding points on curves “A” and “D.”

Referring to curve “D” we see that scrap consumption during the five-year period 1906-1910 was 5.7 million tons. Comparing this with the two boom periods previously mentioned, we find an average of 15.3 million tons of scrap consumed in the 1918 period, an increase of more than 313 per cent, and an average of 23.6 million tons consumed during the 1927 period, an increase of nearly 537 per cent over the 1938 period. During the last five-year period, we find scrap consumption averaging 20.8 million tons, which is still an increase of approximately 462 per cent over the 1908 period. In these same periods the metallic content of Minnesota ore shipments increased 55.4, 51.0 per cent, and lost .7 per cent respectively.

The growing importance of scrap in the steel industry of the United States becomes apparent by looking at curve “E.” Subsequent to the first five-year period shown in 1908, it is evident that there has been a steady and rapid increase in the percentage of scrap in the finished product until today more than 50 per cent of our steel comes from reworked old metal. About 1909, tonnage of open hearth steel produced overtook and passed the production from converters. Today approximately 95 per
percent of the finished steel comes from the open hearth process, which process is able to consume large percentages of scrap and may even he run on 100 per cent scrap. It is this basic change in the metallurgy of steel that accounts for the reduction in iron ore consumption and its replacement by scrap.

The natural question that arises is, “Will not the scrap supply soon be exhausted?” Indications are that this time is not presently in sight. Advancements in metallurgy, particularly in the alloy fields, have brought about changes and improvements in the strength of the product, until now a steel member of one-half the cross-section that was formerly used meets the required design. In addition to this, the properties of the alloys are such that they resist corrosion and wear to such an extent that their life is appreciably extended. Both of these qualities tend to materially extend the scrap supply and to push the iron ores still farther out of the picture.

Prominent men in the scrap industry predict that steel will some day be an ever revolving commodity, and then we will never have to think about iron ore reserves again. This prophecy may or may not come to pass, but the fact remains that in the last 30 years the use of scrap has increased from an average of 3.7 million tons to 20.8 million tons per year. Since one ton of scrap is equivalent to two tons of ore, this increase of 17 million tons displaces 34 million tons of iron ore, and if Minnesota contributes 60 per cent of the total supply of ore used in the United States, this means a curtailment of her production to the extent of 20 million tons annually, or enough to establish substantial prosperity on her iron ranges again.

**IRON ORE PRICE CRUX OF PROBLEM**

The price of iron ore, which is of course governed by the total of all of the costs such as taxes, royalty, production and marketing, holds the solution to the problem. If the price of iron ore is increased 50 cents per ton, it reflects itself in an increase of $1.00 per ton in the cost of the pig iron or steel made from this ore. This increased cost in turn reflects itself in the scrap iron market, since pig iron and scrap are interchangeable in the open hearth furnace, and opens up an enormously increased area from which to draw scrap. This is true because it makes available an additional dollar with which to pay freight charges. If, for instance, the present total freight charges are $3.00 per ton, and $1.00 additional is made available for this purpose, and since areas or circles are to each other as the square of their radii, we have an additional area from which to draw scrap that is nearly twice the size of the original area, or in the ratio of $9(3^2)$ to $16(4^2)$. It is evident, therefore, that any increase in the price of Minnesota ore is detrimental to production, in so far as it only invites scrap competition from a new and extended area. Conversely, a reduction in price and cost should be a beneficial stimulant to ore production and to additional mining employment.

Chart 3 shows curve “E,” percentage of scrap in finished product, superimposed upon curve “C,” percentage of Minnesota metal in finished product. This indicates clearly the steady inroads that scrap has been making, and the consistency with which it has been pushing Minnesota ores out of their previous markets. The last two points on these curves are affected by an abnormal condition existing in 1937, when more than four million tons of scrap were shipped out of this country. This was 100 per cent more than had been exported in any previous year, and it was occasioned by large demands principally from Japan, the United Kingdom, and Poland. The lifting of such a large amount of scrap, the equivalent of eight million tons of iron ore, out of our domestic markets, is indicated by the drop in the scrap curve “E” and the complementary rise in the Minnesota metal curve “C.”

As previously noted, curve “C” records the fact that Minnesota has declined from the position of supplying one-half of the raw material required for steel production in the United States, to a position of supplying a bare one-third. This is a loss of one-sixth of the raw materials market, or a loss of one-third of Minnesota’s formerly contemplated market. Since the values of mineral deposits are based upon the expected life of the mining enterprise, or upon the time in which invested capital may be expected to be returned, this loss of market with its corresponding extension of life becomes of vital concern to operators and to the people of the state as a whole. The loss of one-third of its market means an extension of life amounting to 50 per cent of its formerly contemplated life.

The method of translating such changes into concrete figures of value for purchase purposes, for taxation, or for other financial transactions is by means of the Hoskold tables, universally used by engineers the world over. As an illustration and without attempting to predict the life of Minnesota ore reserves, by referring to the Hoskold tables and assuming a contemplated 30-year life later extended 50 per cent to a 45-year life, we find in the 8-4 per cent discount tables the factors .340727 and
.251774 as representing the values for these respective periods of life. The decrease in value by extending the life 50 per cent is therefore the difference between these two factors, or .088953. This reduction is 26 per cent of the original factor, hence the decrease in value in this assumed case is 26 per cent of the original valuation. In a general way this measures the penalty imposed upon Minnesota ores for having costs so high that competing materials and minerals are enabled to press in and absorb their markets. It has caused serious unemployment, curtailment in the auxiliary industries, shrinkage of payrolls, and loss of tax income to governmental units.

**SITUATION ANALOGOUS TO COAL**

The eastern coal fields were at one time also supreme, and were considered to be invulnerable. However, high costs and competition with other fuels have now placed many large coal operators in a position where they can no longer meet their tax payments. In many communities where the coal company was relied upon to furnish payrolls and tax money, dire need exists today, teachers have not been paid, schools are either closed or the school term has been shortened, municipal and county functions have been curtailed, roads are deteriorating and decadence has set in. For the people of the State of Minnesota, where many of the communities derive over 95 per cent of their tax income from these potential but vanishing values of iron ore, where the state has had its financial burden eased to the extent of $498,000,000 by income from these same ores, the economic study of their iron ore heritage should be a most important subject. Constructive thought and action in the mining communities themselves, and intelligent, fore-sighted policies developed by the state, may still avert a situation similar to that now existing in the eastern coal fields and may help to restore, at least partially, the markets for Minnesota iron ore.

**SINKING A MINE SHAFT HALF A MILE DEEP**

**BY J. C. SULLIVAN, IRONWOOD, MICH.**

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Shaft sinking has become a highly specialized phase of mining operations. Through a combination of careful planning, thorough organization, and the use of efficient modern equipment, continually better records are being made both as regards faster work and lower costs. American mines always have been among the leaders in shaft-sinking efficiency, and their work in late years has sustained their established position. A recent contribution to this creditable record was the sinking of a new shaft to serve the Newport and Bonnie iron mines of the Youngstown Mines Corporation at Ironwood, Mich. These properties, which are on the Gogebic Range, are operated for the owners by Pickands, Mather & Co.

This new shaft, which is the third to be sunk on the property, was required because of deformation of the shaft previously used by subsidence and movement of the ground which inclosed it. The iron-bearing formation, in common with the series of sedimentary rocks in which it occurs, dips at an angle of between 65 and 70 degrees from the horizontal. The original shaft was an inclined one in the footwall and became unusable as the result of a squeezing action set up by shifting and settlement of the overlying ground. In an effort to eliminate the difficulty, a new vertical shaft, known as the Woodbury Shaft, was sunk in 1915-16. It was started at a point about 100 feet from the foot of the surface contact between the iron-bearing horizon and the underlying sedimentary strata. With continued use, however, this shaft was also affected by subsidence and movement of the surrounding rock, and in 1929 preparations were made for the sinking of another shaft. To obviate all possibility of its being subjected to deformation from subsurface movements, it was located in the granite several hundred feet away from the contact with the sedimentary rocks and carried in this granite to its full depth. After careful consideration, a vertical shaft was selected. Although this choice meant more development work in driving underground openings into the ore, its advantages were found to more than offset the hazards involved in running wheeled skips on inclined rails and the greater cost of maintaining a sloping shaft and of hoisting ore through it.

*Mining Engineer, Pickands, Mather & Co., Newport Mine.*
would be favorable for erecting necessary surface structures, for storing and handling required materials, and for disposing of excavated rock.

It was determined beforehand that all holes would be drilled vertically and arranged around a central hole for them to break to, and it was decided to experiment with different methods of drilling, blasting, and mucking in the first 100 feet of depth. To facilitate this plan, a central 6-inch hole 100 feet deep was drilled at the start with a churn drill. This hole was then filled with sand. Sinking through the overburden began on August 28, 1930, using temporary power, water, and compressed-air facilities. Surface material was removed with a scraper and a 25-horsepower electric hoist to a depth of 22 feet, where granite was encountered. The exposed rock surface was then thoroughly cleaned to insure an impervious joint, after which forms were erected for the concrete shaft collar and permanent headframe piers all of which were poured with 428 cubic yards of a 1:3:5 mixture. The collar steel set, supported by a bearing set, was installed with its top flush with the collar and was made rigid to prevent any movement as a result of blasting. A 35-foot temporary timber headframe was erected to serve for the first 100 feet of experimental excavating, and a 50-horsepower single-drum electric hoist was set up about 130 feet from the shaft.

Drilling was started on October 31, 1930, with two shifts of six miners and one foreman each. The primary purpose of this initial work was to reach a suitable distance below the collar before the erection of the permanent steel headframe. To avoid damage to the concrete collar, only short holes were drilled and blasted a few at a time. The drilling round consisted of 91 holes; and, with a view to determining the most efficient drills for the work to follow, various types of machines were tried out. Rock was hoisted in a 1-cubic-yard bucket, dumped over an apron into small cars at the surface, and used to fill in and to level up the area adjacent to the shaft which was to serve for the storage of materials. Steel sets were placed in the shaft as sinking progressed, and had been installed to within 21 feet of the bottom of the 65-foot shaft when all work was suspended on November 24, 1930, to permit the steel erectors to construct the permanent headframe.

The headframe was erected under contract by the Lakeside Bridge & Steel Co., of Milwaukee, Wis., which firm carried out this work between November 25, 1930, and February 21, 1931. The structure is of steel, housed with corrugated iron. It is of the 4-post type with the top sheave wheels 132 feet above the collar. It contains space for the installation of a crusher at such a time in the future as furnace requirements may call for it. In October, 1931, an additional inclosure was built in front of the headframe to keep the snow away from the shaft and also to facilitate the lowering of steel, lath, and other materials in the shaft. It having been decided to use the existing Bonnie change house, 1,000 feet away, as a dressing room for the workmen, it was desirable to provide protection for them when coming off shift in wet clothing. Accordingly, a combination concrete tunnel and corrugated-iron walkway was constructed. The floor of the tunnel section left the shaft 8 feet below the collar; and the men were able to cover the entire distance between the shaft and the change house under shelter.

Meanwhile, the Worden-Allen Company was constructing a steel-frame, brick power house and a heating plant to house equipment and machinery for use in the shaft-sinking operations and for the mining operations which were to follow. Foundations were poured for two compressors, one cage hoist, and one skip hoist, involving 1,077 cubic yards of concrete. A concrete basin for cooling water and two hold-down sheave foundations were also poured.

An Ingersoll-Rand Class PRE-2 compressor, having a piston displacement of 3,600 cfm, and driven by a direct-connected General Electric 700-horsepower synchronous motor, was installed to supply air at 100 pounds pressure to the shaft and shops. An air filter was placed on the intake line and, to insure dry air for the tools, an I-R Class VM-4 aftercooler was interposed in the discharge line between the compressor and the receiver. A 6-inch air line was run to the shaft and shops. A second compressor of the same type and size will be provided to furnish additional air for the mining operations.

For raising and lowering the skips during shaft sinking there was installed an Allis-Chalmers double-drum hoist having drums 10 feet in diameter and 6 feet across the face. It was driven by a G-E 600-horsepower induction motor through herringbone gears and had a maximum rope speed of 1,000 feet per minute. One drum was keyed, and a clutch on the second drum permitted
operating one skip at a time. This hoist, with a drum capacity of 4,500 feet of 1½-inch rope, will be converted into a cage hoist when mining operations are started through the new shaft.

A temporary cage hoist was provided to serve during the shaft-sinking period. This was a Nordberg double-drum unit having drums 7 feet in diameter and 5 feet across the face. Like the skip hoist, one drum was keyed and the other connected through a clutch, thereby making it possible to operate either drum singly or both in unison. This hoist was driven by a G-E 400-horsepower induction motor. It has a capacity of 2,700 feet of 1¼-inch rope and a maximum rope speed of 800 feet per minute. When mining operations begin through the shaft, this hoist will be put in service at the Woodbury Shaft and will be replaced with a large direct-connected skip hoist.

Power purchased from a public-service company was transmitted to the property at 33,000 volts and there transformed to 2,300 volts for driving the motors. A Westinghouse switchboard, with inclosed connections and switches and with visible meters and signal lights, was installed.

A temporary tool-sharpening shop was constructed adjacent to the head frame inclosure and equipped with two Gardner-Denver D.S.8 sharpeners and HP-20 punches for handling “Jackhammer” drill steel and with one Sullivan Class A sharpener for reconditioning the steel used for drilling the large central hole previously mentioned. Three oil furnaces, tempering tanks, racks for drill steel, and a work bench were provided; and an Ingersoll-Rand Type 4K shank grinder helped to reduce drilling costs by-squaring up the shank ends of drill steels and the striking faces of drill pistons. A small adjoining structure was built to serve as a shop for making minor drill repairs. More extensive repairs were handled at the main shops.

THE MEN RESPONSIBLE

The shaft-sinking crew comprised 74 men working under the direction of a captain and four shift bosses. The miners did the drilling and the blasting and installed the steel.

A 10x20-foot wooden structure, located 200 feet from the shaft and heated with steam, was used for making up igniters into primers. It was kept locked when not occupied by a powder-man. Powder was stored in a steel magazine about 100 feet away and 250 feet from the shaft. The former was insulated with 12 inches of sawdust retained by an exterior housing of wood, and this served to prevent freezing of the powder during cold weather. The magazine was fitted with a double lock to keep out all except those authorized to enter it. A small temporary structure was used for the storage of tools and equipment when they were not in service. It was heated with steam to prevent freezing of hoses and other items of equipment which might come from the job with water in them. A steam-heated garage was built to house two G.M.C. 3- to 5-ton trucks which were employed for hauling rock excavated from the shaft. Cinder-surfaced roads were constructed to connect these several structures and the shaft.

Two 4-ton sinking skips of the chair type were built. Some details concerning them are given in connection with an accompanying illustration. Safety devices were provided to enable the skip to grip the guides in case the rope tension should be removed or diminished through any cause. Two sinking cages were also designed and constructed.

All these various and essential structures and items of equipment having been made ready, sinking of the shaft was resumed at the 65-foot elevation on April 30, 1931. An 8-inch flanged pipe line delivered compressed air in the shaft. This line extended to within 100 feet of the bottom, and a 4x4x8-inch tee was attached to its lower end. To each side of this tee there was connected a nipple and elbow, the two elbows having a span of about 4 feet. A length of 3-inch hose was then connected to each elbow and permanently connected to another 4-inch elbow which, in turn, was connected through a nipple to still another 4-inch elbow from which there extended downward a 4-inch pipe, 7 feet 3 inches long, which terminated in a header or manifold. The elbows and their connections were hung from the lowermost steel set by a 1-ton chain block; and, as the hose section was 25 feet long, it was possible merely by lowering the assembly as the shaft progressed in depth to serve several successive cuts before an extension of the main 8-inch pipe line was required. Details of the air-line are shown in an accompanying drawing.

Each of the two headers or manifolds at the terminals of the delivery lines consisted of a 3x9-inch casting bored out in the center. Nine evenly spaced holes were then drilled around the casting and tapped for 1-inch connections. Seven 1-inch hose lines extended to rock drills, and each had interposed in it an I-R Type E line oiler. The two other lines served blowpipes, and each was fitted with a valve conveniently placed for operation by the miners when blowing holes. As two of these headers were used, it was possible to operate fourteen drills and four blowpipes simultaneously. When drilling of a cut was completed, the headers were disconnected, transported up the shaft by means of wire-rope slings underneath the cage, and hung at the shaft collar until they were again needed. They were then taken below
by two men, detailed to that task, in order that drilling might be resumed as soon as mucking was finished.

THE HEAD FRAME

This is a permanent all-steel structure for use in connection with the mining operations through the shaft. The top sheave wheels are 132 feet above the collar. This view, made during its construction, also shows the partly completed temporary steel-sharpening shop.

For all except the center hole, Ingersoll-Rand X-59 "Jackhammers" were used. Each of these machines weighed 72 pounds, and to increase the drilling speed each was equipped with two removable weights aggregating 120 pounds. One-inch hollow, hexagon steel was utilized and made up in lengths up to 13 feet with 12-inch runs. Starting bits of both 4- and 6-point types were used. After holes were started, 4-point bits were employed. Starting bits were 2½ inches in diameter and decreased ¼-inch in size with each change of steel, resulting in a 1½-inch bit with a 13-foot steel. Smaller-diameter holes were impractical for charging with powder. Much time was saved in placing holes in exact position by the use of spacers. Special tools were devised for recovering broken bits and for removing stuck steels.

A Denver No. 17 drifter drill, equipped with handles, was used for the large center hole and was weighted with 73 pounds to increase its production. Hollow, round steel of 1½-inch section having 6-point starters 5½ inches in diameter and 4-point starters 5¾-inches in diameter were used with this machine. Allowing for a decrease of ½-inch with each change of steel, holes 13½ feet deep were bottomed with a diameter of 3½ inches. This center hole was started with the aid of a steel-plate guide having a central hole for the drill steel.

Several drilling rounds were tried out, variations being adopted with changes in the character of the rock. The best results were obtained with one providing for 68 holes. A plan of it is reproduced. The locations of the center hole and of the corners of the shaft area were determined by dropping plumb lines from the steel sets overhead. The miners then posted themselves for drilling so as to interfere with one another as little as possible. Three men were detailed to the center hole, which was designed merely to serve as an opening for the other holes to break to in blasting and which was carried a few inches deeper than the others. About two hours was required to drill this hole. A 12-inch length of 3-inch pipe was inserted in each of the smaller holes after it had been started so as to keep out rock fragments and sludge as drilling continued through it. Completed holes were closed with cedar plugs, pending their loading with powder.

To reduce congestion at the bottom of the shaft, sharp steels were supplied and dull steels removed at frequent intervals. These were transported up and down the shaft in a cage. In order not to interfere with the drilling, the cage was stopped 8 feet above the bottom and the steels were handed into and out of it at that point. Each drilling round required the use of about 800 pieces of steel and the breakage per round ranged from 50 to 100 pieces.

The 68-hole drilling round which was used most of the time called for the drilling of approximately 850 feet of hole. Holes averaged 12½ feet deep, and netted an average advance in depth of 10.83 feet for the 247 cuts which were made. The average drilling time was 9
hours and 32 minutes per round. Hitches for bearer sprags for supporting the steel sets were drilled during regular drilling shifts.

Water presented no problem, as the normal flow of about 4 gpm. was little more than that required for drilling. Excess water was pumped by a small C. P. Quimby centrifugal pump into a skip suspended about 15 feet from the bottom. When a skip had been filled it was hoisted to the surface and the water dumped into a launder, which carried it away from the shaft. A flow of water which entered the shaft through a fault plane at a depth of 215 feet below the collar was handled by cutting a small sump and installing a 50-gpm. Aldrich pump which was operated by a float switch.

Eighty per cent gelatin dynamite in 1½x8-inch cartridges was used for blasting. No. 8 electric blasting caps and delay igniters were employed in accordance with the order of firing shown on the drilling-round plan. Primers were made up in the powder house; tagged with their respective numbers; and given an application of du Pont waterproofing material. Each was then inserted into a stick of powder whose outside paper covering was afterward tied tightly. All igniters were tested with a galvanometer to insure against misfires from faulty material. After it was made up, each primer was inserted into a strong cardboard guard, the inside of which was coated with glue. These guards were used as a precaution against premature blasts from sharp contacts. As a final operation, the igniter conducting wires were sandpapered to assure good connections.

HOW MUCK WAS HANDLED
Broken material was shoveled into either of two trays which, when filled, were raised and dumped into a skip. A hook on the tray slipped over a bar on the skip, and further elevation inclined the tray for dumping. Trays were raised and lowered by two Ingersoll-Rand Type HU 10-horsepower, single-drum air hoists. These were mounted on 4-inch H-frames and secured to the steel in the pipe and ladder compartments overhead. This sketch also shows the manner in which bridles operated to allow cages and skips to be taken below the lowermost steel set, and illustrates the hanging of a stage or dummy set below the steel already in place to protect it against damage from blasts and to serve as a working platform when installing additional sets.

PRINCIPAL DRILLING ROUND
Drill holes were arranged around a large central hole which provided an opening for them to break to. Of the several drilling rounds tried out, this one, providing for 6 8 holes which were fired with thirteen delays, proved most effective. Three men drilled the central hole with one machine, then put in Nos. 0, 1 and 2 holes. Meanwhile, other drillers were at work on the holes shown. All holes were vertical except Nos. 0 and 1, which were inclined very slightly towards the center. The average round required 850 linear feet of drill holes, and was completed in 9½ hours.
there twisted together until the final connection was made.

Following a round of drilling, all holes were blown out; all mechanical equipment was removed to the surface; the lights in the shaft were turned out from the surface; and the work of loading the holes was begun with the crew divided into groups under the supervision of the foreman. Four sticks of powder were placed in the bottom of each hole, except corner holes, and were followed by the primer. The remainder of the powder allotted to each hole was then inserted and the necessary tamping put on top of it. Tamping was done with a ⅛-inch round tamping stick. Powder charges varied for different holes. The zero-delay holes were loaded with eight sticks of powder followed by six sticks of tamping. In holes fired with delays, Nos. 1 to 11 inclusive, fourteen sticks of powder and three sticks of tamping were used. Corner holes were fired with two delays. Two sticks of powder were placed in the bottom of each and followed by a No. 13 delay. Eleven sticks of powder were then inserted, tamping was added to within 4 feet of the collar, and this was followed by a No. 12 delay, four sticks of powder, and tamping extending to the collar. This plan brought out the corners in a very satisfactory manner.

After all the holes had been loaded, the ends of the connecting wires were collected in groups. A double No. 20 lead wire was brought down from the steel sets overhead, laid in a circle around the shaft, and connected directly to the No. 0 delay holes, which were designed for instantaneous firing. One side of the connecting wires of each group of holes was then joined to the lead wire and taped. When all connections had been made on the first lead wire, the second one was connected and taped. The connections made in a manner similar to the first. All connecting wires of each group of holes was then joined to the lead wire and taped. When all connections had been made on the first lead wire, the second one was brought down from the steel sets and the other connections made in a manner similar to the first. All tamping sticks, boxes, etc., were then loaded on a cage and hoisted with the crew to the bottom set of steel. Tamping was done with a ⅛-inch round tamping stick. Powder charges varied for different holes. The zero-delay holes were loaded with eight sticks of powder followed by six sticks of tamping. In holes fired with delays, Nos. 1 to 11 inclusive, fourteen sticks of powder and three sticks of tamping were used. Corner holes were fired with two delays. Two sticks of powder were placed in the bottom of each and followed by a No. 13 delay. Eleven sticks of powder were then inserted, tamping was added to within 4 feet of the collar, and this was followed by a No. 12 delay, four sticks of powder, and tamping extending to the collar. This plan brought out the corners in a very satisfactory manner.

Preparing were then made to begin mucking. The broken rock was shoveled into 35-cubic-foot steel mucking trays which were built like boxes but with the dumping end and the top open. These trays, of which two were used, were raised and lowered by means of a bail placed so that the center of gravity was towards the closed end of the tray to prevent it from tipping forward while being raised. The trays were being filled, a skip was stationed at the safety mark in the lowermost steel set above. Skips and cages were constructed with briddles which enabled them to be lowered below the installed steel sets. These briddles consisted essentially of a vertical steel arm, on each side of the skip or cage which extended upward and engaged guides on the roadways above. The skip briddles were 50 feet long and the cage briddles 52 feet long. As the lowermost steel set was kept within from 35 to 50 feet of the bottom, it was possible at all times to land a skip or cage on the bottom. When one of the mucking trays had been filled, the skip was rung down to the bottom and the tray raised and dumped into it, after which the skip was hoisted to a position about 15 feet above the bottom until the second tray was ready for dumping. The skip had a capacity of about 75 cubic feet, or two trayloads, and upon being filled it was sent to the surface. The mucking trays were handled by means of two Ingersoll-Rand Type HU single-drum air hoists of 10-horsepower capacity. Their method of mounting and control is explained in connection with an accompanying illustration.

By means of a 15-horsepower Coppus Vano blower, ventilating air was forced down the shaft through a 12-inch steel pipe to within 100 feet of the bottom and below that point through a vent tube which was pulled up just prior to blasting by means of a ½-inch hemp rope operated by a hand winch at the collar. Immediately after a blast the vent tube was lowered and the blower started. The outfit was capable of delivering 3,600 cfm. at the surface and 1,500 cfm. at a depth of 3,000 feet. Thirty minutes of blowing sufficed to purify the atmosphere sufficiently to allow the men to return to work. They descended slowly in both a cage and a skip, meanwhile inspecting the shaft and dislodging any loose rock which had been thrown up and was held in precarious positions on the steel sets. Below the level where steel had been installed the side walls were carefully examined and loose rock barred down. Preparations were then made to begin mucking.

The broken rock was shoveled into 35-cubic-foot steel mucking trays which were built like boxes but with the dumping end and the top open. These trays, of which two were used, were raised and lowered by means of a bail placed so that the center of gravity was towards the closed end of the tray to prevent it from tipping forward while being raised. The trays were being filled, a skip was stationed at the safety mark in the lowermost steel set above. Skips and cages were constructed with briddles which enabled them to be lowered below the installed steel sets. These briddles consisted essentially of a vertical steel arm, on each side of the skip or cage which extended upward and engaged guides on the roadways above. The skip briddles were 50 feet long and the cage briddles 52 feet long. As the lowermost steel set was kept within from 35 to 50 feet of the bottom, it was possible at all times to land a skip or cage on the bottom. When one of the mucking trays had been filled, the skip was rung down to the bottom and the tray raised and dumped into it, after which the skip was hoisted to a position about 15 feet above the bottom until the second tray was ready for dumping. The skip had a capacity of about 75 cubic feet, or two trayloads, and upon being filled it was sent to the surface. The mucking trays were handled by means of two Ingersoll-Rand Type HU single-drum air hoists of 10-horsepower capacity. Their method of mounting and control is explained in connection with an accompanying illustration.

As mucking proceeded, the side walls were continually trimmed of loose material. After some of the bottom rock had been shattered, one of the air headers was lowered and connected. Four Ingersoll-Rand L-54 paving breakers fitted with spades and diggers were then used to work it loose. During this cleaning-up period one tray
sufficed, and the work necessarily proceeded at a slackened pace. Mucking continued until a firm bottom was reached. An average of approximately 250 cubic yards of rock was removed from each cut, and the average time required was 23 hours, 5 minutes. The small volume of water encountered during the mucking was pumped into the skips and hoisted with the loose rock to the surface.

About 30 minutes prior to the completion of mucking, two men were sent to get the second air header and the drilling equipment. The cage which brought them down took back the mucking equipment, thereby immediately clearing the space and enabling drilling to be started without delay. When the ascending cage reached the level where the hoists were stationed, these also were loaded aboard and taken to the surface.

Steel sets were installed after each blast unless it meant a distance between the lowermost set and the bottom of less than 35 feet, as it was found that considerable damage resulted from blasting when the steel was carried too low. The maximum distance to which steel could be carried above the bottom was governed by the bridles on the skips and cages, namely 50 feet. When necessary steel was placed immediately after blasting, although it was preferably left for the morning shift. A stage or dummy set of steel was hung by chains 2 feet below the lowermost regular set of steel to serve as a buffer in blasting. This set was a regular shaft set with double dividers strapped together and with tamarack timbers tied to the underside between the flanges of the members. This stage set took the brunt of the blows from the blasts, and very few replacements of shaft steel had to be made. Whenever the stage set became so badly bent or twisted as to impair its usefulness, it was cut into pieces with a torch for removal to the surface and a new set substituted.

The first step when installing steel was to lower the stage set 10 feet below the lowermost permanent set by means of four 1-ton chain blocks hooked on to the second lowest set. A plank platform was then placed on the stage set, and the hanging of wall plates was started. All steel members were carried down the shaft in cages or suspended underneath them. After the two wall plates had been hung on studdles, the end plates and dividers were installed and the set was lined and blocked. The lining was done with the aid of 1-pound weights suspended from diagonal corners of a plumbed set above, the suspension wires being held by a template set over the flanges of the wall plate and the end plate at a point 6 inches from both inside faces of these members. Blocks, which had been cut into prescribed lengths on the surface, were placed at both ends of the dividers, wall plates, and end plates and made tight with pine wedges. Hardwood lath, 2 inches thick and 6 and 8 inches wide, was cut into lengths just short of the 8 feet that divided the steel sets, and these were then inserted vertically in the flanges of the wall and end plates. They were made tight by wedging 2¾-inch filler pieces between the inside flange of the steel and the lath. Two sets of steel were placed and lathed during one installation period. Guides for the skips and cages, made up of 6x10-inch dressed yellow pine 85 per cent hearts, were framed to span two sets of steel and were secured with ¾x7-inch bolts. After the sets were in position, a length of air pipe was lowered underneath the cage and bolted in place.

ARRANGEMENT OF AIR HEADER
Compressed air was carried down the shaft in an 8-inch line to within 100 feet of the bottom, where a valve and tee were installed. To provide air at the bottom, headers, such as the one illustrated, were connected at either side of the tee. These two headers supplied air for fourteen drills and four blowpipes bearing sets were installed at intervals of about 150 feet. The bearers were of 12-inch I-beams which were spaced by 10-inch channel iron. In the course of construction these bearers were strapped to the shaft set with ¾-inch straps. The channels were then put in position and 15-inch tamarack sprags used to support a set until such a time as concrete could replace them. To insure a bearing set being level, it was customary to level up a shaft set a short distance above the elevation at which the bearers were to be installed. Iron ladders were provided in lengths of two sets; and protective collars, of electroforged-steel grating and made up in two pieces, were put in place.

No regular steel crew was maintained, and all shaft construction work was done by the regular force of miners, who also handled the successive operations of drilling, placing powder, and mucking. All told, the organization consisted of 79 men, of whom 52 were miners. The remainder of the personnel was as follows: one captain, four shift bosses, two construction men, one electrician, one pipeman, three landers, three truck drivers, six hoisting engineers, five shopmen, and one powderman. Work was carried on 24 hours a day and
seven days a week. The miners and shift bosses worked four 6-hour shifts, one shift boss and thirteen miners to a shift. An 8-hour shift applied to all other labor except shopmen, who worked one 10-hour shift only.

All bell, power, and light lines were in the ladder road. Light was supplied at the bottom by an electric fixture suspended about 15 feet above the working level and having cellophane over the globe guards to protect the globes from being broken by material blown from the drill holes. Sigaphones on both cages made it possible for them to be stopped or started at any point in the shaft. An induced-current telephone was the means of communication between the cages and the collar or power house. Steel and other equipment were ordered from the bottom by means of an electric-bell signal system. Air and lights were turned on and off in a similar manner.

The utmost care was taken to safeguard the workers from injury. No person was allowed to enter the shaft unless he wore a hard hat, and strict enforcement of this rule reduced the number of head injuries to a minimum. Descending skips and cages stopped at the safety mark at the lowermost set of steel, and were moved lower only upon signal from the bottom of the shaft. Safety chains attached air hoses to rock drills to prevent the hoses from whipping in case they became detached.

The work was in charge of Captain Henry Eplett, and to him, because of his untiring efforts and enthusiasm, must go much of the credit for the successful completion of the undertaking.

GRAVITY CONCENTRATION OF IRONWOOD FORMATION

BY FRANK J. TOLONEN, HOUGHTON, MICH.*

A report on the laboratory work on iron ore beneficiation for the years 1932 to 1935 inclusive. It deals with the treatment of sizes coarser than one-sixteenth of an inch from samples from the Gogebic Range.

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INTRODUCTION

The winning of metals is not a static art. Although at times the progress may be hardly perceptible, at other times under the stimulus of new discoveries, the changes become very marked. At present the use of scientific methods in conjunction with improvements in the materials of construction, in the methods of manipulation, and in the control of processes has made possible many new ways of winning metals. Because of this progress in metallurgy and the improvements in transportation, the cost of metals has decreased despite the hundredfold increase in consumption since the Napoleonic wars. (6) A larger amount and a greater variety of metals have been used in the past thirty years than in all previous history.

This idea of progress in the art of winning metals and changes in economic condition must be considered in any investigation of future ore supplies. A metal-bearing deposit will become a potential source of supply when a method is devised by which that metal can be won and marketed at a cost of approximating the cost of production from competing sources. Needs of national defense and economy may affect the situation in some countries having an exportable surplus.

There are billions of tons of iron-bearing formations in Michigan in addition to the merchantable ores. At some future date these formations will have to be the source of iron if the steel industry dependent on the Lake Superior iron ores is to continue to exist. A study of these formations with a view to their future use is one of the research projects of Michigan College of Mining and Technology.

The problems in this research are both economic and physical. (4, 16) Some of the economic problems are outside the scope of laboratory investigation, while others depend on future conditions in the United States and the rest of the world. However, estimated relative production costs by the different purposed methods must be considered even in an academic investigation.