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Ground Support Systems in Block-Cave Mining, A Survey

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by

P. C. McWilliams[1] and A. E. Gooch[2]

ABSTRACT

The Bureau of Mines investigated ground support problems in four major block-cave mines to assess the current "state-of-the-art" and to delineate areas for future research. The most significant problem is the inability of present support systems to hold moving ground. Further, as mining progresses to greater depth, this problem becomes more serious. Providing adequate temporary support capable of withstanding primary and secondary blasting is the next most significant problem. A detailed analysis of the effect of mining methods on support problems is also recommended for future research.

INTRODUCTION

Currently, productive block-cave mining in the United States is confined to six or seven mines. These mines produce a significant amount of ore--one block-cave mine is the largest underground ore-producing mine in this country. Further, many mining engineers predict that block caving will be more prevalent in the future as ore is mined.

The study was primarily concerned with ground support problems, a primary mission of the Bureau of Mines Spokane Mining Research Center. Four mines were surveyed; the one with the most acute ground control problems was studied in detail. This mine is also much deeper than any of the other mines; the complexity of supporting was compounded as the mine incurred more and more overburden. Looking ahead, it is reasonable to hypothesize similar problems for other deep caving operations.

The objective of this study was to recommend research directions in ground support for block-caving mining. These recommendations are detailed in the concluding section of this report. The primary recommendation--to install a sand-backfill continuous-yielding support medium in moving ground--has been particularized to mine 3, the deep mine with ground control problems. Present support is analyzed, and a cost analysis for installing a continuous-yielding support in a crosscut is included. This analysis indicates that such

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[1]Mathematical statistician.
[2]Mining engineer.

a system would be financially feasible if the continuous-support medium could be virtually maintenance free.

Although a steel liner is presented here as the support mechanism, other possibilities are also worthy of study--aluminum and concrete come to mind. The selection of support media is influenced by the local availability of natural resources.

The very nature of block caving--requiring the ground to move freely in the cave area while it remains intact in the work areas--presents a complex engineering problem. Currently, every support imaginable--steel, concrete, wood, and yielding steel supports--has been tried, to varying degrees of success. The industry is now turning to shotcrete in attempts to increase the overall efficiency of the support installation system. Unfortunately, secondary blasting creates problems with all current supports.

As part of this study, accident statistics--both fatal and nonfatal--were analyzed; the results are contained in appendix B. Not surprisingly, ground-support-related accidents make a major contribution to the total accident picture, accounting for 24 percent of the 1971-72 nonfatal accidents.

It is apparent that a new support system in block-cave mining would be of great benefit. Hopefully, the information reported here will aid in activating the necessary research.

ACKNOWLEDGMENTS

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BACKGROUND

General Discussion

Block caving is a mining method which utilizes gravitational forces to progressively stress and eventually fragment the ore body so that it may be drawn or mucked and removed (9, 14).[3] Extensive development consists of haulageways, which, depending on the structure of the deposit, may or may not be in the ore body; a network of transfer raises and grizzly or slusher drifts; a system of draw holes; and finally undercutting of the ore body itself to initiate caving. The term "block caving" originally referred to caving of isolated blocks, but in current usage of the term, all forms of mass, panel, and block caving are discussed under the general heading of block caving.

[3] Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

Block caving is initiated from a single undercut level, in contrast to sublevel caving in which caving actions is controlled by retreat drilling and blasting along a series of relatively closely spaced intermediate drifts or "sublevels" within the ore body (3). Caving action in sublevel caving is generally induced by blasting, although in some older mines, caving action was induced by simply removing sublevel drift support.

In fact, many mines use a combination of mining methods to achieve the desired results--for example, shrinkage stopes (9) are sometimes used to establish the boundaries of a block-cave area. For deep massive ore bodies, caving is achieved by dividing the ore body into several vertical levels or "lifts." Thus when a lower lift is worked, ore and other matter from the upper levels is also drawn. In mines with a relatively shallow dipping ore body, development usually follows the footwall.

Applicability

Block caving is a low-cost-per-ton mining method particularly suited to massive low-grade ore bodies with suitable fracturing characteristics. Some desirable deposit characteristics are ore structure containing joints, planes or rivulets which reduce mass strength, thus rendering the ore amenable to caving and crushing action; competent footwall rock for permanent development openings such as haulageways or conveyor systems; and capping resistant to crushing to avoid excessive dilution. Thorough study of deposit characteristics and careful engineering planning prior to commitment are essential to a successful operation, for extensive development is required prior to production.

Some History of Block Caving

The following discussion is a brief historical sketch of some highlights in block caving mining in the United States.

Block caving is reported to have started at the Pewabic iron mine in the Menominee Range, Michigan, in 1895 (9). Briefly, the mining process was as follows: Blocks were cut nearly free on the ends, with open stopes; then they were undercut at the haulage level, leaving random pillars sufficient for safety. Finally the pillars were blasted, and the block was allowed to settle for several months. When the ore was considered sufficiently crushed, the block was reentered, and spiling and timber supports were used to develop a system of extraction drifts. Ore was mucked until waste appeared, the timber was blasted, and mucking was resumed, retreating back to two central extraction drifts.

The first instance of caving into chutes in order to draw ore as it caved, rather than reopening caved areas, is noted at the Mowry copper mine, Arizona, in 1907 (9). To develop the block in this system, a square-set stope, two sets high, was opened under the entire block. The upper floor was lagged to form a series of pockets into which ore caved; the ore was then drawn through chutegates in the sill sets.

Subsequent caving mines generally left a pillar of some form between the main haulage level and the undercut level. The first example of this method was the Tobin iron mine, Menominee Range, Michigan, in 1911 (9). More modern operations generally have a grizzly level between the main haulage level and the undercut level, providing both access to the block and draw control above the haulage level. Further, relatively closely spaced finger raises with chutes connect grizzly levels with the caving block; a network of branch raises provides ore passage to the haulage level below. This is current practice in all gravity draw systems. One of several early southwest caving mines was the Boston mine, Utah Copper Co., Bingham, Utah, where the techniques employed preceded block or mass caving practices (2). In this system, after haulage and raise development, the ore body was mined with a succession of adjacent shrinkage stopes partitioned by pillars. When the stopes were mined to capping, the pillars were undercut and caved along with the drawing of the shrink-stope ore.

Early practice at the Ray copper mine, Arizona, about 1915 (2), was similar to the earlier Boston mine method. Evolution of this method to a block-caving method occurred at the Ray mine, which thus became the first mine in the Southwest to use caving on a large scale (9).

Contemporary with the caving at the Ray mine, other southwest copper operations evolved caving methods, notably the Miami, Inspiration, and Ruth mines. The Inspiration mine was developed as a caving mine; however, at the Ruth and Miami mines, caving methods evolved from the prior method of shrinkage stoping and undercutting (thus caving) the remaining pillars.

An interesting sidelight is the successful caving of a limestone deposit at the Crestmore limestone mine, California, in 1927-30 (13). Mine development was basically the same as that used in the Arizona copper deposits; however, the mass strength of the ore was such that a block had to be freed by shrinkage stopes on all four sides prior to undercutting and subsequent caving. During caving, the entire rock mass was "rocked" by extreme care in ore drawing, providing the necessary stresses to fragment the block without allowing large cavities to form in the undercut. This "rocking the block" operation demonstrated that very competent ore may be successfully caved.

The Climax molybdenum mine, Climax, Colo. (4), provided an innovation by omitting the transfer raises, made possible by replacing the grizzly level with a slusher drift; the ore was then scraped from a series of draw points through a drop point directly into cars in the haulageway below. This eliminates extensive ore pass development and provides the ability to handle larger boulders, thus reducing the amount of secondary blasting required. Another Climax advance was the use of concrete-supported drifts and finger raises. A recent change is the use of rubber-tired equipment in some of the slusher and undercut drifts. Load haul-dumps are now replacing slushers in undercut drift development. The Henderson Amax project, presently under development, is designed for all trackless equipment in the production areas.

The preceding is a very brief sketch of the history of block caving. Many innovations are excluded as are the many overseas block-cave mines--that provide further examples of this mining technique.

Potential

The following discussion is based on a paper by C. L. Pillar (10). Two points in Pillar's paper are of particular interest. One is the obvious need for better and more economical ground support as caving operations go to greater depths. The other is the increasing importance of human factors--that is, effort, working environment, and training.

At present the bulk of the world's metal supply is from surface mining. As the economic surface deposits are depleted, the only answer will be more underground mining. (By "surface" is meant deposits of a sufficiently low overburden-to-ore ratio to be economically mined by opencut methods.) Block caving comes closest to approaching open pit mining in cost, but the deposits must be massive to be amenable to this mining method. Two other relatively low-cost methods are room-and-pillar and sublevel caving. The nature of the deposit may determine the basic mining method long before it becomes a question of cost trade off between methods (say open pit versus block cave). As we go deeper for minerals and discover more massive deep deposits, we should see more utilization of the block cave method. Currently planned projects are down to 5,000 feet.

The design and successful operation of present block-cave properties remain largely an art. However, the science of rock mechanics coupled with the development of modern computational tools will increase the soundness of block-cave mine design. Data are constantly being gathered concerning rock-mass behavior, thus clarifying mass properties; however, this is still a very difficult problem in underground design.

Of particular value is the development of the finite-element analysis (16). More recent capabilities are analysis of inelastic materials and three-dimensional structures. We may be approaching the point where we have a good solution for any specific underground opening problem provided we have the right mass properties, original stress, and boundary conditions. Pillar (10) implies that block-cave mining will increase in number manyfold over today's active block-cave mines.

FIELD SURVEY OF BLOCK-CAVE MINES

General Discussion

Four of the six active block-cave mines in the United States were visited. The primary concern was analysis of ground support problems in these mines. It was hoped that any common problems would be revealed, but the first significant finding is that the block-caving operations differ greatly from mine to mine. The deposits range from porous friable hematite and martite to blocky altered schist and granite. In some operations, two or three lifts are mined--that is, caving occurs under previous workings--while other mines employ block caving but follow an inclined ore body to its depth. Further, in two mines mobile equipment has replaced the traditional slusher drift scraper (in one totally, in the other partially). As to support, every conceivable conventional support mechanism is being utilized to varying degrees in these mines. Mining depth ranges from 600 to 3,500 feet, and in general, the problems compound as mining goes deeper. In the following mine descriptions, no attempt is made to detail either the geology or the mining methods, for this information is voluminous and has been already presented in the literature of the mining industry. Ground support information pertinent to each of the visited mines is presented, and tables 1-6 summarize the ground support mechanism used in these four mines.

TABLE 1. - Undercut level support

Mine	Size[1]	Primary	Supplementary	Life	Maintenance
1	5- by 7-foot undercut drifts.	Round wood posts, 6- by 8-inch caps, 6- by 8-inch sills, 2- by 12-inch back and side lagging.	None........	1 month.	None.
2	9- by 9-1/2 foot sub-drifts.	5 or 7 foot bolts, spacing as required.	2-inch shotcrete as required.	3 months to 1 year.	Occasional during adjacent drift advance.
3	All undercutting drilled from slusher drift.	NAp	NAp	NAp	NAp
4	7- by 9-foot drift.	Occasional bolts if required.	None........	6 months.	None.

NAp Not applicable.
[1]Width is the first dimension; height is second.

TABLE 2. - Fingers and draw points support

Mine	Size[1]	Primary	Supplementary	Life	Maintenance
1	Vertical, 5 to 15 feet high.	None............	Concrete mouth and H-beam brow.	1 year.	Considerable, at draw points.
2	Driven 11 by 9 feet at 45° pitch.	Poured concrete in lower 15 feet to give 4-1/2- by 9-foot opening.	None..........	1 to 3 years.	Occasional from secondary blasting, drawpoint abrasion, or ground pressure.
3	4-foot diameter.	None............	Wide-flange or yield steel brow reinforcement.	6 months.	Frequent due to ground pressure.
4	None (LHD units draw from undercut drift).	Poured concrete brows with wide-flange steel sets.	None..........	2-1/2 years.	Occasional from secondary blasting.

[1]Width is the first dimension; height is second.

TABLE 3. - Slusher and grizzly level support

Mine	Size[1]	Primary	Supplementary	Life	Maintenance
1	Driven 8 by 9 feet.	2-foot-thick poured concrete.	Rock bolt during advance prior to concreting.	1 to 2 years.	Variable.
2	Driven 9 by 12 feet.	Poured concrete to finished 7 by 9 feet.	Bolt and mesh while advancing if required, experimental shotcrete.	1 to 3 years.	Considerable due to secondary and adjacent blasting and ground pressure.
3	6 by 8 feet.	40- and 58-pound wide-flange steel sets.	1-inch by 10-foot pipe spiling, 4- to 6-inch post lagging.	6 months.	Frequent due to ground pressure.
4	None.......	None............	None............	NAp.....	NAp.

NAp Not applicable.
[1]Width is the first dimension; height is second.

TABLE 4. - Ore pass and transfer raise support

Mine	Size	Primary	Supplementary	Life	Maintenance
1	4 foot square inside.	6- by 8-inch crib with 4- by 3-inch angles on upper edges.	None..........	1 to 2 years.	Occasional.
2	Driven 10 by 10 feet.	Poured concrete if required.	Bolts and mesh during advance if required.	Life of level.	Do.
3	4 feet square inside.	Crib with angles on upper edges.	None..........	2 years.	Do.
4	10-foot diameter.	Concrete................	.....do.......	4 to 6 years.	Do.

TABLE 5. - Crosscut support

Mine	Size[1]	Primary	Supplementary	Life	Maintenance
1	11 by 8-1/2 feet inside.	Poured--arched concrete.	Steel sets during advance prior to concreting.	3 to 4 years.	Occasional.
2	Driven 14 by 12 feet.	12- by 12-inch timber with 3- by 12-inch lagging.	Poured concrete during subsequent cutout development with rock bolts through finished concrete.	5 years.	Occasional to extensive depending on area.
3	Driven 12 by 10 feet.	58-pound wide-flange steel sets.	1- by 2-inch by 10-foot pipe spiling, 4- to 6-inch pole lagging.	..do...	Extensive.
4	12-1/2 by 10 feet (finished).	Poured concrete with wide-flange steel sets on 5-foot centers.	Occasional bolting.	1 to 3 years.	Occasional.

[1]Width is the first dimension; height is second.

TABLE 6. - Main haulage support

Mine	Size[1]	Primary	Supplementary	Life	Maintenance
1	10 feet 10 inches by 8 feet 8 inches.	12- by 12-inch timber.	Concrete.........	Life of level.	Minor.
2	14 by 12 feet......	12- by 12-inch timber if required.	None.............	..do...	Little.
3	Driven 12 by 10 feet.	Occasional rock bolting.	40-pound wide-flange steel sets when required.	..do...	Practically none.
4	12-1/2 by 10 feet..	Steel sets and poured concrete.	Bolting and shot-creting during advance.	2-1/2 years.	10-percent probability of repair.

[1]Width is the first dimension; height is second.

Mine 1

Producing over 60,000 tons per day, this western mine employs around 2,000 workers. The mine operates 3 shifts per day, 7 days per week. The mine has been productive for 18 years, with expectation of production for at least 40 more years. The mine uses a grizzly drift mining system. The ore body is mined in blocks, with caving heights of 300 to 600 feet. Undercutting (which is heavily timbered) is blasted; ore is then caved through vertical finger raises to the grizzly level. Here oversized rock is broken manually by sledge and dropped into inclined ore passes (60 feet vertically) above the main haulage. Trains collect the ore and haul it to the pocket from which skips hoist it from the mine. Current production is at 2,000-foot depth. The mine is almost classical in design; a typical caving level comprises haulage drifts, ore passes, a grizzly level, finger raises, and an undercut level.

Concrete is extensively used for support; the main, panel, fringe, and grizzly drifts are concreted to varying degrees. Timber, though expensive, is used as both temporary (undercut drifts) and permanent supporting (main haulage, fringe drifts, etc.). The increased cost of timbering is quite a concern at the mine, as is the scarcity of iron for rock bolts, which are used primarily to support grizzly drifts prior to concreting. Many support mechanisms have been tried at the mine, including cushioning concrete with foam and felt; results have been variable. Concrete cribbing is being considered for future work. Yielding sets have also been tried, but were not deemed too successful. Some of the preceding ideas may warrant further consideration, for there is little or no information as to the completeness of past experiments.

The ground varies from very competent to highly fractured rock. If much activity has been present in the area, the more difficult problems have existed at the physical center of the overall subsidence area, where drifts often collapse before extraction is completed. A typical grizzly drift should be maintained for about a year. Surface subsidence is of concern to the mine operators, and some surface facilities have already been affected by peripheral cracking. Facilities along the trace of a major fault have been affected by a slow differential movement caused by ground on one side moving towards the cave area. Support problems are always present because of the dynamic activities at this mine; approximately 4,000 to 5,000 feet of drift is driven (and must be supported) each month. Five main shafts are currently functional to maintain this operation; two more will be added in 1975.

Ground support problems are less prevalent than at other mines. Maintenance crews represent less than 10 percent of the work effort. Some typical points of concern are--

1. Prediction of surface subsidence and cracking.

2. Draw point destruction due to secondary blast damage and abrasion from running muck.

3. Premature closing of grizzly, panel, and fringe drifts.

To elaborate more on support problems, in concreted drifts there are two distinct problem areas: A gradual but persisting closure (squeezing effect), and rapid cracking and destruction of the concrete, requiring secondary steel sets to hold up large slabs of concrete.

The first type of failure has proved very difficult to combat; in incompetent ground this type of failure has shut down some productive areas. The cracking failures tend to deform towards the actively mined areas. Resupporting is often successful for the time required to remove the ore.

In grizzly drifts, the troubles are usually related either to secondary blasting damage or to draw practices. Yielding supports may prove advantageous in a slusher drift. A potential disadvantage of a continuous-support medium would become manifest if there were to be "lifts" below such installation. Drawing a long continuous liner (for example, 150 feet) through the next lift level would be a paramount problem. Perhaps continuous liners can only be used on the bottom lift, or concrete would have to be the continuous medium rather than steel. These problems must be considered realistically prior to further experimentation with a continuous medium.

Concreting will continue to be the primary support medium if at all possible. It is relatively inexpensive, and the mine production is geared to concrete support operations. As mentioned previously, cushioning behind concrete (back-packing) has been attempted with mixed results. By not concreting the floor, the drifts have done better, with the floor taking the weight via floor heave. By mucking off the excess, the drift is left open; 1 to 2 years is the average required time for maintaining the slusher drifts, panel drifts should stay open for 3 to 4 years, and fringe drifts should be open for about 10 years.

In summary, this mine is productive and is capable of coping with current support problems. Surface subsidence is already irksome, and since several lower level lifts are proposed, increased ground pressure at depth is a future problem of some magnitude.

Mine 2

Production figures for mine 2 are impressive; the usual output is over 40,000 tons per day. This western mine has little overburden; current depth is 600 feet below the surface. This mine has been productive for some 50 years.

Currently, 2,200 employees are involved in a 3-shift-per-day, 7-day-per-week operation. A slusher drift arrangement is used.

As to support problems, roof bolting is the first priority problem. For years the mine has experimented with a variety of innovations in roof bolting. Naturally, the bolting system must be both effective and cost-efficient. Cost has eliminated from routine use some of the alternatives--grouted, the hollow threaded rebar and the fiberglass volts cost approximately five times more than does the standard steel bolt. Bolts that emitted noxious odors (for example, liquid resin bolts), were unsatisfactory. The wooden ash dowels sometimes used in coal mines were used for a time with some success; they were more cost-competitive than the other alternatives. Wood blocks were tried under the bolts, but this idea was very poor because the shock from blasting caused the wood to give so that support became nonexistent. Bolting remains the number one support problem at this mine.

Shotcreting is currently replacing poured concrete wherever possible. Some slusher and undercut drifts that were formerly lined with poured concrete are now being shotcreted; immediate support and enhanced stability of surrounding openings subject to blast effects are the motivations for this change. The proponents of shotcrete argue that either material (concrete or shotcrete) primarily maintains the integrity of the surface rock, rather than providing functional support. In shotcreting today, repairing cracks seems to be of concern.

In summary, this mine is highly productive with some support problems. None of these is overwhelming, but the mine is interested in a better but cost-competitive rock bolt and in further information on shotcrete.

Mine 3

Mine 3 is a smaller mine, producing around 7,500 tons per day of ore. Total work force at this eastern mine is 740. Both a slusher and a transfer drift system (located immediately below the slusher drift) are required to move the ore to haulage. Blasting is done directly from the slusher drifts, requiring no undercut drifts. The ore body is massive and irregular in shape. The upper part dips 28°; thus mining follows the footwall down. There are few "lifts" at this mine. This mine has major problems in ground control; perhaps the current depth of mining (3,500 feet, deepest of the four mines) is the cause. Squeezing ground is the main problem, with the opening collapsing from all sides. Incidentally, for the initial 15 years of mining, ground support was not particularly troublesome.

The primary support is steel, often complemented by wood lagging. Steel has been used in may ways; for example, in the haulageway square steel sets composed of 58-pound wide-flange H-beams, steel semiarch sets, and steel with knee braces are used. Depending on ground condition spacing is 5, 4, and 2 feet. Cedar lagging of various diameters (3 to 6 inches) and lengths cut to conform with set spacing is used. Steel is used universally in both subdrifts and slusher drifts (piggyback system); steel posts and caps with cedar lagging are used. During the mid-1960's yielding arches were tried as a ground support, but the results were not satisfactory. Roof bolts are of little utility in this mine.

The subdrifts and slusher drifts in bad ground often close down in a few weeks or months. Closure is not violent but is often sudden and causes both delays and repair problems. Twenty percent of the working crews are assigned to maintenance, and nearly one-third of the total operating cost is devoted to ground control.

In spite of the foregoing difficulties, the mine output is commensurate with the size of work force. Efficiency is favorable compared with that of other mines. This mine is discussed in more detail later in this paper.

Mine 4

This eastern mine produces in excess of 8,000 tons per day. The total work force (office, mill, etc.) involved in mining is about 800, working 3 shifts daily. This mine also works an inclined tabular deposit, and it uses mobile equipment exclusively. No complex maze system of raises, subdrifts, and slusher drifts is present; rather, the haulage drifts are developed in the footwall with a parallel access drift 50 feet below the ore contact. The headings are ring-drilled, drawing from 200 to 300 feet of good ore from each entry. Drawpoints are approximately 50 feet apart.

LHD (load-haul-dump) diesel equipment of various sizes is used to muck, haul, and dump the ore. The newest models are Wagner 8-cubic-yard-capacity machines. Ramps are used for the mobile equipment with a maximum grade of 10 percent for ore haulage.

Conveyor belts transport the ore to the shaft. The ore is initially dumped into a bin with three access entries at different elevations and then passes to the underground crusher.

Priority research problems at the mine are--

1. Mud created by mobile equipment during mucking creates problems at times.

2. Underground support in relation to undercutting abutment pressures.

3. Production sequencing--because the area is currently being mined, there are problems with draw control. Sometimes the bin is filled from above, blocking access from lower fill points, etc.

The mine engineers are satisfied with their present support system. Concrete and steel are primary supports; roof bolting is used in potential roof-fall areas. Shotcrete is utilized primarily to prevent weathering; for example, along the conveyor from crusher to skip. In regard to concreting, the sequencing of driving entries is being currently studied; the idea is to optimize supporting to agree with passage of maximum pressure. For shotcreting to be effective as a primary support medium, it must be placed almost immediately after mucking or driving entries. Then it can aid the rock in attaining a stable equilibrium. Another problem is that concrete is sometimes crushed in newly supported areas. In nonproduction areas, support is roof bolts (four across on 3-foot centers), shotcrete, or concrete, dependent on severity of the problem. In general, support is required only beneath the ore body itself. Between draw points there are pillars spread at 40- to 50-foot intervals which are in the footwall, not in the ore.

Square steel sets are used throughout the mine and have proved generally satisfactory. Although they are not as structurally sound as "braced sets," cost and ease of handling, and of installation are felt to justify their use.

Formed concrete is used both in the haulageway and at all drawpoint brows. Shotcrete is used to fill voids over drawpoint brows. Also, dry-process shotcrete is used--not primarily as a support, but to prevent weathering.

In summary, many ground support problems exist at this mine. A particular problem is found at the brow of the draw points where secondary blasting weakens this point of division between ground that should be stable and ground that should cave. There are other support problems in the haulage area, and areas in fault zones. All fundamental medias of support are used at this mine--concrete, steel, timber, and rock bolts.

DETAILED STUDY--MINE 3

Since its ground control problems were the most difficult encountered, mine 3 was chosen for detailed study. This mine was visited a second time; both authors were on site for 2 weeks each. The mine's management was most cooperative in providing necessary information for the following summary. All cost information is coded, relative to 1974 expenses. Areas of investigation include geology, the mining plant and procedures, underground supports, and cost analysis.

Geology

Figure 1 illustrates the stratigraphic column of Precambrian Negaunee iron and related formation of the cave area. The mine is located on the north limb of an E-W syncline which plunges slightly to the west. The north limb dips about 45° south, bottoming out at about 3,600 feet below the surface. This structure is cut by two sets of faults which trend E-W and SE-NW. Many of these fault zones have been subsequently intruded by diorite dikes which may be up to 60 feet thick. The footwall structure consists principally of a hard massive graywacke; the upper hundred feet, or less, however, is composed of soft massive to bedded argillite, containing an occasional band of martite ore, lying 10 to 30 feet below the main ore body. The graywacke is somewhat jointed in places, and the argillite is often highly jointed and blocky with very little strength. The main ore is composed of massive to bedded, soft, friable, earthy hematite and martite. This ore occurs in thicknesses of up to 300 feet and is usually found above, or in conjunction with, the faults and dikes. The host or cap rock of the ore is a cherty hematite iron formation which is thinly bedded and generally hard and solid. This materials extends

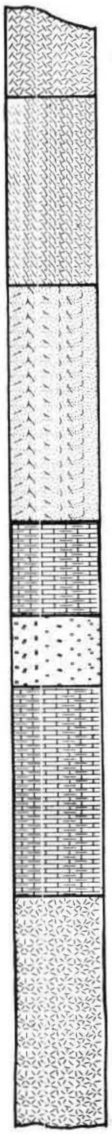


FIGURE 1. - Stratigraphic column of Precambrian Negaunee iron and related formations.

all the way to the surface and is cut by occasional diorite sills of up to 400 feet in thickness.

General Description

Mine development began with the sinking of the No. 1 shaft; work began in January 1941 with first ore production in September 1943. The No. 2 shaft was begun in April 1947, and production started in October 1950. The surface plant was completed in December 1950. The No. 2 shaft is now the only production shaft; the No. 1 shaft is maintained for ventilation and emergency escape. The shafts are about 1-1/2 miles apart. Present production, scheduled for 2 million long tons per year, is from the mine's 11 and 12 levels, which are at 3,250 and 3,500 feet, respectively; the shaft's bottom is at 3,600 feet. The mine presently employs about 620 people.

The man and ore hoists are both of Nordberg[4] manufacture; the drums are 12 feet in diameter with an 8-foot face and a 1-7/8-inch rope. The main hoist is powered by three 1,500-hp dc motors, hoisting 14-1/2-ton skips at 2,800 fpm, resulting in a single skip cycle of about 2 minutes. The man hoist is powered by a single 1,500-hp dc motor hoisting a two-deck 6- by 12-foot man cage at 1,625 fpm; capacity is 70 men. The ore hoist is automatic, with TV monitoring of loading and dumping points. The man hoist is manually operated.

Compressed air is supplied by four Ingersoll-Rand 2,700-cfm compressors powered by 500-hp motors. Normally two are in operation, one on standby, and one available for maintenance. Air pressure is 100 psi at the compressor.

The mine uses all-trolley haulage. Locomotives range from 5 to 15 tons and 25 to 78 hp, respectively, operating a tandem (master and slave units).

[4]Reference to specific equipment does not imply endorsement by the Bureau of Mines.

Cars per train range from 5 to 10 owing to varying ground conditions in the crosscuts. Cars are Sanford Day bottom dump, 8-ton (133-cubic foot) capacity. Track is 60-pound, 30-inch gage, and maintained on crushed rock ballast.

Ore is moved in slusher and transfer drifts with 30- and 40-hp slushers driving 42- and 48-inch box-type scrapers. An underground crushing plant reduces ore to minus 5 inches; however, due to the friable nature of the rock, a screen in series with the crusher (Blake-type jaw) bypasses the undersize, which is the bulk of the ore. Surface handling is by a conveyor belt system from the shaft directly to rail dump or surge piles as required. Waste rock is transported from a shaft pocket by truck.

Haulage headings are drilled with a two-arm hydraulic jumbo. Undercut development is drilled with a post-mounted CP-555 rotary. An RB-12 rotary airleg is used for secondary blasting.

Primary ventilation is provided by two 150-hp and one 100-hp axial fans operating at 4-1/2 inches of water gage, exhausting 183,000 cfm through the No. 1 shaft.

Economic priorities at this mine are somewhat different than those at nonferrous mines in the West. Except for some oversize material which is direct blast-furnace feed the mine "crude" is used as direct pelletizing plant feed. Production and grade are dictated by the plant. Grade is critical (60.1 percent), and the production schedule of 2 million long tons per year must be met. Cost considerations allow no increase in work force, and the ensuing production demands disallow some of the options usually available to the operation, such as a more flexible draw control plan.

Ground Support Systems

General

Ground control is the paramount problem in this mine. The main haulage, developed in a competent graywacke footwall rock, requires only occasional rock bolts; however, as the ore body is approached, a layer of unpredictable, sometimes troublesome argillite precedes the ore itself--a heavy friable martitic hematite. Heavy squeezing ground is typical in the lower levels. Support problems have increased with depth; the present lowest level is at 3,500 feet. This is probably the ultimate level with the present shaft and underground crusher.

The squeezing ground problem appears to be universal in all ore development. Variations in operations and development used in other mines--such as draw control, pillar dimensions and arrangements, development orientation, and block dimensions--have not alleviated the support problems of this mine. The heavy ground is troublesome when developed and remains so until abandoned; continuing operation is contingent upon continuous maintenance. Squeezing ground occurs in unmined areas and under caving blocks being drawn at a maximum rate. Direction as well as intensity of ground pressure is unpredictable. Instances were noted where a drift stood well, encountered heavy ground for 50 to 100 feet, and then was satisfactory again.

Other normally used mining techniques, such as careful blocking of corners and relieving squeezing ground, both of which minimize bending stress, are not applicable; closely spaced long pipe spiling (10 feet) in the back and some spiling in the side are prerequisite to safe installation of sets. This support technique precludes using any of the other support methods mentioned above.

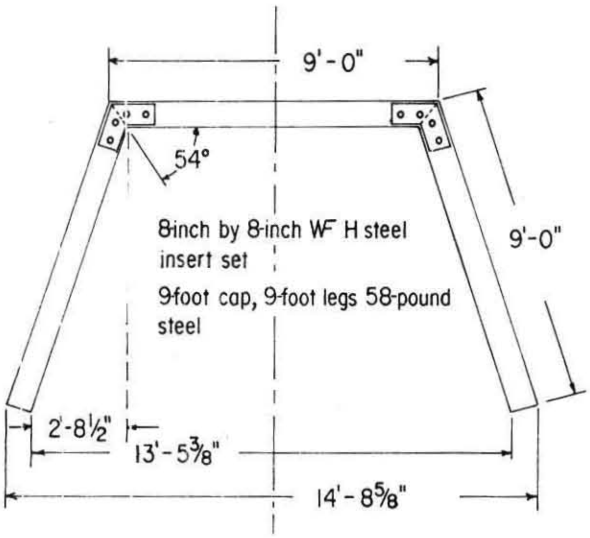


FIGURE 2. - Typical crosscut set.

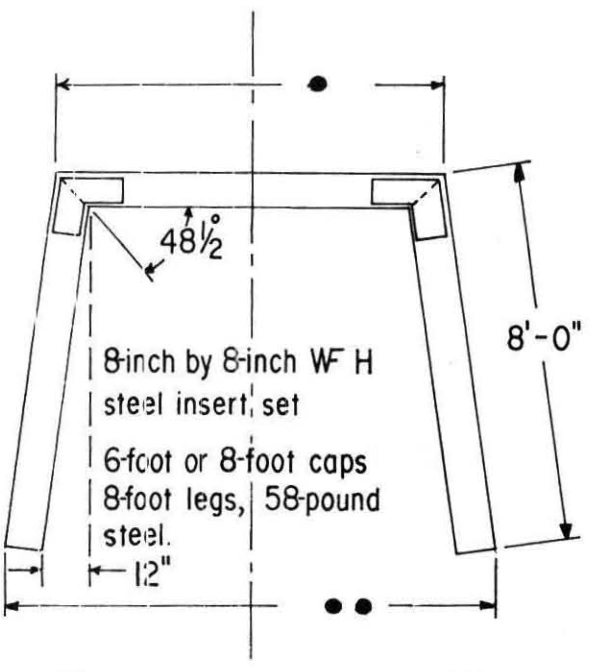


FIGURE 3. - Typical top-timber and transfer set.

Present Support System

Primary support presently installed consists of wide-flange (WF) H steel posts and caps. This includes crosscut sets, transfer drifts, top timber, and slusher drifts as illustrated in figures 2 through 4. Usually, 58-pound section is used, although 40-pound is used if considered adequate. Steel is set with flanges parallel to the drift axis; the cap-point contact is beveled with inserts in the web (fig. 5). Knee braces are used when they are needed (fig. 6). Two 2-1/2-inch by 5-foot angle bridles located approximately 18 inches below the cap and at breast height maintain the haulageway set spacing of 5 feet 4 inches, but the bridles are not structurally important; in fact, the lower bridle may be removed and used in the installation of succeeding sets. Bridles are used the same way in slusher drifts except as 4-foot lengths to maintain 4-foot 4-inch set spacing. Twelve- by eighteen-inch sill plates are used under all posts except those resting on a floor of timber (fig. 7). Top timber (over haulageways or transfer drifts) is bolted directly to the caps below through plates (fig. 3). Figure 8 shows a typical block layout and relationship of different types of sets.

Spiling, the usual temporary roof support mechanism, precedes breaking in all advancing development in either ore or incompetent footwall. Spiling in crosscuts consists of closely spaced 2-inch by 10-foot pipe in the back, and 1-inch pipe in ribs; spacing is dependent on ground conditions. In addition to

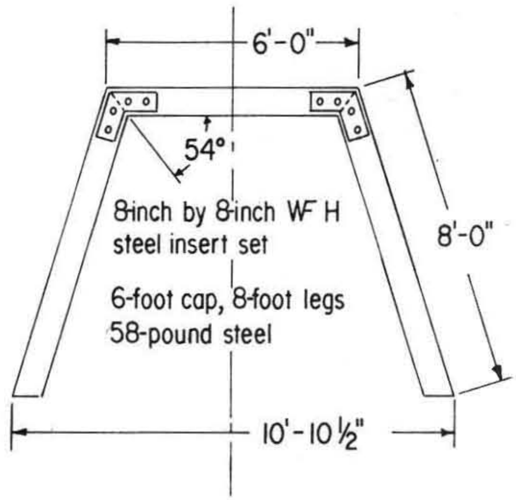


FIGURE 4. - Typical slusher set.

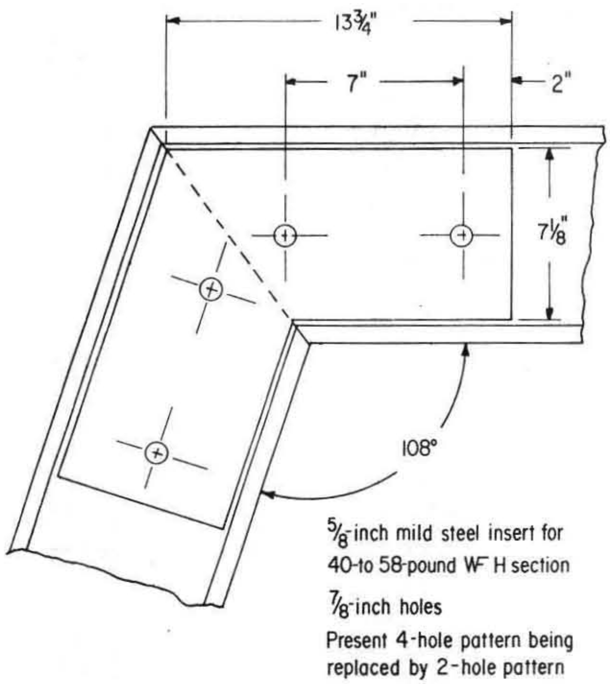


FIGURE 5. - Typical insert.

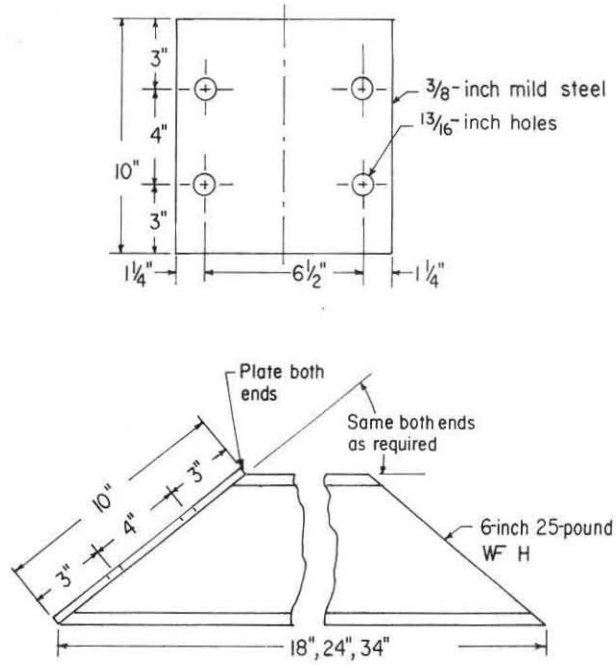
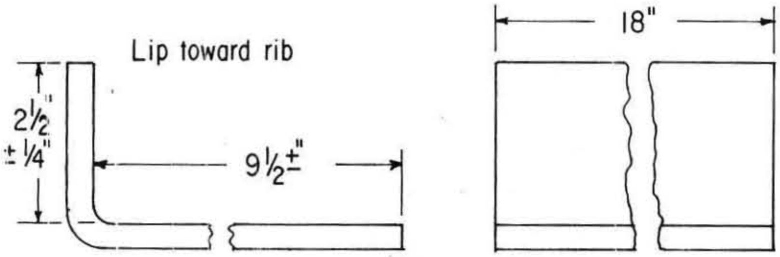


FIGURE 6. - Typical knee brace.



3/8-by 12-by 18-inch mild steel, 23 pounds each

FIGURE 7. - Typical sill plate.

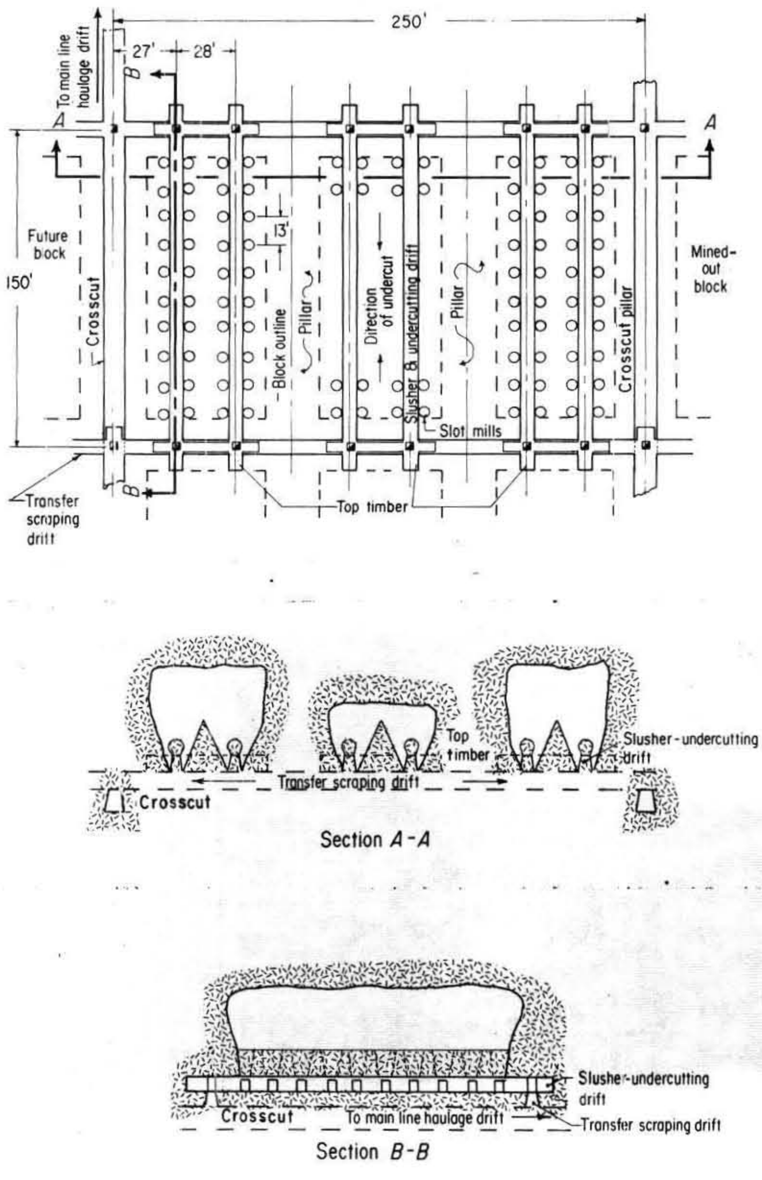


FIGURE 8. - Present predominant development layout.

spiling, additional 2-inch pipe and round pole and 4- to 6-inch lagging are installed in the back and ribs; again the amount of support required depends on ground conditions. This support scheme often results in a "yielding basket" in the back, due to increasing ground pressures compressing the lagging and spiling. The long spiling, though eventually distorted, prevents a complete collapse of support between sets (figs. 9-10).

Support Innovations

Many types of steel support have been tried, as have variations in the mining method. The principal change in mining method was to change the block layout from parallel to the strike to perpendicular (figs. 8 and 11). Although not strictly adhered to, particularly where the extra floor results in loss of ore in areas of greatly reduced ore height, the new layout appears to have reduced support problems by--

1. Reducing asymmetry of stresses perpendicular to opening axis; thus reducing the tendency of the drift to "roll over."

2. Reducing the extent of two-floor opening development (compare figs. 8 and 11), which is obviously more difficult to hold. Although not a support consideration, the operator's opinion seems to be that an additional

FIGURE 9. - Crosscut sets showing distortion of sets and "basket" effect of back spiling.

FIGURE 10. - Crosscut sets with yielding of sets apparent.

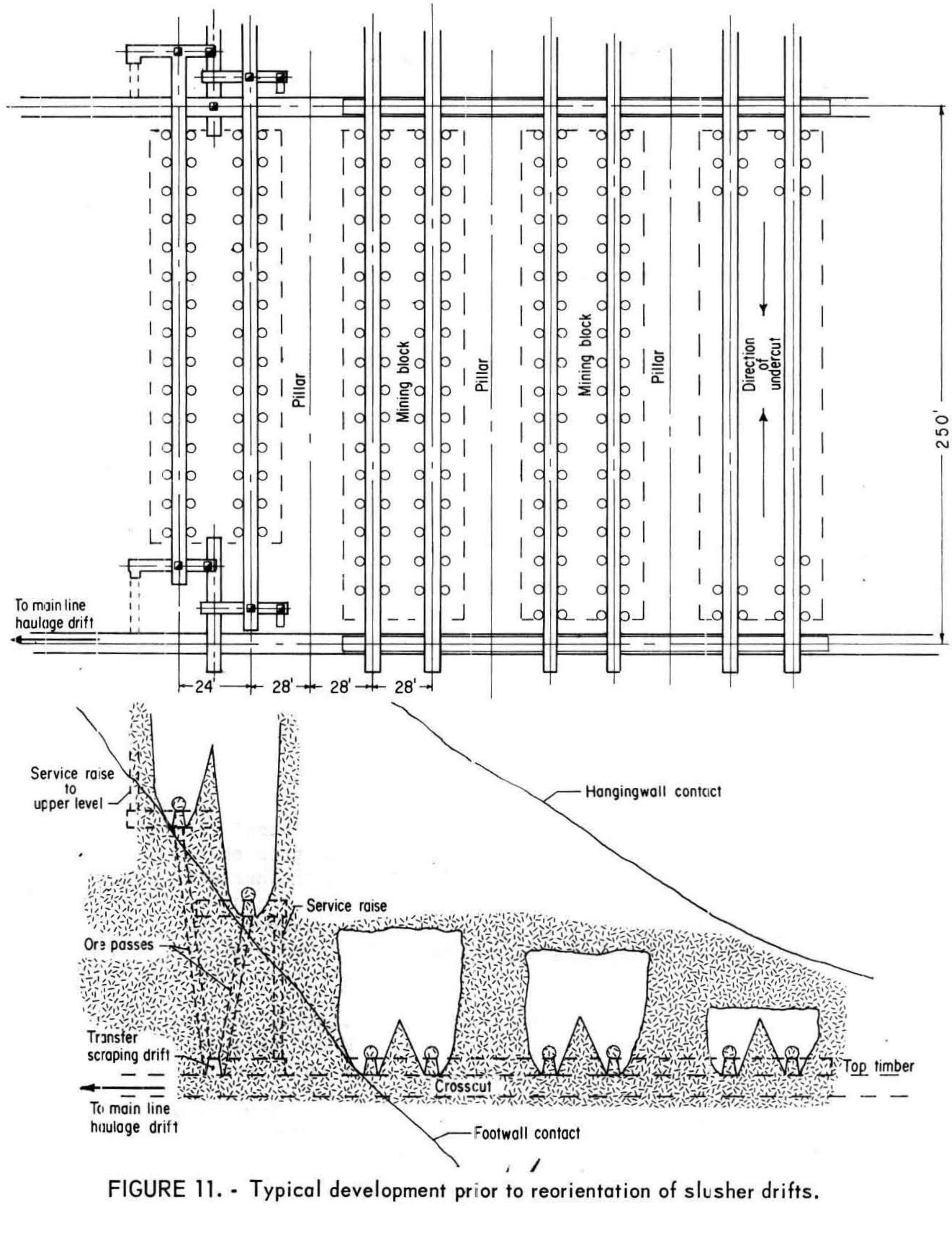


FIGURE 11. - Typical development prior to reorientation of slusher drifts.

benefit derived from the new system is the capability of storing muck in the transfer drifts, thus increasing loading and haulage efficiency.

As previously mentioned, yieldable steel horseshoe sets were used quite extensively in the upper mining levels, but were found to be completely inadequate in the present lower mining levels.

In the short lengths of bored haulageway present at the mine, support was typical circular tunnel steel, 16-pound H section assembled in quadrants with butt plates at spring, invert, and back center lines. Although some failure was noted, this structure was not used in areas of heavy ground requiring extensive support; therefore, conclusive evaluation of its performance cannot be made.

Prior to the present single H-beam support, posts and caps were fabricated from two 4- by 8-inch, 23-pound H-beams welded at the flanges to form a box section of relatively equal modulus (2/3:1) in both directions. This was a satisfactory section, but is now replaced by the single H-beam owing to costs.

Two innovations tried at another mine in this area with very similiar ground conditions were the West German-designed Toussaint-Heintzman (T-H) arches and a cap and hydraulic prop support.

The T-H arches proved to be somewhat more difficult to install than the typical U.S. design. As is frequently the case in differences of opinion between a user and a manufacturer, the manufacturer felt the arch was the best possible heavy ground support and attributed difficulties to improper installation, plus lack of attention to bolt torque at installation and periodically during the life of the opening. We cannot determine the difficulties precisely but would assume, given this mine's serious concern with ground support, that every attempt was made to install and maintain the arches properly.

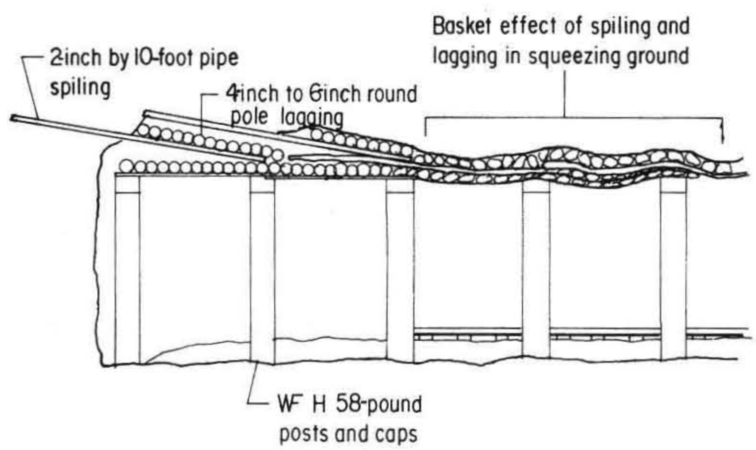


FIGURE 12. - Cross section of typical crosscut.

Steel caps with hydraulic prop posts were installed in slusher drifts in hope that the support would yield to accommodate increasing ground pressure. Failure occurred when side pressures "locked" the cylinders, preventing movements in props. When this occurred, the props soon buckled inward.

Typical Crosscut
Support Installation

Table 7 shows a job breakdown with typical task times for the installation of a crosscut set. Figure 12 shows a cross section of support as presently installed. Although the temporary protection of back spiling is not always required, it is used universally (in ore body development) because of its utility in holding eventually squeezing ground. (Note the "basket effect.")

TABLE 7. - Task breakdown, crosscut advance

Operation	Description	Time required
Drill..........	40 to 42 holes, 6-foot round, 5 or more holes in each rib (1- by 10-inch spiling), 12 to 15 holes in back (2-inch by 10-foot spiling).	3 hours if no problems.
Spile..........	Place back spiles with jumbo, side spiles by hand.	15 to 20 minutes.
Shoot..........	Ventilate to clear smoke..........	20 to 30 minutes.
Bar down.......	Also lag over spiling............	1 hour.
Muck...........	9 to 10 8-ton cars................	3 to 4 hours minimum.[1]
Sets...........	Erect and install back lagging....	1-1/2 to 2 hours.
Miscellaneous..	Switching equipment, setup........	1 hour.
		10.1 to 11.8 hours total.
Periodic.......	Vent and water line installation. Minor maintenance. Help electricians with trolley line installation. (Track crews install permanent track and ballast.)	

[1]Frequent delays due to switching and main haulage train interference.

Holes for the round and spiling are drilled with one setup of a two-arm hydraulic jumbo. Spiling is placed prior to blasting; thus the miners have the protection of a double layer of spiling at all times. Mucking is conventional, with a 1-cubic-yard mucking machine. After barring down, supplementary 2-inch pipe and round lagging are placed as required.

Hitches are located for the sill plates; then one post and cap are completely assembled on the sill. The other post is temporarily left free to dangle from the cap, held by a single bolt through the insert (fig. 13). Figure 14 shows the assembled opposite corner set. The three-piece set is then lifted into position with a bracket and chain attached to the lip of the loader. Longitudinal spacing is assured by attaching a lightweight angle "bridle" to adjacent sets. The last step after posts are properly positioned is to complete the assembly of the "dangling" post and cap. Pipe and timber back and side lagging complete the installation for the next cycle.

In crosscut advance, the long-term average has been 1.1 feet per man-shift. However, in 1973 and 1974, due to short headings and bad ground, this rate was reduced to 0.75 foot per man-shift. A good advance rate is 1.33 feet per man-shift--this represents a one-set (5 feet 4 inches) advance with two-man crews for two shifts. Blasting occurs as required rather than at fixed time intervals, in contrast to the situation at most mines.

FIGURE 13. - Fifty-eight-pound wide-flange steel set in position prior to final assembly of post and cap.

FIGURE 14. - Right-hand corner of set shown in figure 13. Vertical misalinement is due to intentional lowering of track grade at this particular point.

Support Problems

With six to eight four-man crews on main level repair, utilizing 16 to 20 percent of the underground work force, underground support maintenance is a major cost item and could be the determining factor in the life of the mine. The primary concern is with the main level crosscuts, which require a rather long life of about 5 years. Further, a minimum critical size must be maintained in those crosscuts. The Federal mining code requires a minimum trolley wire height, a 30-inch minimum manway clearance, and no storage within 24 inches of track clearance. Additonally, efficient haulage depends on good track conditions. Transfer and slusher drifts also require extensive repair at times; however, because of their smaller size, less critical operating conditions, and relatively short life (6 months average), they are less of a maintenance problem.

The heavy ground in this mine has a most visible effect on the supports. A typical common distortion is compression of the inner flanges of the H-sections at the juncture of the post and caps (figs. 15-16). The bolts in the inserts have little structural value but simply hold the inserts in position and will frequently shear when deformation becomes apparent. The inserts become "locked" in place as ground pressure distorts the sets. Figures 17 and 18 show twisting and bending of caps. Knee braces, where used, frequently failed in bolt shear. Other instances were noted where the braces held, but resulted in distortion of the adjoining cap (figs. 19-20).

Gross deformation of openings was apparent both in crosscuts and in slusher drifts (figs. 20-22). This can result from any combination of back pressure, side pressure, or heaving of sill. Track heave of up to 4 feet was observed. The 3/8-inch sill plates are sometimes completely penetrated by the posts. Tension cracks in inner flanges of slusher posts due to bending from high side pressures were noted.

Near the edges of the ore body, areas were observed in which the sets leaned longitudinally owing to differential ground movement between the sill and the back.

Repair

To speak generally, repair consists of maintaining the continued utility of the opening. The ultimate "repair" is driving a new opening. One area was observed in which a third section of parallel crosscut was driven before it could be kept open. In spite of the closure of the first two drifts, the final drift stood quite well, indicating a possible relationship between the relief of adjacent ground and ground pressures in the usable drift. An only slightly less drastic procedure is the rather frequent need to completely replace support in an existing drift (figs. 23-24).

FIGURE 15. - Common distortion of sets. Note pipe spiling at upper right.

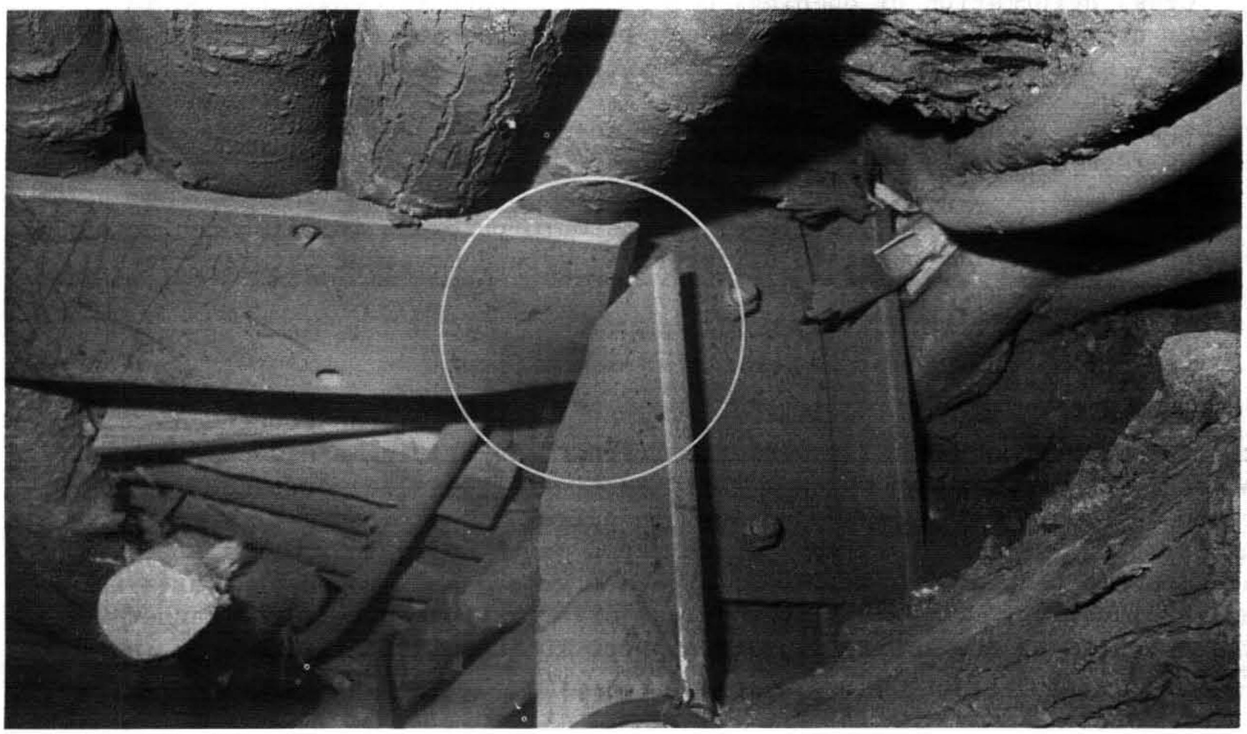


FIGURE 16. - Twisting flanges.

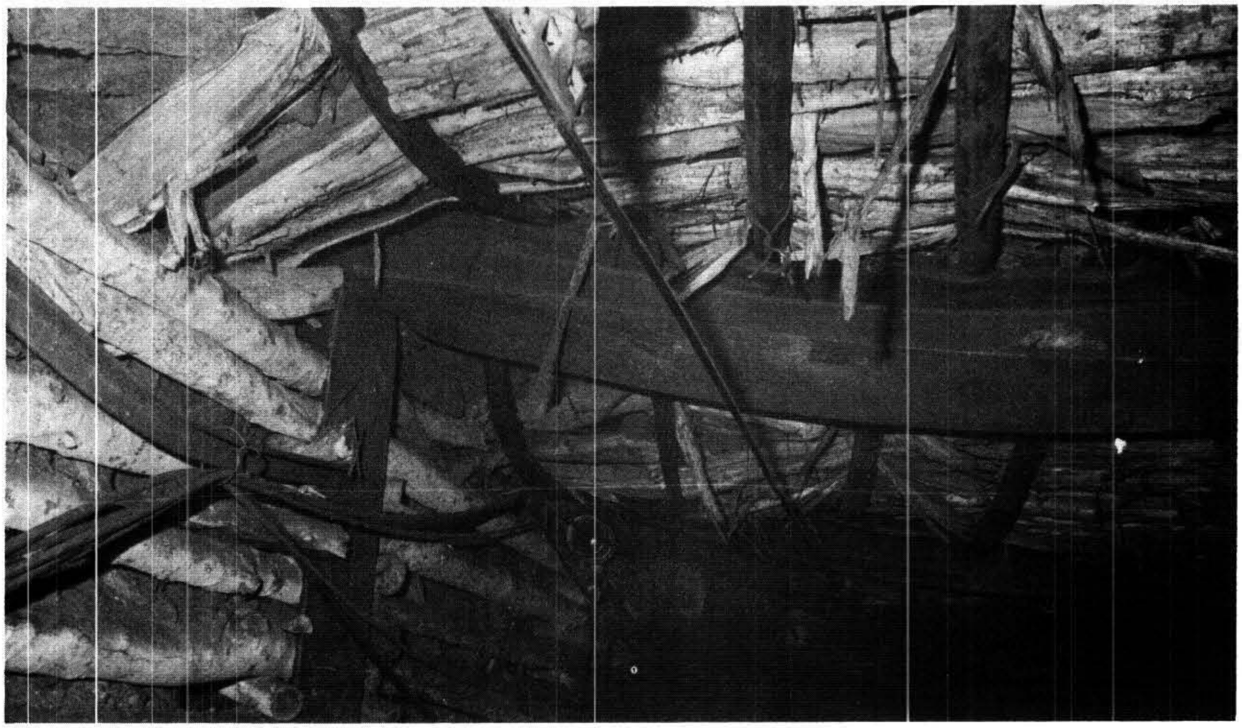


FIGURE 17. - Bending of caps due to back pressure.

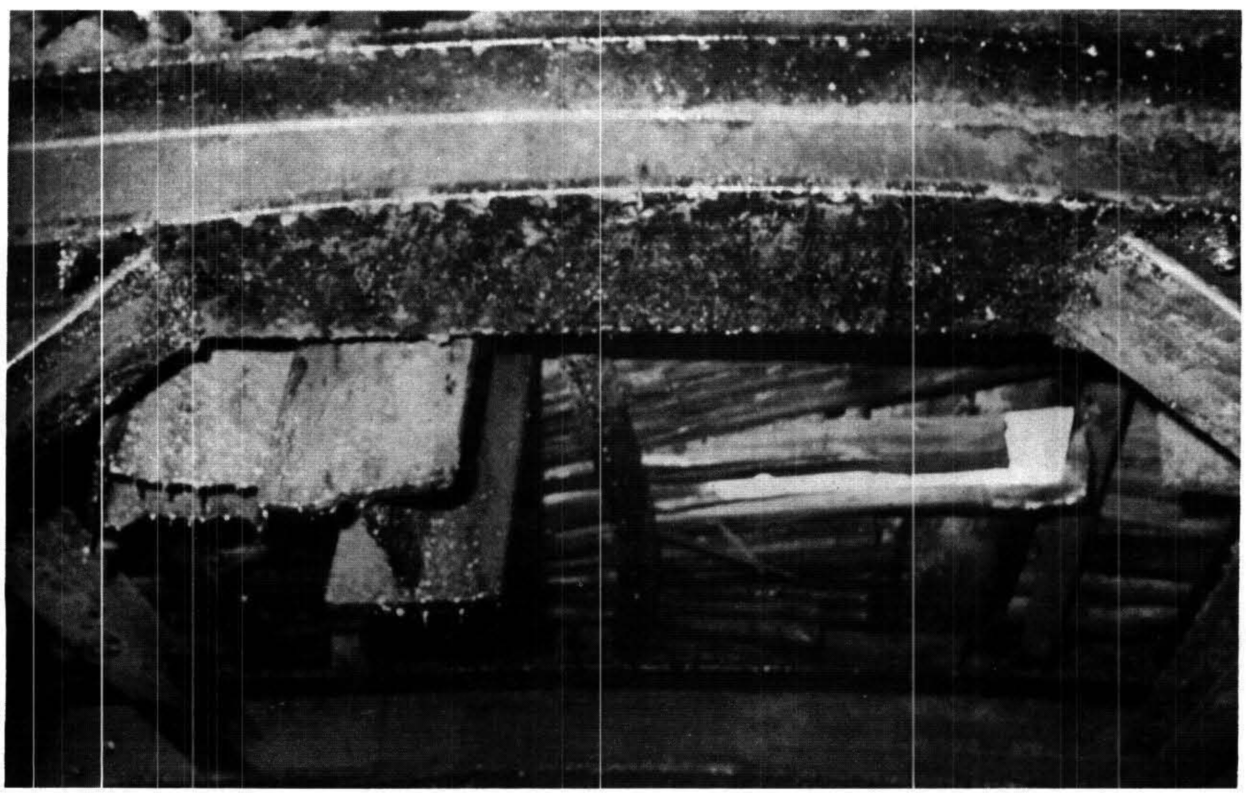


FIGURE 18. - Bending up of cap due to side pressure.

FIGURE 19. - Local deformation at ends of knee braces.

FIGURE 20. - "General" distortion of sets, haulageway.

FIGURE 21. - Crosscut showing side pressure.

FIGURE 22. - Slusher drift in late failure stage.

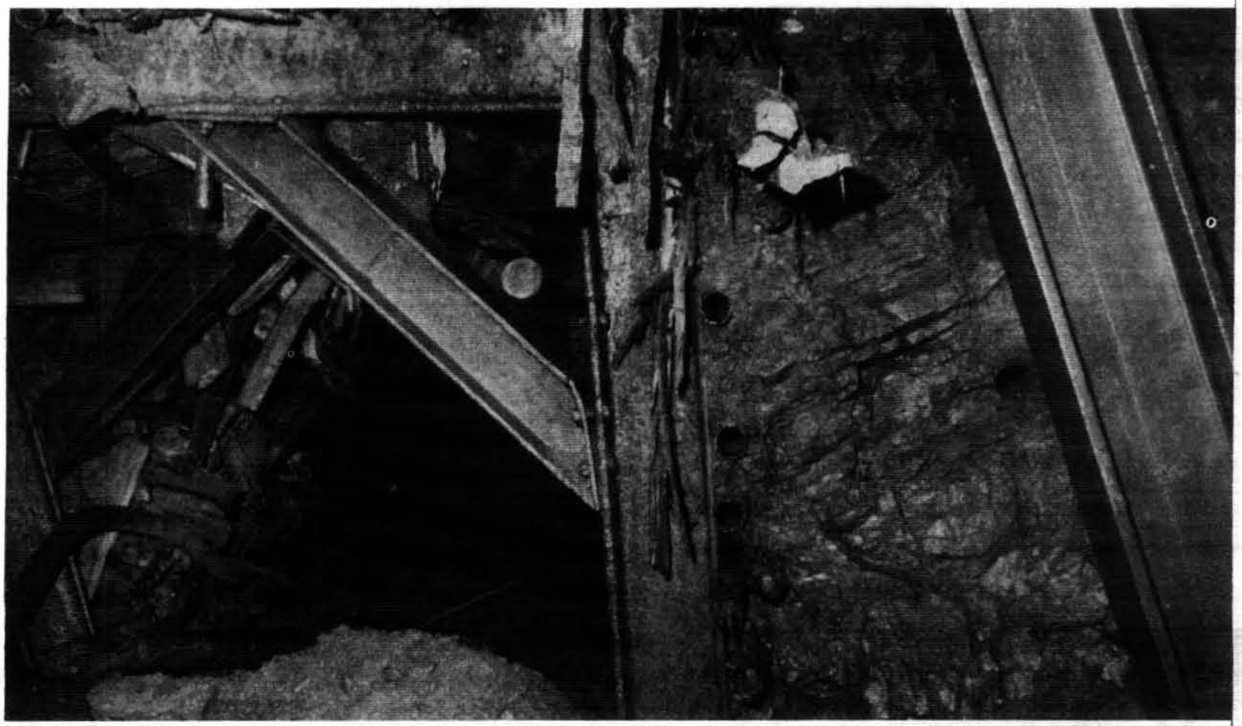


FIGURE 23. - Reopening an existing crosscut—right side view. New post visible in right foreground.

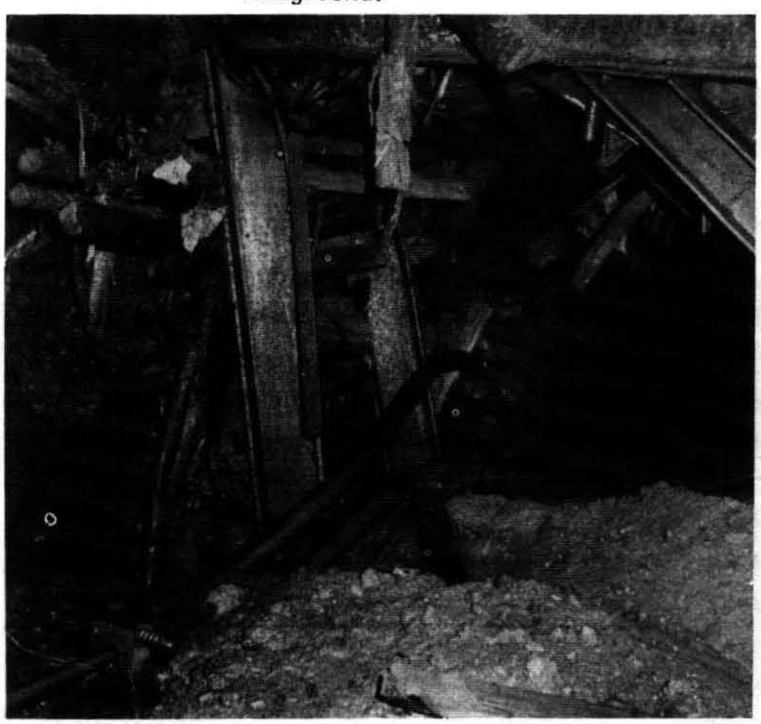


FIGURE 24. - Left rib of crosscut shown in figure 23. New post visible in left foreground.

A common "repair," after deformation of sets has progressed to the point of impeding haulage, is to flame-cut out knee braces to allow car passage (fig. 25). This expedient weakens the support, but no subsequent buckling of the braces was observed in this case.

Where extensive repairs are required in hopper areas, top timber, and slusher drifts, the procedure is typically to cut the member, and then to fit and strengthen as required with steel I's or H's in the same manner that wooden timber would be repaired.

In crosscuts driven in areas of apparently high ground pressure, helper sets may be installed between

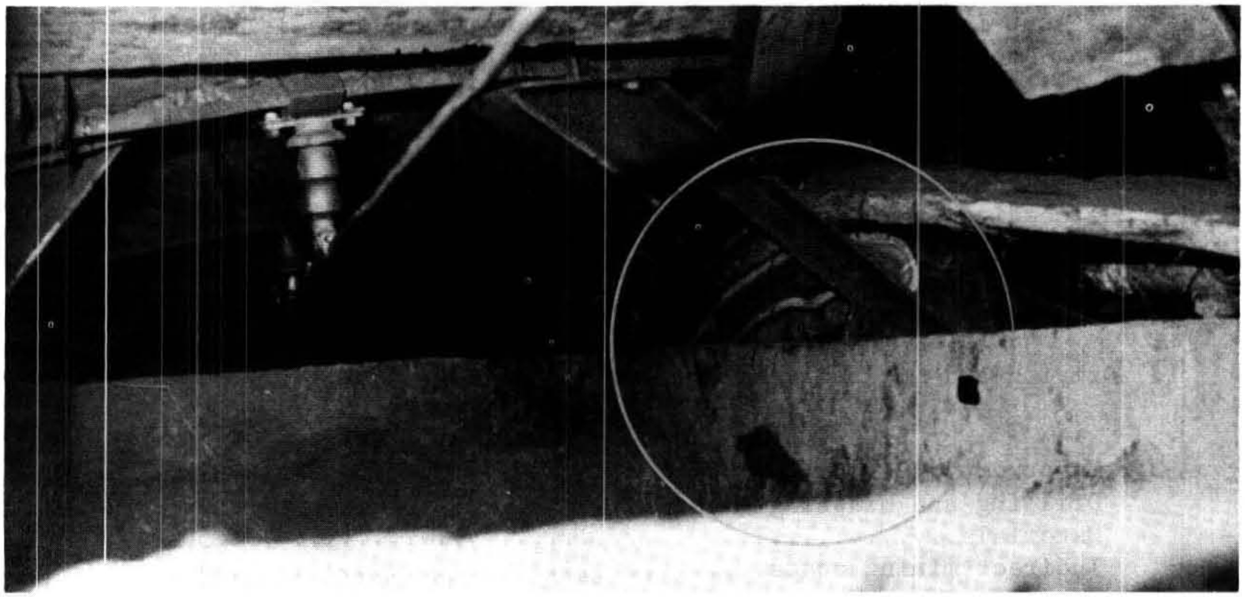


FIGURE 25. - Knee braces notched to permit train passage.

original sets. In areas of longitudinal ("leaning") differential ground movement, "raker" sets, girts, or single posts may be installed at an angle opposite to the leaning existing support.

Attempts have been made to relieve ground behind bending caps or posts; however, this proved to be too costly and impractical.

In areas of heaving ground, track must eventually be removed, the area regraded, and the track replaced.

From the preceding discussion, it is apparent that repair is an extensive integral part of this mine's operation. Any significant alleviation of these problems would improve productivity and safety at the mine.

Comparative Cost Analysis

A continuous-yielding support system is an available but untried alternative to today's ground support systems. A logical initial installation would be in a crosscut, moving through bad ground adjacent to the ore body. Such openings are a current repair problem, requiring approximately 1 to 1-1/2 times the initial installation expense for upkeep and maintenance.

The costing to follow is based on installing a 200-foot-long sand-backfilled steel liner (3/8-inch by 12-foot diameter) in a production area crosscut. The liner support system shall be required to hold the failing squeezing ground for 5 years without major repair. It is assumed that an advancing technique can be devised that will be as rapid and cost the same as today's methods. In order for a new support system to be accepted, it must be cost competitive with the current support system (over entire life of support).

Since a crosscut installation is proposed, the cost analysis is relative to the advancing and repairing of a typical crosscut drift. Items that are relative to mining but independent of the particular support--compressed air, water, ventilation, equipment maintenance, and depreciation are either included as indirect cost or omitted in this first approximation of costing. It is assumed that conventional drill and blasting will be the method used to advance the heading. Costing is coded, with the total present advancing mining cost being expressed in terms of a unit of cost.

Present costs

1. Driving and mucking (cost per foot)

	Units
Manpower	0.33
Drilling and blasting materials	.02
Materials	.25
Indirect mining costs	.40
Total	1.00

2. Cost of repair = 0.3 unit per year x 5 years (expected life of crosscut) = 1.50 units.

3. Total cost of installing and maintaining crosscut = 2.5 units.

Estimated cost--continuous-yielding support

1. Driving and mucking (cost per foot)

	Units
Manpower	0.33
Drill and blasting materials	.02
Support materials	1.00
Backfill materials	Not estimated[5]
Indirect mining costs	.40
Total	[6]1.75

2. Cost of repair = unknown.

3. Total cost = 1.75 plus backfill plus maintenance.

[5]Backfill cost cannot be accurately estimated in a productive sense at this time because technique (transporting sand from surface, grading advance muck, etc.) is not yet firm.

[6]Although the geometries of the openings are quite different--the present opening is a parallelogram, while the new opening is a 13-foot-diameter circle--the volumes removed are equal. Any increase or decrease in volume of rock removed to drive an opening would, of course, affect the totals. Change of shape alone may increase costs; however, in a practical operation, applicable appropriate equipment and procedures should offset this.

Thus to keep this system cost competitive, assuming all benefits are derived from the support system only, less than 0.75 unit of cost can be spent for repair and maintenance. In addition, there should be more benefit than in support alone, for if the crosscut maintenance could be substantially reduced, productivity could be made more efficient by increased output/or by reduction of work force. No estimate is made for this important added benefit.

The major added expense is the increased cost of support material by a factor of 4. Justification of this is very contingent on the support being relatively repair-free. This is obviously essential and must be experimentally proven before any mining company would risk such an increase in capital investment.

RESEARCH RECOMMENDATIONS

Future research that should help alleviate today's ground support problems in block-cave mining may be ordered as follows:

1. Install a sand-backfilled continuous-yielding liner in a working production drift of a block-cave mine with acute ground control problems. The goals of this project would be--

    a. To solve the ground control problem at the particular mine, thereby providing a possible solution to problems at other block-cave mines and any other mines with similiar support problems.

    b. To provide a conclusive field test in a production area, testing the feasibility of this relatively unproven support technique.

2. Investigate the temporary support problem in greater detail--is there a substitute for the conventional rock bolt that will better survive the effects of near-vicinity blasting? Can shotcrete be made more plastic and not fracture due to blasting?

3. Perform finite-element analysis (16) of artificial support systems. In particular, modify existing finite-element codes or use more recently developed codes to determine critical buckling loads and true large-scale deformations during and after support failure (for small-strain deformation theory is not applicable).

4. The need to do long-range research relating mine development and production technique to ground support should be apparent. For example, in considering the optimal size of an underground opening, a conveyor belt drift 40 to 50 square feet in cross-sectional area may have the same materials-handling capacity as a rail haulage drift requiring two or three times that area. Block-cave mining engineers are quite cognizant that production planning has a very important effect on support, but further research in this area is needed.

Block-cave mining appears to be the method of choice for many future mining ventures, and the following specific suggestions for long-range research are offered:

1. Detailed underground studies relating opening deformations to development and production.

2. Finite-element analysis (16) of model underground development systems to determine best methods of development in relation to support requirements.

3. Determination of large in situ mass material properties. One approach might be to "work backwards" with a finite-element analysis of producing mines whose deformation and loads are known and then fit the unknown material properties to these known values.

4. Analysis of the effect of breaking rock on eventual support requirements; for example, directly relating the effect of smooth wall blastings or boring to supports.

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APPENDIX A.--SAFETY REGULATIONS AFFECTING BLOCK-CAVE OPERATIONS

In general, Federal and State mining regulations make little or no specific reference to block-cave mining; however, some points should be made concerning pertinent regulations relative to underground support.

Because of the small number of producing cave operations, individual State mining codes (referenced below) can only affect one or two mines:

Arizona--San Manuel (6)[1]
Colorado--Climax and Urad (1)
Michigan--Mather
Pennsylvania--Grace (15)
Wyoming--Sunrise (8)

However, State regulations generally parallel Federal regulations; in particular, the State of Michigan follows the Federal Code in lieu of having a State code.

Federal and State regulations concerning underground support specify the use of temporary or permanent support as required to maintain safe working conditions; specific support requirements are based on the characteristics of the property. Fire prevention must be considered in connection with artificial support since under conditions timber may contribute to the fire hazard. The Code of Federal Regulations (3, title 30, Chapter 1) states that fan housings and ducts must be fire resistant. Some regulations concerning work in grizzly and slusher drifts directly affect support design in these areas; for example, a mandatory regulation (57.9-103) states "Collars of open draw holes shall be kept free of muck and material."

As the preceding examples illustrate, reference to support in block-caving is usually indirect rather than specific in content.

[1]Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

APPENDIX B.--BLOCK-CAVE MINE INJURIES

The tables in this appendix classify underground injuries in four block-cave mines in a manner that should delineate the safety problem involved in underground artificial support. The data base is all reported lost-time injuries occurring in 1971 and 1972 at four large block-cave mines. "Accident" or "injury" as subsequently used refers to the circumstances of a lost-time accidental injury as reported to the Bureau of Mines, Form 6-1435. Reports were manually examined and catagorized. By definition these injuries are temporarily disabling. Use of a standard time-loss charge for permanent partial disability is noted in the tables (11).[1] This provides a reasonably sized and recent data base for typical caving operations. The reporting method changed in 1973 and would not be consistent with previous reports.

Each cave mine is unique in its mining method, support requirements, material availability, etc., and since there are only a few large cave mines in operation, there would be no point in summarizing or analyzing support problems in terms of a single "typical" operation.

Accident Classification

The best "guess" of accident location is estimated from the accident category and knowledge of the particular mine. Although it would be desirable to classify accidents by location and also to determine whether an incident concerned original installation or maintenance, the brevity of the reports' descriptions generally precludes this approach. It would also be desirable to isolate maintenance accidents from the total accident base, but again this is not possible with the data base.

The classifications used in the following paragraphs suggested themselves during review of the accident descriptions and are intended for the purposes of this study only.

Ground-Fall and Support Injuries (I)

Classification I considers all lost-time underground injuries that are in any way related to falls of ground or ground support. These injuries are subclassified as follows:

(a) Ground fall during support installation.--These injuries are a direct result of falls of ground in the immediate vicinity and during the installation of any type of artificial support. The accidents usually resulted from disturbing the back (overhead rock) while installing a support member, but some occurred while barring down in preparation for support installation.

(b) Ground fall in a working place, not during support installation.--These injuries were generally the result of falls of breast or back while drilling, but also includes all falls occurring while a man was working in the

[1]Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

area. Falls when barring down where support was not be be installed are included.

(c) Support installation--not ground fall.--These injuries generally occurred while handling or assembling members at the installation point, excluding the above. In concreting, this category includes cement burns.

(d) Incidental to support.--These injuries usually occurred while handling support material other than at the time and place of actual installation. (Any injury, excluding (c) above, that could be identified with a specific support item was included.) Occasional accidents were included that appeared to be the result of a specific type of support (other than material handling) such as "struck head on broken lagging" or "arcing on steel post." Injuries that appeared to be independent of the nature of support are excluded.

(e) Ground fall, undisturbed.--Injuries due to a truly undisturbed fall of ground are rare enough to be considered a "freak accident." Falls of ground in working places are shown in previous categories.

Run of Ore (II)

Although injuries in classification II are predominantly the result of falling rock, this hazard is inherent to a caving method and is not a ground support consideration. In addition to injuries occurring directly from falling ore, injuries resulting from barring hangups, breaking boulders on grizzlies, etc., are included. Most injuries occur in grizzly or slusher drifts, and occasionally at car-loading points. Accidents occurring in grizzly or slusher areas not directly attributable to ore flow are excluded.

Not Related to Ground Fall or Support (III)

All remaining underground injuries are included in classification III. The bulk of these involve rock breaking, underground utilities or haulage.

Tables B-1 through B-6 and the discussion of mine 4 (p. 42) summarize accidents at four principal block-cave mines for 1971-72. Tables B-7, B-8, and B-9 are compilations of the previous tables; in table B-7 and B-8 entries are percentages, and in table B-9 they are accumulated totals.

TABLE B-1. - Mine 1: 1971 accident summary

(256 lost-time injuries, 222 underground injuries; total employment 1,900, 41,700 tpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	57	1,027	18.0
a. Ground fall during support installation.	15	295	19.7
b. Ground fall in a working place, not during support installation...........	11	128	11.6
c. Support installation--not ground fall...	16	153	9.6
d. Incidental to support..................	15	451	30.1
e. Ground fall--undisturbed...............	0	0	0
II. Run of ore............................	59	1,449	24.6
III. Not related to ground fall or support[1]..	105	1,715	16.3
Total.................................	221	4,191	19.0

[1] 1 fatality excluded.

TABLE B-2. - Mine 1: 1972 accident summary

(255 lost-time injuries, 187 underground injuries; total employment 2,000, 61,000 tpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	47	915	19.5
a. Ground fall during support installation.	13	371	28.5
b. Ground fall in a working place, not during support installation...........	11	116	10.5
c. Support installation--not ground fall...	14	125	8.9
d. Incidental to support..................	9	303	33.7
e. Ground fall--undisturbed...............	0	0	0
II. Run of ore[1]............................	54	436	8.1
III. Not related to ground fall or support[2]..	83	1,375	16.6
Total.................................	184	2,726	14.8

[1] 2 fatalities excluded.

[2] 1 fatality excluded.

TABLE B-3. - Mine 2: 1971 accident summary

(82 lost-time injuries, 71 underground injuries; underground employment 1,188, 38,466 tpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	7	52	7.4
a. Ground fall during support installation.	2	12	6.0
b. Ground fall in a working place, not during support installation...........	0	0	0
c. Support installation--not ground fall...	2	10	5.0
d. Incidental to support..................	3	30	10.0
e. Ground fall--undisturbed...............	0	0	0
II. Run of ore............................	9	[1]357	39.7
III. Not related to ground fall or support	55	[1]5,387	97.9
Total..................................	71	5,796	81.6

[1]Includes standard time loss charge for permanent-partial disability (11).

TABLE B-4. - Mine 2: 1972 accident summary

(49 lost-time injuries, 41 underground injuries; underground employment 949, 38,168 tpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	5	148	29.6
a. Ground fall during support installation.	0	0	0
b. Ground fall in a working place, not during support installation...........	4	31	7.8
c. Support installation--not ground fall...	1	117	117.0
d. Incidental to support..................	0	0	0
e. Ground fall--undisturbed...............	0	0	0
II. Run of ore............................	6	[1]185	30.8
III. Not related to ground fall or support...	30	[1]968	32.3
Total..................................	41	1,301	31.7

[1]Includes standard time loss charge for permanent-partial disability (11).

TABLE B-5. - Mine 3: 1971 accident summary

(88 lost-time injuries, 86 underground injuries; total employment 740, 7,555 ltpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	25	566	22.6
a. Ground fall during support installation.	5	47	9.4
b. Ground fall in a working place, not during support installation...........	5	124	24.8
c. Support installation--not ground fall...	6	119	19.8
d. Incidental to support..................	8	[1]271	33.9
e. Ground fall--undisturbed...............	1	5	5.0
II. Run of ore............................	15	[1]271	18.1
III. Not related to ground fall or support...	46	914	19.9
Total.................................	86	1,751	20.4

[1]Includes standard time loss charge for permanent partial disability (11).

TABLE B-6. - Mine 3: 1972 accident summary

(78 lost-time injuries, 77 underground injuries; total employment 740, 7,555 ltpd)

Classification	No.	Lost time, days	Av severity (days/No.)
I. All ground-fall and support injuries......	25	315	12.6
a. Ground fall during support installation.	6	58	9.7
b. Ground fall in a working place, not during support installation...........	5	47	9.4
c. support installation--not ground fall...	8	83	10.4
d. Incidental to support..................	6	127	21.2
e. Ground fall--undisturbed...............	0	0	0
II. Run of ore............................	14	201	14.4
III. Not related to ground fall or support...	38	370	9.7
Total.................................	77	886	11.5

Mine 4: Accident summary

In 1971 there were 12 lost-time injuries, including 9 underground injuries. There were 300 miners underground; production was 8,000 tpd.

Four of the nine underground accidents are pertinent to this study. Three can be classed as "Ground fall in a working place, not during support installation." All three were serious injuries (multiple fractures), and the lost time was undetermined since the employee was not working at reporting time. The fourth accident is classed "run of ore" with standard time loss charge of 300 days for a permanent partial disability--partial amputation of right thumb.

In 1972 there were seven lost-time injuries, of which five were underground injuries. As in 1971, there were 300 miners underground and production was 8,000 tpd.

Two of the five underground accidents are relevant to this study. One was a "support installation--not ground fall" accident with 14 days lost; the other was an "incidental to support" accident with 29 days lost.

TABLE B-7. - Distribution of underground injuries in three mines, 1971[1]

Classification	Mine 1	Mine 2	Mine 3
I. Ground fall and support as percent of total	25.8 / 24.5	9.9 / 0.9	29.1 / 32.3
a. Ground fall during support installation as percent of ground fall and support total	26.3 / 28.7	28.6 / 23.1	20.0 / 8.3
b. Ground fall, working place--not during support installation as percent of ground fall and support total	19.3 / 12.5	None	20.0 / 21.9
c. Support installation--not ground fall as percent of ground fall and support total	28.1 / 14.9	28.6 / 19.2	24.0 / 21.0
d. Incidental to support as percent of ground fall and support total	26.3 / 43.9	42.9 / 57.7	32.0 / 47.9
e. Ground fall, undisturbed as percent of ground fall and support total	None	None	4.0 / 0.9
II. Run of ore as percent of total	26.7 / 34.6	12.7 / 6.2	17.4 / 15.5
III. Unrelated injuries as percent of total	47.5 / 40.9	77.5 / 92.9	53.5 / 52.2

[1]The numerator is indicative of the number of injuries, the denominator denotes days lost--both values are expressed as a percentage.
Mine 4 had too few injuries for this type of summary.

TABLE B-8. - Distribution of underground injuries in three mines, 1972[1]

Classification	Mine 1	Mine 2	Mine 3
I. Ground fall and support as percent of total	25.5 / 33.6	12.2 / 11.4	32.5 / 35.6
a. Ground fall during support installation as percent of ground fall and support total	27.7 / 20.5	None	24.0 / 18.4
b. Ground fall, working place--not during support installation as percent of ground fall and support total	23.4 / 12.7	80.0 / 20.9	20.0 / 14.9
c. Support installation--not ground fall as percent of ground fall and support total	29.8 / 13.7	20.0 / 79.0	32.0 / 26.3
d. Incidental to support as percent of ground fall and support total	19.1 / 33.1	None	24.0 / 40.3
e. Ground fall, undisturbed as percent of ground fall and support total	None	None	None
II. Run of ore as percent of total	29.3 / 16.0	14.6 / 14.2	18.2 / 22.7
III. Unrelated injuries as percent of total	45.1 / 50.4	73.2 / 74.4	49.4 / 41.8

[1]The numerator is indicative of the number of injuries, the denominator denotes day lost--both values are expressed as a percentage.
Mine 4 had too few injuries for this type of summary.

Discussion of Injury Data

Due to the brevity of the accident descriptions, a considerable amount of judgment was necessarily involved in classification. Note the number-of-injuries distribution in tables B-7 and B-8 for the same mine, in 1971 and 1972; similiarity of the percent distribution, between the 2 years, indicates a reasonable consistency in evaluating accidents. The mines varied considerably as to the completeness of detail in the accident and injury description, and this report has been conservative in assigning injuries to the ground support category; injuries of questionable description were not included. Note that the data base is only for 2 years; thus, caution must be exercised in drawing overall conclusions, particularly in regard to severity.

The number of injuries appears to be a much more consistent basis for evaluating the hazards of a given situation than time lost. Factors complicating "lost time" and "severity" evaluations are (1) the infrequency and thus unpredictability of extremely serious (much time off) injuries, and (2) the high time-lost values assigned (according to standard time-lost charges) to relatively infrequent injuries resulting in permanent partial disabilities.

Fatal accidents were excluded from the tables due to the high time-lost charge of 6,000 days. To maintain consistency they were excluded from both number of injuries and time-lost tabulations. Also, the rarity, seriousness, and generally unique nature of the fatal accidents suggested that they should be examined individually, and this was done.

Another general conclusion is that support-related injuries are divided into approximately even groupings within subclassifications a through d. It becomes obvious, strictly from the safety aspect, that support problems do not result from the inadequacy of support presently used, but from all phases of its handling, installation, and maintenance. The balance of support-related injuries result from an absence of support.

Individual Mine Comments (see table B-7)

A comparison of the underground hazard of the four mines concerned is shown below (1971 injuries):

	Injuries per man-shift
Mine 1....................	6.2×10^{-4}
Mine 2....................	2.8×10^{-4}
Mine 3....................	5.7×10^{-4}
Mine 4....................	$.9 \times 10^{-4}$

Mine 1.--The large number of injuries provided the most valid basis for accident classification. Although the total number of injuries in 1972 was considerably less than in 1971, the percentage distributions did not change significantly.

A 1969 fatality was the direct result of fall of support material (concrete slab) during drift repair. A contributing factor was disregard of the necessity of some temporary support during repair. This accident would be classified under I-c, "Support installation--not ground fall."

A 1971 fatality was the direct result of fall of ground while drilling the back for bolting without the use of temporary (stull) support. This accident is classified under I-a, "Ground fall during support installation."

In 1972, two fatalities were the result of drawing ore where running rock pulled the men from the grizzly drift into the transfer raise. This is not actually a support problem but would be of serious concern to the designer of a support system in the grizzly drifts of this mine. Classification of these accidents would be under II, "Run of ore."

Mine 2.--Extreme brevity of accident descriptions made injuries in this mine the most difficult to classify. It was apparent, however, that the percentage of support-related accidents was much less than at the other operations. Percentage distribution of the different accident subclassifications of support-related accidents are shown, but due to inadequate descriptions and small number of injuries, they are of questionable value.

In 1969 a fatal accident was the direct result of a fall of unsupported ground in undercut development. This would be classified under I-b, "Ground fall in a working place, not during support installation." The undercutting mining method used at the time has since been abandoned.

Mine 3.--Good accident descriptions made this the easiest of the mines to summarize. This mine definitely has support and ground-fall problems as described earlier in this report.

Mine 4.--This mine has by far the best safety record of the four mines studied. In fact, the difference in number of accidents is a very startling figure. Undoubtedly the safety program is very good, and the mine is also most fortunate in having competent footwall rock.

In 1968 a fatal accident was the direct result of the fall of a concrete (support material) slab during slusher drift rehabilitation; it would be classified under I-c, "Support installation--not ground fall."

Statistical Analysis

A reasonable hypothesis to test is whether the type of injury--ground support or fall, run or ore, injury not related to either of these--is evenly distributed among the three mines. To make this test, the X^2 goodness-of-fit test was applied to the data of table B-9 (5). The test rejected the hypothesis that accidents are proportionally distributed among the mines; there is definitely a different accident distribution dependent on the mine involved. The statistical values of interest are chi-square of data = 36.65, degrees of freedon = 4, and the test (or critical) chi-square value = 9.49 at the 0.05 significance level. Since the tabular value greatly exceeds the test value, the conclusion is that mines differ as to type of accident. Thus, ground support contributes differently in each mine to the mine's total accident picture.

TABLE B-9. - Summary of total injuries, 1971 and 1972, by injury type

Injury type	Mine 1	Mine 2	Mine 3
Ground support......................	104	12	50
Run of ore..........................	113	15	29
Other...............................	188	85	84

Concluding Remarks--Safety Analysis

The preceding discussion indicates that ground control problems are a major contributor to accidents in block-cave mines. Mine 4 has an outstanding accident record relative to the group. As to the overall accident picture, some specific comments are--

1. Two recent fatalities occurred when the miners were rehabilitating massive concrete support.

2. Roughly one-fourth of support-related accidents were "materials handling" accidents.

3. More critical consideration should be given to the need for some immediate temporary protection in unsupported working places.

It is apparent that improvement of ground control procedures will result in a safer working environment, accompanied by an increase in efficiency and reduction in costs in block-cave mines.

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