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# MINING METHODS AND TECHNIQUES USED AT THE RADON LONGWALL OPERATION, HECLA MINING CO., SAN JUAN COUNTY, UTAH

By W. L. Dare and P. M. Lindstrom



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UNITED STATES DEPARTMENT OF THE INTERIOR  
BUREAU OF MINES

(1961)



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UNITED STATES DEPARTMENT OF THE INTERIOR  
Stewart L. Udall, Secretary

BUREAU OF MINES  
Marling J. Ankeny, Director

This publication has been cataloged as follows:

**Dare, Wilbert Leland.**

Mining methods and techniques used at the Radon longwall operation, Hecla Mining Co., San Juan County, Utah, by W. L. Dare and P. M. Lindstrom. [Washington] U.S. Dept. of the Interior, Bureau of Mines [1961]

ii, 54 p. illus., map, tables. 26 cm. (U.S. Bureau of Mines. Information circular 8004)

Bibliography: p. 53-54.

1. Uranium mines and mining -- Utah -- San Juan Co. I. Lindstrom, P.M., joint author. II. Hecla Mining Co. III. Title. IV. Title: Radon longwall operation. (Series)

TN23.U71 no. 8004 622.06173

U.S. Dept. of the Int. Library



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# MINING METHODS AND TECHNIQUES USED AT THE RADON LONGWALL OPERATION, HECLA MINING CO., SAN JUAN COUNTY, UTAH<sup>1/</sup>

by

W. L. Dare<sup>2/</sup> and P. M. Lindstrom<sup>3/</sup>

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## SUMMARY AND INTRODUCTION

The Federal Bureau of Mines is publishing a series of studies on the methods and costs of mining uranium on the Colorado Plateau. This report describes the longwall-mining methods and techniques used at the Radon mine, Big Indian district, San Juan County, Utah.

The tabular Radon ore body lies on the southwest flank of the Lisbon Valley anticline, strikes N. 10° W., dips 7° SW., and has an average thickness of 4-1/2 to 5 ft. The mine is developed by a three-compartment vertical shaft, a haulage level driven beneath the ore, and a system of drifts driven within and parallel to the strike of the ore.

The longwall retreat method used at the Radon mine is an adaptation of similar methods that have long been used in European coal mines. The Radon operation is probably the second metal mine that has steel yielding props to support full-cave retreating longwalls. The other mine is the San Francisco mine at Autlan, Jalisco, Mexico. Variations of longwall mining, either with or without timber support, have been used in other metal mines in the United States. The copper mining companies in northern Michigan adopted retreat methods to increase the rate of extraction and to reduce the cost of ground support as the workings were deepened (2).<sup>4/</sup> One author (12, p. 5) in 1931 referred to a "systematic longwall retreating method" that was used in a Michigan copper mine since 1908. The backs in the approximately 36° dipping stopes were held by large closely spaced stulls; roof control was regulated by the number of stulls used. Staggered retreat faces in the steplike blocks were similar, but on a much larger scale, to a longwall advance method used in the Ruhr district, Germany (1, pp. 19-20).

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<sup>4/</sup> Underlined numbers in parentheses refer to items in the bibliography at the end of this report. Page references refer to pages in the items and not in this report.

T. A. Rickard described (10, p. 383) what he called, "fundamentally a long-wall system," in the underground placer gold mines south of Fairbanks, Alaska. Little or no support timbers were used in these small operations where the irregular and curved longwall faces were retreated from opposite directions through the frozen, buried channels toward the shafts. Similar longwall mining, either advancing or retreating and called breasting or drift mining, was used in placer deposits in California and Montana.

According to Peele (8, p. 505), the fundamental principle of longwall mining is the complete extracting of the entire seam in one operation along a continuous face, leaving no pillars and allowing the roof to cave behind the face. In the longwall advance system employed in some coal mines in the United States, the longwall is a series of tangential faces that are advanced outward from a centrally located shaft pillar.

This publication does not cover the theories on rock pressure that have been developed by many investigators, most of whom are Europeans. Two relatively current publications, both available in English translations, deal with theories on longwall ground behavior and with longwall support equipment. The first is "International Conference about Rock Pressures and Support in the Workings," which covers the proceedings of the 1951 conference held in Liege, Belgium; the other is "Face Supports in Steel and Light Metals," by Dr. Fritz Spruth. Haarmann, in his introduction in Spruth's book (11), listed the following points about back support where yielding props are used:

1. The exposed area of roof should be supported at the earliest possible moment.
2. The supports should be distributed as evenly as possible.
3. The question whether "early" or "late-bearing" supports\* should be used may also be regarded as settled. The roof should be maintained with the maximum freedom from cracks, and this is only possible if every step is taken to prevent bed separation in the roof. In simple terms, the roof should be "left where it is".

\* "Early-bearing props" are yielding props which take maximum load after a minimum yield.

#### ACKNOWLEDGMENTS

The authors wish to thank L. J. Randall, President, and W. H. Love, General Manager, Hecla Mining Co., Wallace, Idaho, for permission to write and publish this report. Mr. Love was formerly superintendent of the Radon mine, and most of the work to inaugurate the longwall system was done under his direction. The assistance and cooperation given by the supervisory personnel and the mine crew are gratefully acknowledged.

A number of articles have been published in which various aspects of the Radon operation were described. Some were drawn upon in preparing this report.



As far as known all articles that deal with the Radon mine are listed in the bibliography regardless of whether or not they were used for this report.

#### LOCATION AND PHYSICAL FEATURES

The Radon mine is in secs. 28 and 33, T. 29 S., R. 24 E., Salt Lake meridian, San Juan County, Utah (fig. 1). The mine is at the northwestern end of the Big Indian-Chinle ore belt. Big Indian Wash, from which the district got its name, lies immediately east of the mine.

The mine is approximately 35 mi. southeast of Moab, Utah. The road is paved except for the last one-half mi.

The topography in the area is fairly rough, and much of the surface is sandstone benches and slopes upon which grow pinon pine, juniper, and sage brush. Annual precipitation is approximately 13 in. The mine plant is near the head of a small drywash that cuts northwestward off the southwestward sloping flank of the Lisbon Valley anticline (fig. 2). The drywash runs northwestward past the Rattlesnake opencut uranium mine to West Coyote Creek at Utah State Route 46. The ground near the mine plant has nearly a dip slope. The south end of the La Sal Mountains is about 10 mi. north of the mine. Mount Peale, the highest peak, rises to 12,721 ft. above sea level. Altitudes within about 1 mi. of the Radon mine range from 6,600 to 7,200 ft. The altitude of the collar of the Radon shaft is about 6,850 ft.

#### HISTORY AND PRODUCTION

Hecla Mining Co. operates the Radon and Hot Rock groups of mining claims through an operating agreement with Federal Resources, Inc.

The Radon group of claims was located in March and April of 1953 by R. B. Daniel and A. J. Pryor, Sr., some 8 months following Charles Steen's original Big Indian Wash discovery at the MiVida mine.

The Hot Rock group of claims was located immediately west of the Radon group by W. E. McCormick and R. B. Daniel in November and December of 1953.

Both groups of claims are controlled by Federal Resources, Inc., and subject to an operating agreement with Hecla Mining Co., which serves as operator of the property. The Radon group is subject to a 15 pct. gross overriding royalty, and Hecla receives 25 pct. of the net income after royalty. The Hot Rock group is subject to an overriding royalty of 10 pct. and Hecla receives 50 pct. of the net income after royalty. Hecla received 100 pct. of the net income from each property until preproduction costs were recovered.

Total Radon and Hot Rock production through 1959 was 258,644 dry tons assaying 0.65 pct. uranium oxide ( $U_3O_8$ ).

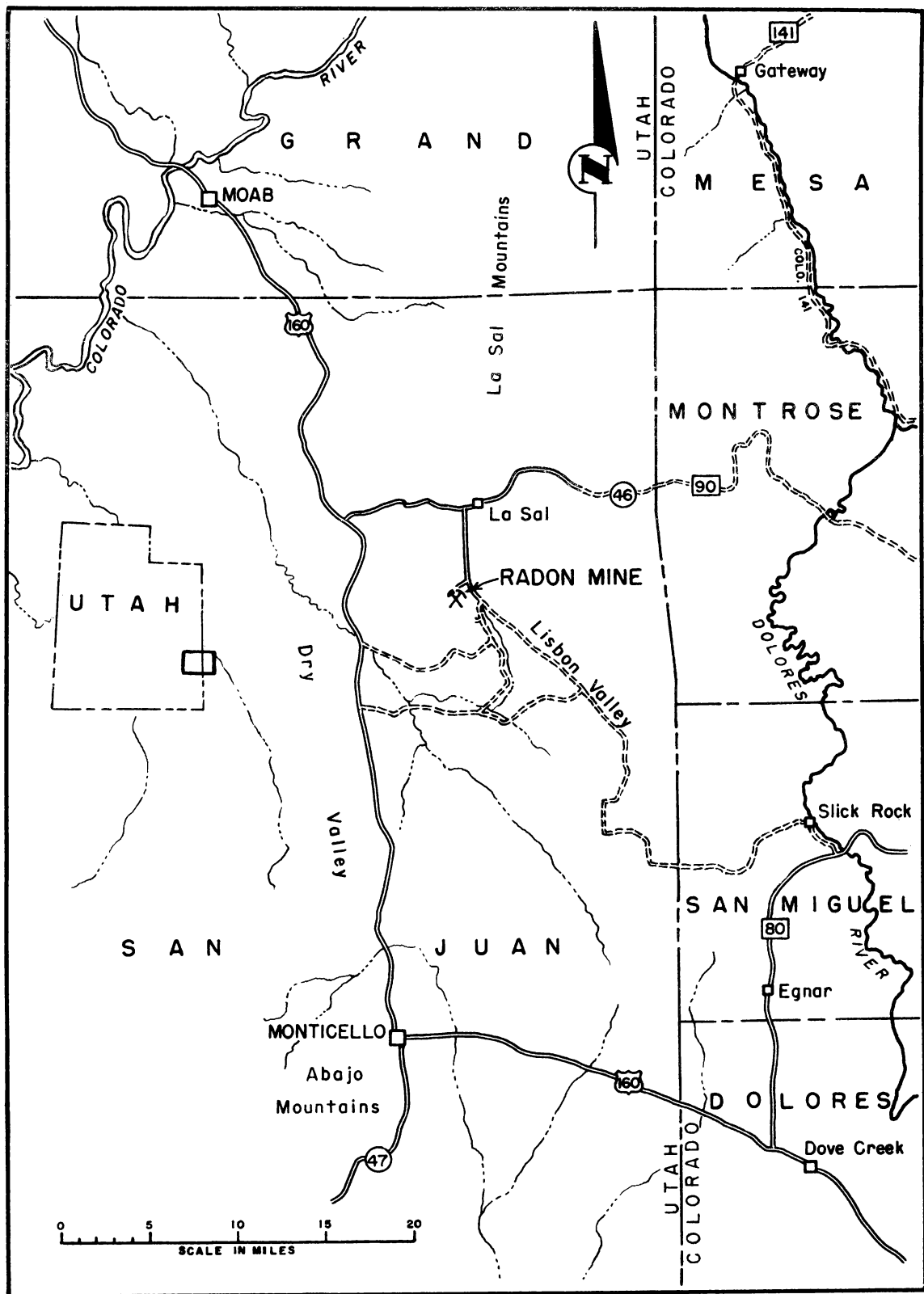


FIGURE 1. - Location Map, Radon Mine, San Juan County, Utah.



**FIGURE 2 - View Northeast From Radon Mine Plant Toward La Sal Mountains.**

#### LABOR AND LIVING CONDITIONS

Although much larger than most uranium mines in the Colorado Plateau region, with the exception of those at Ambrosia Lake, N. Mex., the Big Indian mines will be relatively short-lived operations, and expensive housing developments are not warranted. To keep their men, the Big Indian operators had to provide adequate living facilities. Most of them built trailer camps with the necessary utilities; some purchased the trailers to rent to men with families, and some built more permanent structures. Most companies built and operated boardinghouses to accommodate single men. One company constructed permanent homes at the town of La Sal, 8 mi. away, for its staff and key personnel.

Despite efforts to house the men and their families near the mines, the family men preferred to live in Moab or Monticello and to commute, despite the distance. The availability of schools in town eliminated long rides for the children or the cost of boarding them in town and was an important reason for moving to town. Even the men who lived in the boardinghouses became more

reluctant to be away from the conveniences and communion of town life. As the number who lived in the camps decreased, the cost per occupant became excessive, and most of the camps were closed down. Most of the Radon men now live in Moab; a few live in trailers at La Sal.

Hecla Mining Co. built a small trailer park at the mine for a few who wished to stay there, and in 1955 constructed a housing project in Moab (fig. 3). This project contains 15 two- and three-bedroom homes, which are occupied by key personnel.



**FIGURE 3. - Hecla Mining Co.'s Housing Project. La Sal Mountains in background.**

Many of Hecla's contract miners and most of its staff came from its Idaho operations. Because of the high wages in the district, the labor supply is good. The labor turnover at the Radon has been relatively low.

#### GENERAL GEOLOGY AND DESCRIPTION OF THE DEPOSIT (6, 3)

##### General Geology

The Radon ore body lies on the southwest flank of the Lisbon Valley anticline, an asymmetrical salt structure that strikes northwest. The Lisbon Valley fault lies along the axis of the anticline. Lisbon Valley occupies the collapsed portion of the anticline, and Big Indian Wash cuts obliquely across its southwest flank. The Lisbon Valley fault displaced the Hermosa limestone (Pennsylvanian) on the southwest upthrown side, so that it contacts the Dakota sandstone (Cretaceous) on the northeast downthrown side. Parts of the fault zone are copper mineralized.

Formations exposed in the vicinity of the Radon mine range from the Hermosa limestone (Pennsylvanian) to the Mancos shale (Cretaceous) (fig. 4). Rocks exposed along the escarpment, which forms the west wall of Big Indian Wash, and exposed westward down the flank of the anticline include the Cutler formation (Permian), Chinle formation (Triassic), Wingate sandstone (Triassic), Kayenta formation (Jurassic), and Navajo sandstone (Jurassic). The beds in the vicinity of the Radon mine strike N. 10° W. and dip from 5° to 7° SW.



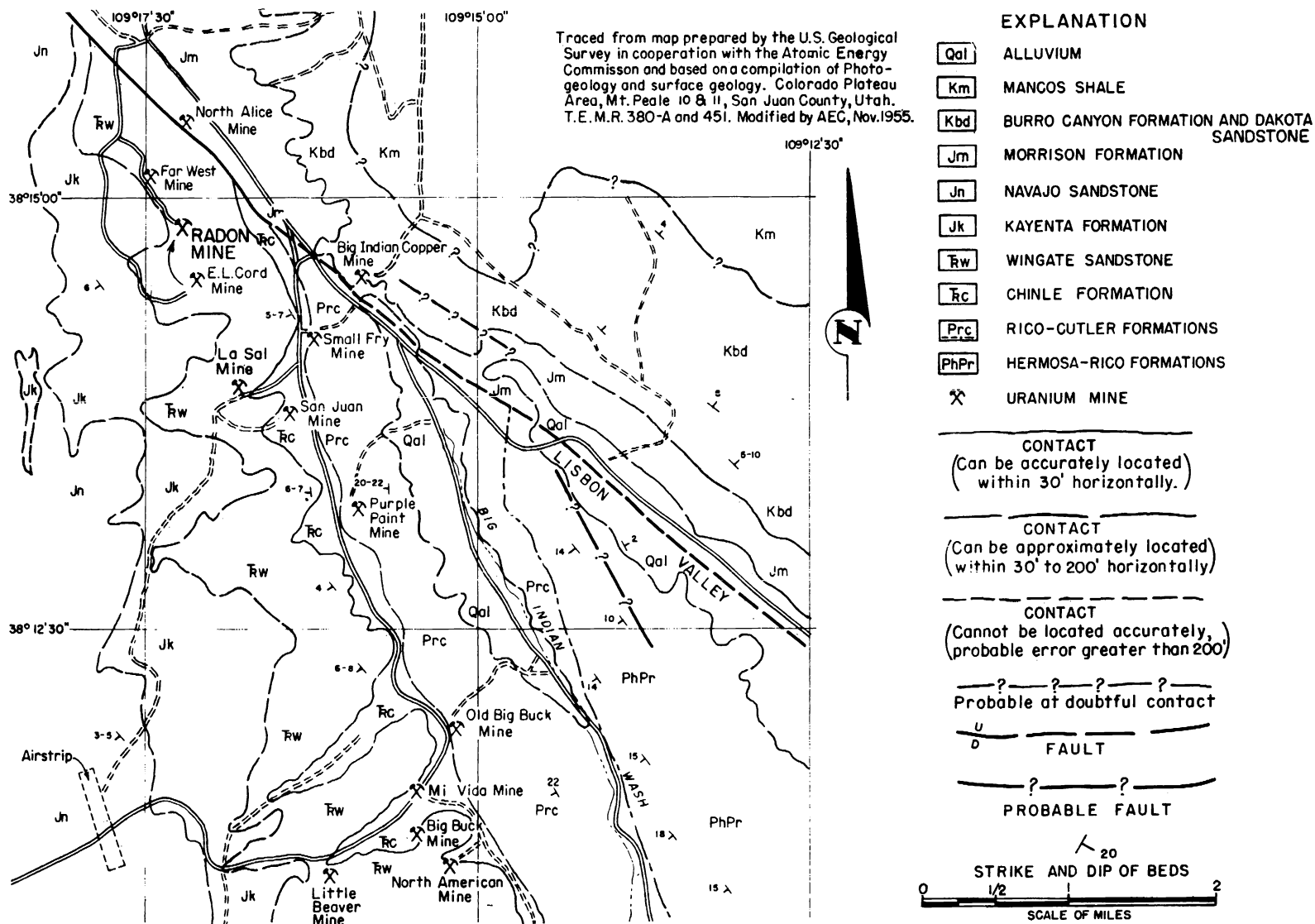


FIGURE 4. - Geologic Map of Lisbon Valley and Big Indian Wash, San Juan County, Utah.

The surface rock in the immediate vicinity of the mine camp is Wingate sandstone, a massive buff cliff-forming sandstone, which is about 350 ft. thick in the area. Rapid advance can be made when driving or sinking through the Wingate sandstone because it can be drilled rapidly, broken cleanly, and loaded easily. Beneath the Wingate sandstone lies the Chinle formation, which contains the Big Indian ore deposits. Chinle rocks are variable in color and consist predominantly of red and green mudstone and siltstone, interbedded with conglomerate and fine-grained cross-bedded sandstone. The Chinle is about 420 ft. thick in the area and can be recognized by the slopes it forms. The Cutler formation underlies the Chinle formation. Its upper rocks are medium- to coarse-grained sandstone, with discontinuous lenses of siltstone, mudstone, and coarse-grained arkosic sandstone. These are essentially red or red-brown and, locally, lavender and white. A few small ore bodies found in the arkosic lenses have been mined at the Small Fry, Purple Paint, and old Big Buck workings along the escarpment.

#### Description of the Deposit

The Radon ore body lies along the bottom of the basal or Moss Back member of the Chinle formation. The Moss Back member is composed mostly of gray or greenish-gray sandstone, interlensed with discontinuous bands of green mudstone, siltstone, and conglomerate. Usually, the ore lies within 2 ft. of the contact between the Moss Back member and the underlying Cutler formation.

Ore occurs principally within a gray sandstone stratum that is 3 to 10 ft. thick and usually contains a coarse-grained arkose (or quartzose) lens. The harder and better cemented arkose lens ranges from 2 in. to 8 ft. in thickness and has an average thickness of about 2-1/2 ft. Mudstone, siltstone, and mudstone-pebble conglomerate lenses interfinger the sandstone. Uraninite, the ore mineral, occurs as disseminated particles. The highest grade ore is almost always the arkose lens, and for the most part, the better cemented and harder the rock, the higher will be its uranium content. Thin mudstone and siltstone lenses that lie within the ore sandstone, or mudstone and siltstone lenses that enclose the sandstone, are often ore grade. Petrographic studies by the Atomic Energy Commission (AEC) have shown that the uraninite occurs in the interstices between the quartz grains, having partly or completely replaced the original calcite-clay cement. Uraninite has also invaded and partly replaced feldspars along cleavage partings. Through 1959, ore produced from the Radon mine averaged 0.65 pct.  $U_3O_8$ . The vanadium pentoxide ( $V_2O_5$ ) content was less than 0.05 pct., and the calcium carbonate ( $CaCO_3$ ) content was about 10.5 pct. The moisture content averaged 4.6 pct.

The Radon ore body is approximately 2,150 ft. long, 400 to 700 ft. wide, and lies between 550 and 750 ft. beneath the surface. The thickness of the ore has ranged from 1 to 8 ft. and has averaged 4-1/2 to 5 ft. Its trend is roughly parallel to the strike of the beds, or approximately N. 10° W., and its dip is about 7° SW. The ore body is part of a large mineralized zone that trends about N. 23° W. along that part of the southwest flank of the Lisbon Valley anticline.

## EXPLORATION

The Radon ore body was discovered by drilling begun in June 1954 by the U. & I. Uranium Co. Four holes drilled before that time failed to find ore. The drilling program covered the course of a year, and was carried on by U. & I. Uranium Co. from June through November 1954 and by Hecla Mining Co. from January to June 1955, when shaft sinking began. Sixty-three holes, which ranged from 553 to 750 ft. in depth and aggregated 39,270 ft., were drilled. The Hot Rock claim group was tested by another drilling program between December 1955 and 1956, partly financed by the Defense Minerals Exploration Administration (DMEA). It comprised 39 holes, which ranged from 554 to 1,007 ft. in depth and aggregated 27,520 ft. Total exploratory drilling on the Radon and Hot Rock claim groups, including 4 holes not drilled as part of the two main programs, amounted to 68,458 ft. drilled in 106 holes.

Exploratory holes were drilled with 4-1/2-in. three-cone roller-type bits to within 30 to 50 ft. of the contact of the Moss Back member of the Chinle formation and Cutler formation. From there the holes were core drilled, NX size, through 5 to 20 ft. of the underlying Cutler formation. The Cutler formation was recognized by maroon sandstone, siltstone, or mudstone, which contained light-gray irregular bands, often micaceous. Radioactive cores were split and assayed chemically for the U<sub>3</sub>O<sub>8</sub> content. In early ore reserve estimates, a tonnage factor of 14 cu. ft. per ton was used, but this figure has been lowered. Ore produced in 1959 had an average volume of 12.5 cu. ft. per ton in place.

Holes were drilled with a Mayhew 1000 drill rig<sup>5/</sup>, and the contractor was paid \$2.75 a foot for noncore drilling and \$4.50 a foot for core drilling. Drill roads and drill sites were provided by the company, which had them prepared by bulldozing contractors who were paid \$10 an hour. In 1959 the cost for noncore drilling was \$1.75 per foot, and the cost for core drilling was still \$4.50 per foot.

## DEVELOPMENT AND MINING

### Development

The Radon ore body is developed by a 690-foot, three-compartment vertical shaft, a tracked haulage level driven beneath the tabular ore body, and a system of roughly parallel strike drifts driven within the ore body (fig. 5). The strike drifts are connected to the shaft by two dip inclines and to the haulage level by one- and two-compartment raises. Two outlying ore bodies northwest and south of the main ore body are developed through headings driven from the shaft.

The shaft was sunk to intersect the rectangular ore body about midway along its downdip edge. This limited the haulage distances in the ore zone to a maximum of about 1,100 ft. The site chosen for the shaft collar was a gently and evenly dipping outcrop of massive sandstone large enough to accommodate a

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<sup>5/</sup> Reference to specific makes or models of equipment is made to facilitate understanding and does not imply endorsement of such brands by the Bureau of Mines.

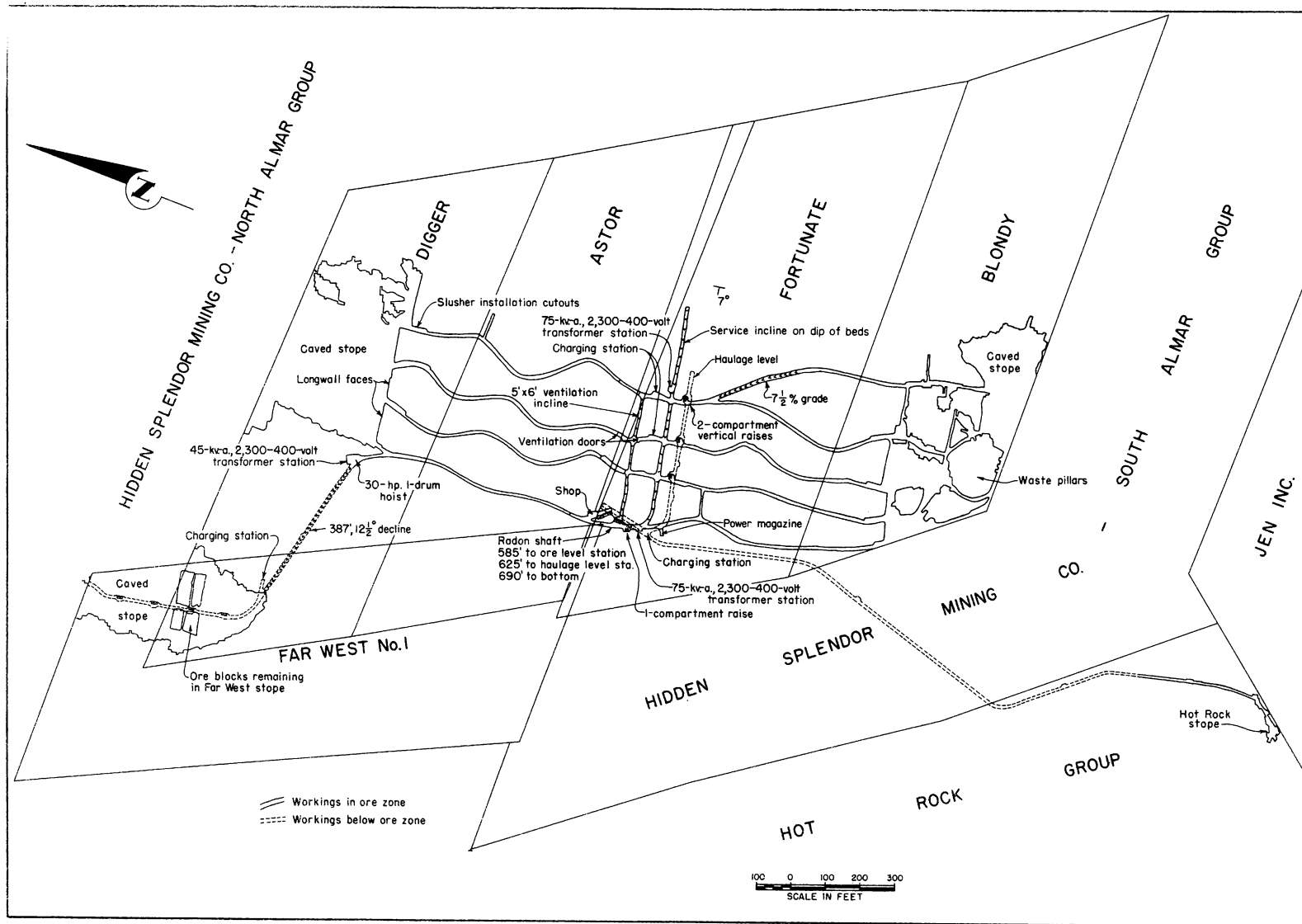


FIGURE 5. - Plan of Radon Workings.



small mine plant. Although far enough above the nearby wash to be safe from summer downpours, it had the advantage of being lower than the ground immediately surrounding it.

The shaft was collared early in June, and, while the headframe was being erected, the first 21 ft. of the shaft was sunk and timbered. A crawler-mounted crane and clamshell were used to remove the broken rock and to lower timber. On June 20, 1955, sinking was started from the 21-ft. depth by a contract shaft crew brought in from Hecla's Wallace, Idaho operations. On August 29, 1955, 65-1/2 working days after they had started, the shaftmen bottomed the shaft at 690 ft., after an average advance of 10.21 ft. a day. Contract labor worked 790 man-shifts to sink the shaft 669 ft., making an average advance of 0.85 ft. per man-shift.

The shaft was sunk with a size of about 8 by 17 ft. The ground was broken with bench rounds, and powder consumption averaged 28.8 lbs. per foot. Rounds were mucked with a Riddell mucker using a 3/8-cu.-yd. clam shell to load the 30-cu.-ft. sinking buckets. Two buckets were used, and 10.7 bucketloads were hoisted per foot of advance. The shaft was divided into three equal compartments, two hoistways and one manway, with 8- by 8-in. Douglas fir shaft sets hung on 5-ft. 3-in. centers. The compartments are 4 ft. 4 in. by 5 ft. 6 in. in the clear. The sets were lagged tight with 2- by 10-in. cedar. Twelve-foot ladders were staggered every other set. Skip guides are Philippine mahogany, 4 by 6 in., 21 ft. long. Four pairs of 7/8-in. hanger rods were used per set of timber. A 6-in. compressed air line, 2-in. waterline, 2-in. shaft-sprinkler line, 30-in. metal ventilation duct, 2,300-v. lead-covered power cable, and telephone and hoist-signal cables are installed in the manway. The 2-in. fire-sprinkler line is hung along the dividers with Spraying Systems Co. fog nozzles protruding through the lagging between the manway and the hoistways. The nozzles are spaced every other set, alternately pointing down and across the shaft. The sprinkler line is hung across the top of both shaft stations. In case of fire, a valve at the rear corner of the warehouse is opened. Shaft blocking has held well and no major repairs have been necessary to date (1960). The shaft is dry.

Two stations were cut from the shaft, one on the ore level and the other farther down in waste. The upper station is at 585 ft.--the level of the lowest strike drift. At this elevation, the shaft passes through the southwesterly dipping ore layer just inside its downdip edge; the contact of the Moss Back member of the Chinle formation and the Cutler formation crosses the station. North of the shaft station, the downdip edge of the ore swings sharply updip and the underground shop, which was excavated 50 ft. to the north on the same side of the drift, is in waste.

From a point about 90 ft. south of the upper shaft station, a 7- by 8-ft. service incline was driven 640 ft. updip on a bearing of N. 80° E. It approximately bisects the ore body and extends about 40 ft. in waste past the updip ore boundary. This incline is the main entry to the stopes. It is tracked with 30-lb. rail on 24-in. gage. A single-drum 7-1/2-hp. air tugger hoist, anchored at the upper end of the incline, is used to hoist the supply and tool

cars to the levels; C. S. Card ball-bearing turntables are set at each level intersection. A parallel 5- by 6-ft. ventilation incline, driven updip opposite the manway compartment of the shaft, also connects the shaft with the strike drifts.

Four drifts driven along the strike in both directions from the service incline to the claim boundaries divide the main Radon ore body lengthwise into irregular strips. Measured along the dip, the drifts are an average of 120 ft. apart; measured vertically, they are 20 ft. apart. South of the service incline, the upper strike drift forks into two headings to develop a wider part of the ore body. The upper split was driven on a 7.5-pct. grade for 250 ft. to gain additional elevation at this end. A 3-ton Mancha locomotive can pull this grade when returning one empty car to the face. The strike drifts are 8 ft. wide and 7 ft. high, and were driven so that the contact of the Moss Back member of the Chinle formation and the Cutler formation was kept about 1 ft. above track in the updip wall, if possible. Where the ore was more than drift height, the drifts were advanced along the top of the ore. Thus, the roof bolts could be placed against waste rather than against ore, and an even back was formed later when the longwall faces crossed the drifts. The benches that are formed at the junctions of the drifts and their updip longwall faces provide storage room for the ore moved downdip from the faces. More emphasis was placed on keeping the contact in sight than on keeping the drifts on a fixed grade. Because of local irregularities along the top of the Cutler formation, the miners drove the headings on a somewhat winding course to keep the ore exposed and the openings near level.

Forty ft. below the ore-level station, or 625 ft. beneath the shaft collar, a haulage-level station was cut in the red Cutler formation. The lower station is simply a 12-ft.-wide by 7-ft.-high drift along the wall-plate side of the shaft, terminating 50 ft. to the north. On the shaft side of the drift, a flat-bottomed trench, about 5 ft. wide and 10 ft. deep, was cut out for a distance of 50 ft. on each side of the shaft. This is the slusher-loading pocket into which the mine cars are dumped and from which the skip measuring pockets are loaded.

On the south end of the slusher pocket a 7- by 8-ft. tracked haulage drift was driven N. 80° E., (the same bearing as the dip of the beds) under and at right angles to the four strike drifts. Vertical raises (the longest is 90 ft. high) connect the haulage level to each strike drift. Two 40-ft.-high one-compartment raw ore passes were raised near each end of the slusher trench directly to the lowest strike drift; two-compartment raises, manways cribbed off from the untimbered ore passes, connect the haulage level with the other three strike drifts. The manways are 5 by 6 ft. in section outside the 4- by 8-in. cribbing. The ore passes were about 6 by 7 ft. in section when first opened, but have become enlarged an estimated 30 pct. by the abrasion of the ore against the walls. Three-inch spaces between the cribbing rings permit miners to bar down hangups from the manways, but few hangups have occurred because the ore is dry and runs well.

Two small outlying ore bodies, which are extensions of other large ore bodies that extend into ground of Federal Resources, Inc., are developed

through the Radon shaft. Both lie downdip from the main Radon deposit, one on either side of the shaft. The ore body to the north is developed from the upper shaft station; the one to the south is developed from the lower. To reach the north ore, in January 1957, a heading was turned westward from the lowest strike drift about 700 ft. north of the shaft. After the turn was made, a small hoisting station was cut and a 387-ft., untimbered, tracked, 22-pct. incline (or possibly better termed a "decline") driven through the beds on a bearing of N. 76° W. It was bottomed in the red Cutler formation 20 ft. beneath the ore. This ore is an extension of Hidden Splendor Mining Co.'s Far West or North Almar ore body and lies within the boundaries of the Far West No. 1 claim of Federal Resources, Inc. From the bottom of the incline a 500-ft. haulage drift was driven northward beneath the long axis of the ore. Four two-compartment timbered raises connect the drift with the ore zone. The raises were spaced on 100-ft. centers so that single-pass slushing could be used to move the ore from any place on the upper level to a raise. The longest slusher haul was 180 ft. Mining in the Far West began in September 1957.

The other outlying ore body owned by Federal Resources, Inc., lies within the adjoining Hot Rock claim group. It is an extension of the Cord ore body of Jen, Inc. and also is possibly an extension of the South Almar ore body of Hidden Splendor Mining Co. From the lower Radon shaft station, Hecla Mining Co. drove a 1,950-ft. development drift southward to the Cord ore body property line. The first 1,150 ft. was driven on a 1/2-pct. grade to the Hot Rock claim group property line. The grade was then sharply increased until the Chinle contact was cut. A right-of-way easement was obtained from Hidden Splendor Mining Co. to drive the haulage drift across its South Almar claims.

All developemnt work was completed under incentive contracts. In a standard 8- by 7-ft. development heading, a crew received \$10 per lineal foot of advance, which included payment for laying track and hanging pipe. Six men, three on a shift, made up a contract crew on a strike drift level. Each crew drove its drift in both directions from the dip incline, and on each shift two men were assigned to drill and blast at one face while the third man mucked out and installed pipe and track in the other.

Two small truck-mounted jumbos in addition to leg-mounted drills were used in developemnt work. The jumbos were built by mounting two Cleveland 6-ft. air-motor-powered booms and one hydraulic anchoring stinger on a truck. The booms were equipped with 4-ft. aluminum shells with Ingersoll-Rand DB-50 drifters on them. The leg-mounted drills were used from bull-hose connections on timber trucks. Drill steel was 7/8-in. hexagonal, and the tungsten carbide cross bits were 1-1/2 in. in diameter. Pyramid- and bottom-draw-cuts were drilled with the jumbo drills; burn cuts were usually drilled with the leg-mounted drills, but in soft and springy mudstone the bottom 12 in. or so of a burn cut sometimes froze and failed to be cleared by the blast. Chisel bits were not used in mudstone as they tended to bind and stick. Even with cross bits, frequent blowing and clearing of a hole was necessary to keep the mud from building up behind the bit and sticking the steel.

Except at the shaft stations and chutes, no timber has been used for back support in the development workings. Roof bolts were used extensively. Power

is distributed underground from two 75-kv.-a. 2,300-400-v. centrally located transformers near the upper and lower ends of the service incline and a 45-kv.-a. 2,300-440-v. unit near the hoist station at the top of the Far West incline.

### Mining

The tabular, gently dipping Radon ore body is being stoped by a longwall system patterned after the coal mining methods used in many European mines. The longwall faces retreat from the north and south property lines parallel to and toward the service incline. The ore is broken out in 4-ft. slices, and the supported back--a strip that ranges from 7 to 12 ft. in width--is held by two rows of steel props and an intermediate row of cribs. The back is allowed to cave as the supports are moved forward; the cave is fully controlled. Scrapers pull the ore down dip to the nearest drift, where it is loaded and transported by conventional track equipment. No underground water has been encountered. Retreating, followed by full caving, was chosen to gain complete and selective extraction of the high-grade ore body.

When planning a stoping method, a system which required a minimum of openings was attractive. Room-and-pillar mining would not yield the desired recovery primarily because of the weak rock that overlies much of the ore. Weak backs would have increased the problem of ore dilution, and probably would have resulted in the loss of ore in unrecovered pillars. In longwall mining, a weak back is an advantage because the back must fall soon after the supports are withdrawn, provided that the back is not so soft as to permit the props to penetrate it. A strong back, or one which breaks abruptly, is usually difficult to control along a uniform break line (8, p. 506). Because of the thinness of the ore body, the operator chose small conventional production equipment that would not require overbreaking to gain operating height.

In order to mine clean ore and to accomplish complete extraction, some sacrifice was expected in the number of tons produced per man-shift. However, if costs per pound of  $U_3O_8$  produced were considered, lower final cost should result. Furthermore, there is an advantage in shipping a high-grade product and thereby obtaining premiums offered in the purchasing schedule of the AEC.

The focal point of Radon stoping is the back-support method. All phases of the stoping operation are keyed, at least in part, to the capabilities of the Becorit yieldable steel props and supplemental cribs. Much of the published information concerning the current practice of steel-prop support in European coal mines was consulted in planning the stopes. Only light charges, if any, are used to break the much softer coal, whereas heavy charges are necessary to break the ore. At the Radon mine, the props have a secondary function, that of supporting blasting boards that confine the ore.

The north and south claim boundaries of the Radon ore body lie  $10^{\circ}$ - $14^{\circ}$  obliquely off the bearing of the dip of the beds. Therefore, at both ends of the property the first longwall cuts taken from the ore were wedge shaped so that the faces would lie on the dip. Mining the longwalls on the dip has two advantages: The longwall scrapers travel better between the face and the blasting boards; final recovery at the service incline will be easier.



### Description of the Becorit Mine Prop

The German-made Becorit Mine prop is one of many that have been developed for ground support in European coal mines. The first steel mine props introduced in Germany were two-piece and extensible, and appeared in 1901-02 (11, p. 154). Various types of yielding designs evolved, and by 1928, yielding props were used extensively in the coal mines in Germany (5, p. 280). The construction and behavior of these early props fitted the full-stowing or back-filling methods of longwall mining used in Germany at that time. They were built to have an initial carrying load from 2 to 3 tons, which gradually increased as the props yielded with the general subsidence of the back onto the waste packs or stowage. A change in the view on the function of props began when caving with partial packing was introduced, a method that was patterned after the longwall caving operations developed in England. In the newer method, the back was held with as little yield as possible instead of lowering it slowly onto the stowed waste. Stronger props were needed to hold the back until it broke off along a uniform line behind the last row of props. Props were developed that had higher initial setting loads, had higher maximum carrying loads, and reached their maximum carrying loads after very little yield of the telescoping components. At first, heavier and more rigid props were built along the same lines as the older, weaker props. Gradually, newer designs were built that had better yielding properties. A positive locking mechanism was needed that not only developed the maximum carrying load quickly, but afterward would yield only when the roof load threatened to deform the prop. Spruth reported (11, p. 25) that, despite constant improvement and stronger construction, not until 1945 were designs improved to a point where the maximum prop loads were reached after very little subsidence of the back.

The first model of the Becorit mine prop was developed in 1940-41 by the Becorit Mine Support Co., Ltd., Recklinghausen, Germany (11, p. 169). The Becorit prop is made of three main parts: an upper shaft, a lower shaft, and a lock assembly (see fig. 6). The upper shaft is made of two strong E-sections welded together to form a column that is about 2-7/8 in. square. The middle bars on the E's butt to provide internal bracing to resist the pressure of the lock assembly. The upper shaft slides through the lock assembly, which is at the top and integral with the lower shaft, and telescopes into the hollow lower shaft, which is built of two 4-in. channels welded at their flanges. The four prongs on the top of the upper shaft were designed to hold a steel roof bar, but at the Radon mine they are used to hold a wood friction block. The base of the lower shaft, which rests on the ground, is convex. The lock assembly is a frictional device, which progressively tightens while the upper shaft moves downward 3/8-in. The principle feature of the lock is its rotating wedge, which when hammered tight into the assembly locks the two shafts together, and when hammered loose releases them. As the roof load increases, the additional wedging reaction increases the prop's resistance to the back load until the prop's maximum carrying load is reached.

The lock assembly is held together at the top of the lower shaft by two spring-steel clamping bands. The four-piece assembly consists of the backup

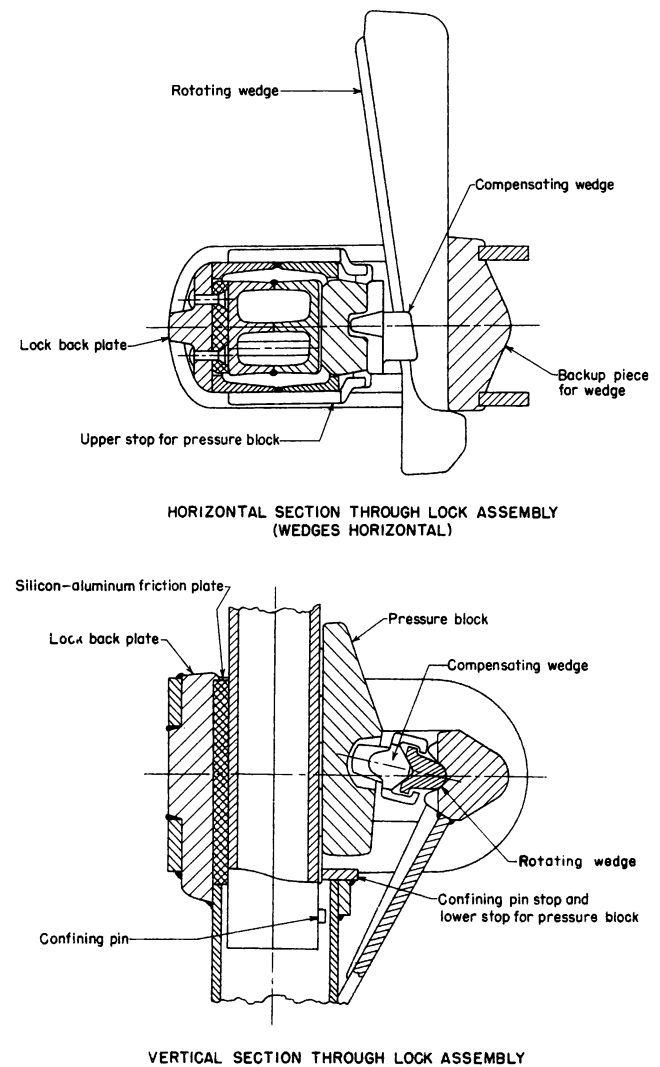
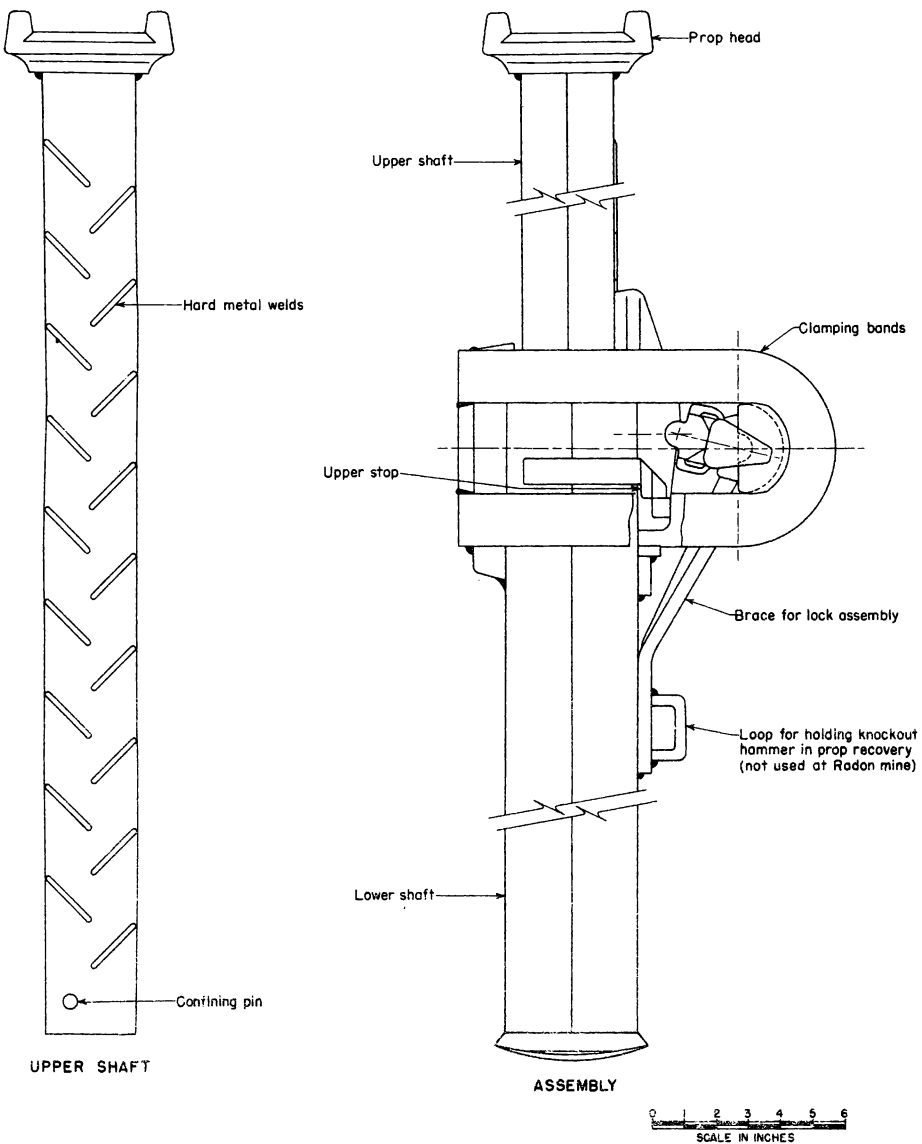


FIGURE 6. - Becorit Prop.

piece, which is held within the curvature of the clamping bands, and which is itself a seat for the rotating wedge; the rotating wedge, which pivots within the backup piece; the compensating wedge, which is the link between the rotating wedge and the pressure block; and the movable pressure block, which fits flat against one side of the upper shaft. Fastened to the other side of the lower shaft to oppose the pressure of the pressure block is a silicon-aluminum friction plate. The friction plate is analogous to the stationary jaw of a clamp while the pressure block acts as the adjustable jaw. When the prop is being set, the rotating and compensating wedges are tilted upward at their maximum angle of approximately  $12^\circ$  with the horizontal so that the pressure block is at its highest position. When the prop is jacked to the setting load, the wedge is driven tightly into the lock with a hammer. Thin hard-metal welds on the pressure-block side of the upper shaft cut into the softer steel of the block and insures a gripping action. When the roof load exceeds the setting load, the pressure block moves downward with the upper shaft. The vertical movement of the block imparts a twisting motion to the rotating wedge because the rotating wedge can move only by pivoting downward about its axis in the seat of the backup piece. The downward travel of the block ceases when the block contacts its stop; the rotating and compensating wedges are then approximately  $1^\circ$  below the horizontal. As the friction shoe moves downward, the wedges act as if their combined width was being increased and thereby place increasing stress on the spring-steel clamping bands. The bands are welded to the lock back plate and not to the sides of the prop, and their full elasticity is preserved. The full carrying load of the prop, rated at 40 to 50 tons, is reached when the upper shaft has traveled downward  $3/8$ -in. As the weight of the roof surpasses the maximum carrying load, the prop yields although it is still exerting maximum resistance, and the upper shaft planes some metal from the pressure block. The upper shaft has a slight taper, about 1:300, throughout its length so that a continuous yielding resistance to the maximum carrying load is achieved.

The extended height of the Becorit prop used at the Radon mine is 1.8 m. or about 71 in. The collapsed height of the Becorit prop is 42 in. Without an extension, a prop weighs 153 lb. Extra heavy 4-in. pipe, which has a 0.337-in. wall thickness and a weight of 15 lb. per foot, has been welded in 1-, 2-, and 3-ft. lengths to the bottom of the props to reach higher backs. The 3-ft. extensions have been found to be too long for good support. In 1956-57, 1,400 Becorit props were purchased at a delivered cost of about \$70 each. To minimize prop loss, a number is welded on each.

### Drilling

Leg-mounted drills are used exclusively in the stopes (fig. 7). The 45-lb. drills are equipped with 4-ft. telescopic legs, which are advantageous in stopes that are confined by low backs, nearby caves, and closely spaced roof-support equipment. The telescopic legs permit using one steel length, and the stingers on the short legs do not have to be placed out under potentially dangerous ground to drill bottom holes. All production drills have  $4\frac{1}{4}$ -in. chucks. Drill steel is  $7/8$  in. hexagonal,  $4\frac{1}{2}$  ft. long. Many types of one-use and multiuse drill bits have been tried. In January 1960, the company was



**FIGURE 7. - Drilling Wet at Longwall Face.**

experimenting with two makes of socketed one-use tungsten carbide insert cross bits. Both had 1-5/8-in.-gage, two side waterholes, and a greater than average skirt clearance behind the wings. In general, the bits most favored at the Radon mine are those with a similar design. For fast drilling in the fairly soft rock, bits that have two or four side waterholes and ample skirt clearance for easy passage of the cuttings are desired. Mudstone will often plug center waterholes, and, when bits are received that have them, they are plugged with hard-facing Stooddy 21 alloy rod. Most steel has broken at the bit end, whether the bits are threaded or socketed.

Water from the drill machines is troublesome at the longwall faces because it turns the soft mudstone and friable sandstone underlying the ore into mud. Together with the drill cuttings, the bottomed is churned into a slippery mire by the normal traffic of the miners. It takes an average of 2 to 3-1/2 man-shifts to drill out a 200- to 350-hole longwall round, and although most of the water drains from the face, some water stands where the dip flattens locally or is held from draining off by the layer of saturated mud. Ditching keeps the water moving away from the prop line, but for the most part it has been difficult to keep the ditches open in the confined working areas. Contract miners

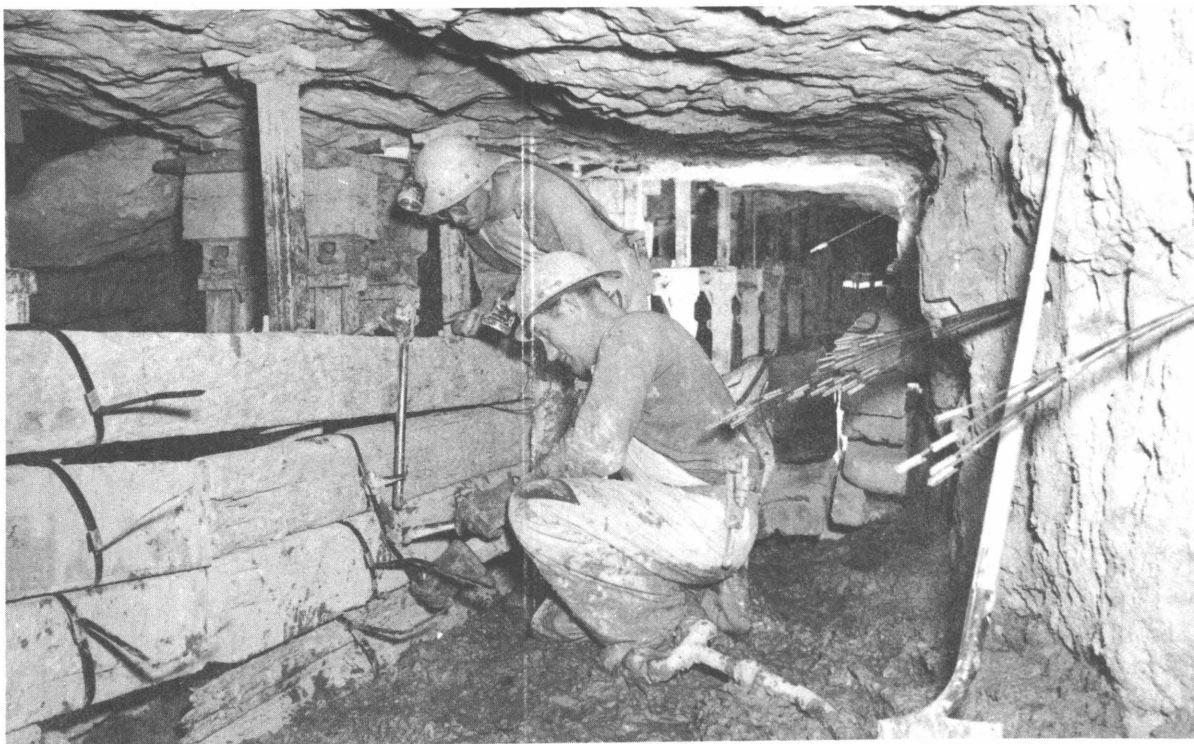
are naturally more concerned with completing their rounds than with attending to the ditches. Trouble could result should the ground beneath the inner row of props be softened enough to relieve their compressive loads so that some might be knocked down by the blasts that follow.

For 2 yr. the company experimented with dry-drilling machines to eliminate the problem caused by wet drilling. During 1959, a Cleveland Dry Drill was tried with Cleveland 1-1/2-in.-gauge, tapered-socket, one-use, Dust Collector type bits and 1-in. hexagonal steel. The steel had a 3/8-in. hole and the bits had four side holes and one center hole for removing chips. The bits cost \$0.35 each when purchased in lots of 350. The Cleveland Dry Drill used at the mine was reported to be the first of its type to be built for sale west of the Mississippi River. Soft, damp mudstone encountered in the ore stratum was the bane of dry drilling. The soft mudstone chips moved rather sluggishly through the steel and collecting apparatus, and tended to clog and lower the carrying capacity of the air. Dry drilling did improve back support and reduced the slipping hazard. Also, the hauling costs of water was reduced, water hoses were eliminated, and a cleaner hole eliminated hole blowing. Compare the stope bottoms in figures 7 and 8 where the holes were drilled wet to the bottom shown in figure 9 where the holes were drilled dry. However, dry drilling was abandoned because of the difficulty in wetting the dust during slushing. Excessive dust accumulated at the grizzlies.

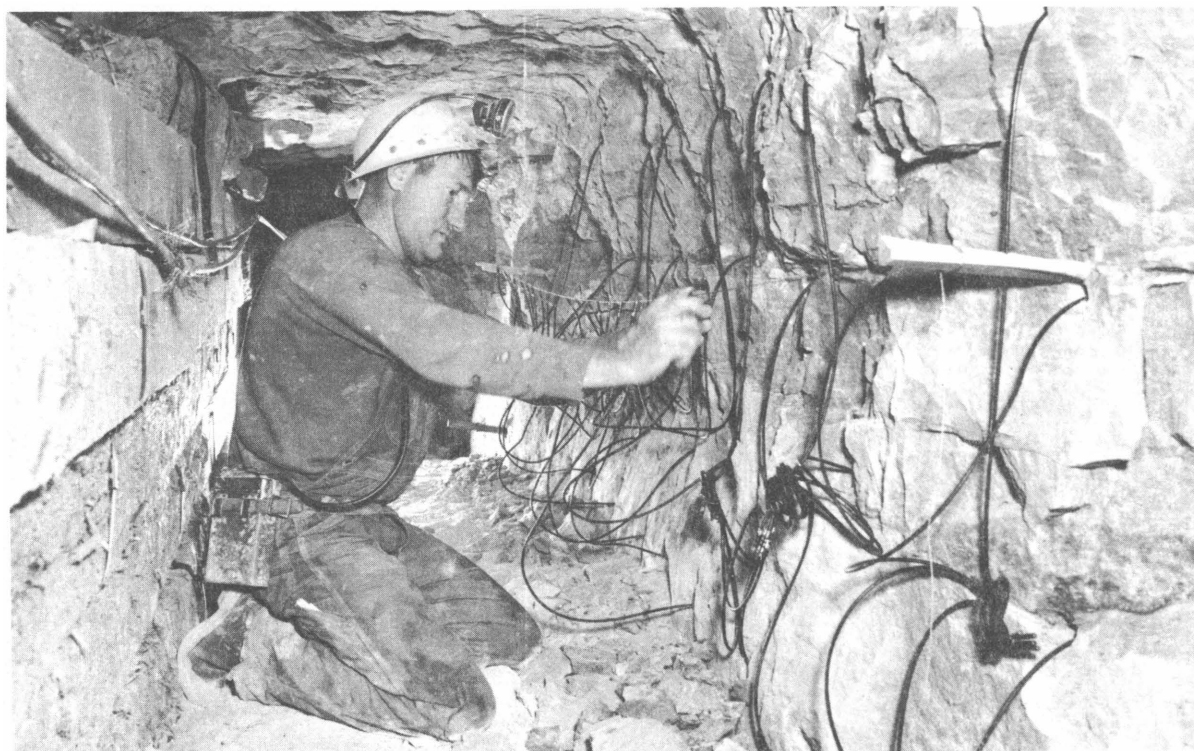
Longwall blast holes are drilled 4 ft. deep in vertical rows of 4 holes each. The rows are spaced about 18 in. apart. Vertically, the two center holes in each row are staggered according to the height of the ore; however, at least one hole is placed in the harder arkose lens for good fragmentation. All holes are drilled as nearly normal to the plane of the face as possible. Particular effort is made to keep the top and bottom holes flat to break a smooth back and bottom. This helps when the next row of props are set, minimizes dilution, and facilitates the installation of blasting boards. Sometimes the holes necessarily have to be angled where the ore thickens or where its elevation changes abruptly. Drillers are reminded to keep a piece of drill steel or tamping stick protruding from a nearby hole to help align their machines. As frequently happens, the props or cribs obstruct the driller's machine, and deviations from the ideal are expedient. The optimum depth for a longwall round at the Radon mine is 4 ft.; this depth fits the support pattern of the props. The maximum number of 4-ft. holes drilled by one man in an 8-hr. shift was 168; two partners drilled a record 331 4-ft. holes in one 8-hr. shift.

### Lagging

After a longwall round has been drilled, 16-ft. lengths of 2- by 8-in. oak lagging are strapped on the face side of the inner row of props to confine the ore. Before oak was tried, 4- by 8-in. Douglas fir was used. One 16-ft. length of Douglas fir lagging normally survived about 10 blasts. Broken pieces are used to block irregular openings along the back. In 1959, Douglas fir lagging cost \$118 per thousand board feet and averaged \$0.13 per dry ton of ore hoisted. Oak has been found to be cheaper than fir, even though its initial cost is \$155 per thousand board feet.



**FIGURE 8. - Strapping Lagging to Mine Props.**



**FIGURE 9. - Pinching Connectors Onto Igniter Cord.**

The lagging is strapped to the props with heavy-duty steel packaging bands, one placed at each end of each plank (fig. 8). The bands are  $\frac{3}{4}$  in. wide, 0.035 in. thick and have an approximate tensile strength of 2,350 lb. Signode Steel strapping in 4-ft. lengths is shipped to the mine in 500-lb. boxes that contain 1,500 straps each. Endwise, the planks are butted approximately midway between the props, and staggered so that not too many butt between the same props. The miners first wrap the bands loosely around the timbers with one man shoving the ends of the straps between the lagging from the opposite side. The bands are pulled tight with a Signode model PH-1 CB869 stretcher and clinched at their attached cleats with a Signode SYC 3435 sealer.

In 1959, the cost of the bands averaged 2.01 ct. per foot, or 8.04 ct. for a 4-ft. length. The Shoc Seal cleats, which are fixed to one end to hold the other, cost 0.75 ct. each. Adding 0.50 ct. for freight to the cost of each band, plus the cost of labor to affix its cleat, the total cost for each strap in 1959, was very close to \$0.10. The general surface man is charged with the job of making up the straps in the shop. They are taken underground in sheaves of 25 for easy count and easy handling. On the average, three sheaves are necessary to lag-off a 100- to 110-ft. stretch of longwall face.

Strapping is a quick and sure method of fastening wood to steel. For a cost comparison with nails--if the props were timber and 60-d. nails could be used--strapping would cost approximately twice as much as nailing, using two nails in place of each strap. Moreover, the tools for strapping are more expensive. On the other hand, the straps have decided advantages over nails. They clamp the width of the lagging as a unit securely to the prop, which reduces splitting from jolts by the blasts; they can be quickly cut to free the lagging easily, thereby eliminating pounding and prying that also breaks timber; and they eliminate the hazard of protruding nails--all of which offset the disadvantage of a higher cost.

### Blasting

The company uses 1-1/8-in.-diameter Trojan Trojamite C dynamite, detonated by No. 6 caps, safety fuse, and Thermalite igniter cord. Primers, placed at the bottoms of the holes, are made of 8-in. cartridges, and 16-in. cartridges are used to complete the charge. A 50-lb. case of Trojamite C dynamite has an average cartridge count of 190 sticks of 1-1/8- by 8-in. powder; in 1959 it cost in 15-ton lots \$18.30 per hundred pounds, or about 4.8 ct. a stick. The igniter cord is strung horizontally along the face to be blasted, suspended from two pegs wedged in the outer holes on each side, much like a bus wire for electric-cap circuits (fig. 9). The type A Thermalite igniter cord burns at a speed of 8-10 sec. per foot, and cost \$13.29 per 1,000 ft. in 1959. The connectors, which cost \$20.75 per thousand, are pinched along the igniter cord at 3- to 4-in. intervals. This spacing staggers the blasts about 3 sec. apart. Fuse trimming is unnecessary. The 3- to 4-in. spacing may be insufficient in a few cases to offset a slight but normal variation in the burning time of the fuse, but hole rotation is not so critical as to require greater spacing. The connectors are fastened so that the hole next to the bottom is shot first, the bottom hole second, followed by the third and fourth holes from the bottom in

that order. Shooting the upper two holes down reduces the amount of ore thrown against the lagging. The vertical rows are slabbed one at a time progressing updip. The ground could be broken with fewer but more heavily charged holes; but with more holes and lighter charges the back and the props are disturbed less.

Blasting begins at the strike drift on the downdip end of a section of a longwall face. In the first round 10 to 20 rows, or 40 to 80 holes are shot, whereas 6 to 10 rows, or 24 to 40 holes are shot in succeeding rounds. More ground can be broken by the first round because the ore is heaved directly into the drift. Updip, there is less capacity at the face, and slushing would be hindered if more holes were shot. The contractors blast at any time during the shift, but station themselves at points of access to the stoping area before each blast. One two-man crew has blasted as many as 10 times in 8 hr. in one longwall stope.

The cost to detonate a round with igniter cord is approximately 30 pct. more than the cost of caps and safety fuse if the latter alone were used. In 1959 a 7-ft. fuse complete with cap and connector, a prorated length of igniter cord, plus the labor to make up the fuse cost 14.64 ct. The surety of ignition and firing sequence, plus safety at the low, muddy, and often slippery faces are advantages of igniter cord. In 1959, dynamite consumption averaged 1.25 lb. per hole, or 2.6 lb. per dry ton of ore and waste at a cost of \$0.48 per ton. Tons broken per hole averaged 0.48.

### Scraping

Scrapers convey the ore broken from the longwall faces downdip to the nearest drifts. During the blasting and scraping and the cleanup that follows, speed is stressed so that props and cribs can be moved up quickly to reduce the supported span from 12 to 8 ft. Two-drum, 20-hp., 440-v. electric remote-controlled hoists pull 54-in. 1,000-lb. Cate scrapers. The hoists are anchored with rock bolts on 4-ft.-wide benches cut out on the updip wall of the drifts out of the way of the track equipment, 1 to 2 ft. above track level. Blocks hung where the longwall faces intersect the drifts turn the scraper ropes updip along the faces. When the retreat has progressed to within 15 ft. of a hoist installation, another 75- to 100-ft. extension of the bench is broken out. The hoist is then moved to about 8 ft. from the end of the new cut. Four 4-ft. vertical bolts through the base of the hoist anchor it; a fifth bolt placed in the wall is used as a tieback. The contractors receive \$3 for each anchor bolt installed. The bottom of the drift is from 2 to 3 ft. below the bottom of the longwall face, an offset that provides storage on the track for the ore pulled downdip. The scrapers have Joy air-electric remote-controlled clutches. The enclosures for the magnetic starter, the on-off-station relay, the low voltage transformer, and the solenoid valves that control the clutch cylinders are mounted on a 5-ft. length of 2-in. lagging that stands out of the way behind the hoist (fig. 10). This control unit can be easily disconnected at its quick connectors when the hoist is moved or the control has to be repaired. Separate from the hoist, it is free from vibrations and out of the way of the scraper ropes.



To pull the scrapers 1/2-in., 6 by 19, wire-center, Seale construction wire rope is used. Angle blocks in the backs of the drifts are hung on roof bolts; tail blocks are hung from flexible eyebolts. The Homestake eyebolts are made of 5/8-in., 6 by 19, right-lay Seale rope, lead socketed to a thin-walled steel truncated conical shell. For this use, lead socketing has proved satisfactory. Their overall length is 20-3/4 in. (fig. 11). They are anchored in vertical pinholes drilled in the back of the next drift updip. In very low stopes the eyebolts are anchored in regular blastholes beyond the rows to be shot. Using the flexible eyebolts, the pull on the return cable is partly absorbed by the rock at the collar of the hole. With less chance of the wedge working loose and the eyebolt pulling out, the flexible eyebolts have a special advantage in soft ground that often overlies the ore. The 54-in. scrapers travel 250 to 300 ft. a minute through the 7- to 9-ft.-wide aisle between the new face and the lagging.

The scraper operator usually stands near the lagging where it crosses the strike drift. From there he can direct the scraper through his low-voltage remote-control push-button stations. As the scraping distance increases and the operator is unable to see the muckpile clearly with his cap lamp, he is aided by lamp signals from his partner who stations himself on the other side of the tail sheave alongside the lagging.

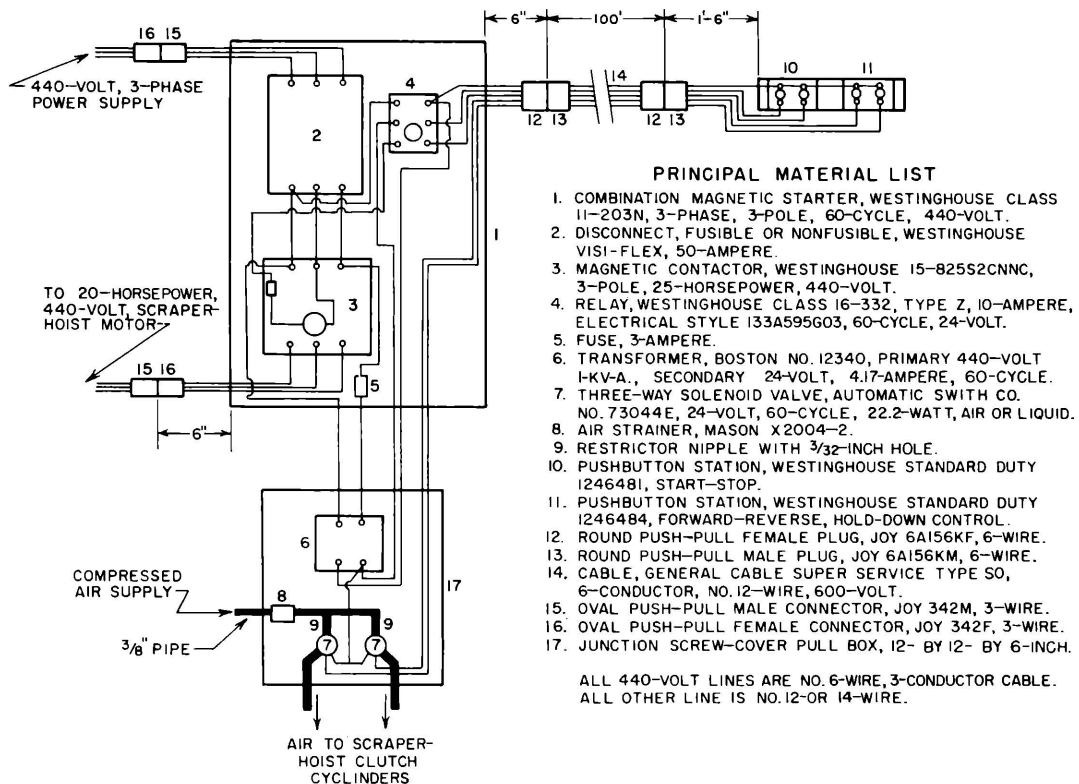
In the Far West stope, 20-hp. three-drum slusher hoists are used to convey the ore directly to the raises. There, the three-drum hoists are advantageous, because a number of faces are cleaned from one scraper setup.

#### Removing Lagging and Cleaning Up

To remove the lagging, the steel straps are cut with an axe. At first, only parts of the lagging are removed to permit shoveling of any ore lying behind the lagging into the path of the scraper at the face. The rest of the lagging is taken down after the scraper has made its final pass. The tail sheave of the scraper is left hanging in its final position with the rope in place, and, where there is not enough room to stack the lagging along the face, some is pulled with the scraper hoist to the closest drift.

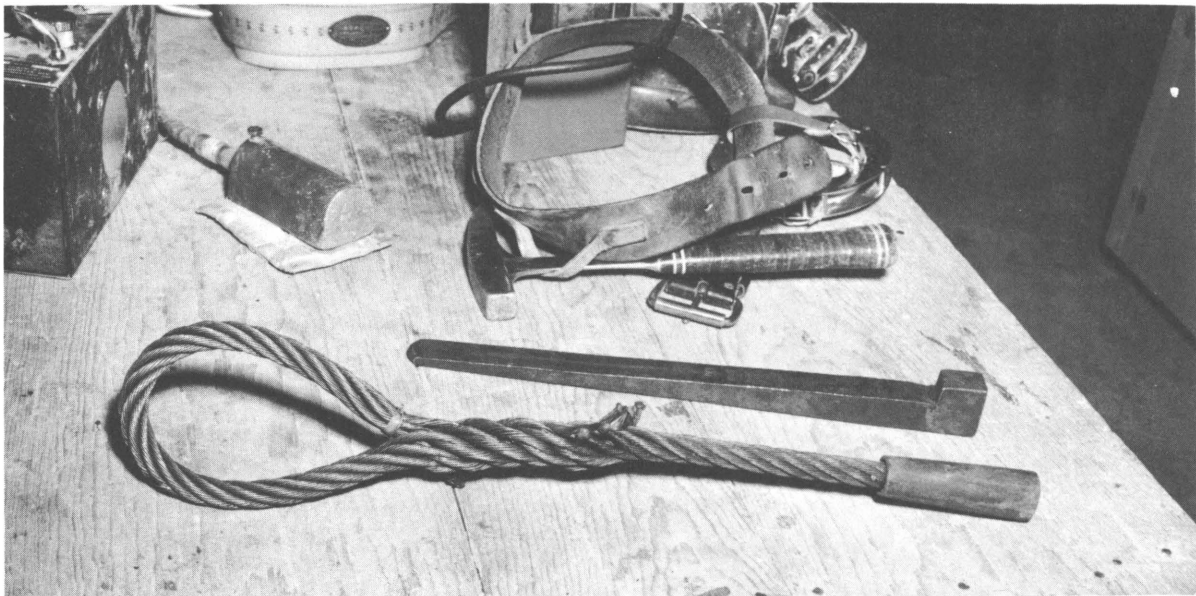
#### Moving Props and Cribs

Before any props are moved from the back row, a line is established 4 ft. ahead of the front row by stretching a string between two tubular props temporarily set at both ends of the longwall. By alining props carefully, lagging is made easier and better roof control is maintained; a straight break line helps to equalize prop load. Still keeping the back row intact, a few extra props are first set along the new line beginning at the updip end. Then, one by one and working downdip, the back props are shifted forward to the new line (fig. 12). Set on 3-1/2- or 4-ft. centers according to ground conditions, the props are alined opposite those in the second row. The shift is diagonal because each prop, when moved to the new row, occupies a position farther downdip than it did in the old row. There is a temporary overlap of three props opposite each other at the point of transfer.



NOT DRAWN TO SCALE

**FIGURE 10. - Diagrammatic Sketch of Remote-Control System Used at the Radon Mine With Joy A2F-211 Scrapers.**



**FIGURE 11. - Flexible Eyebolt and Wedge for Scraper Tail Sheave.**



**FIGURE 12. - Setting Prop Along New Line.**

Prop pulling is a two-man job. Each prop is collapsed by hammering its locking wedge loose. Occasionally the wedge must be struck a number of blows to loosen it; an 8-lb. hammer is the best weight for this job. The props are set with their locking wedges parallel to the face so that the wedges can be driven loose from right to left--an aid to a right-handed swing. Normally, to loosen the wedge the hammerman swings with the hammer in his right hand while he stands behind the next inside prop and holds onto it with his left hand or by the crook of his left arm. Working from a supported area, the prop usually can be pulled over with the hammer to fall into safe ground. Only occasionally is the slusher-hoist cable fastened to the top of a prop to pull it under

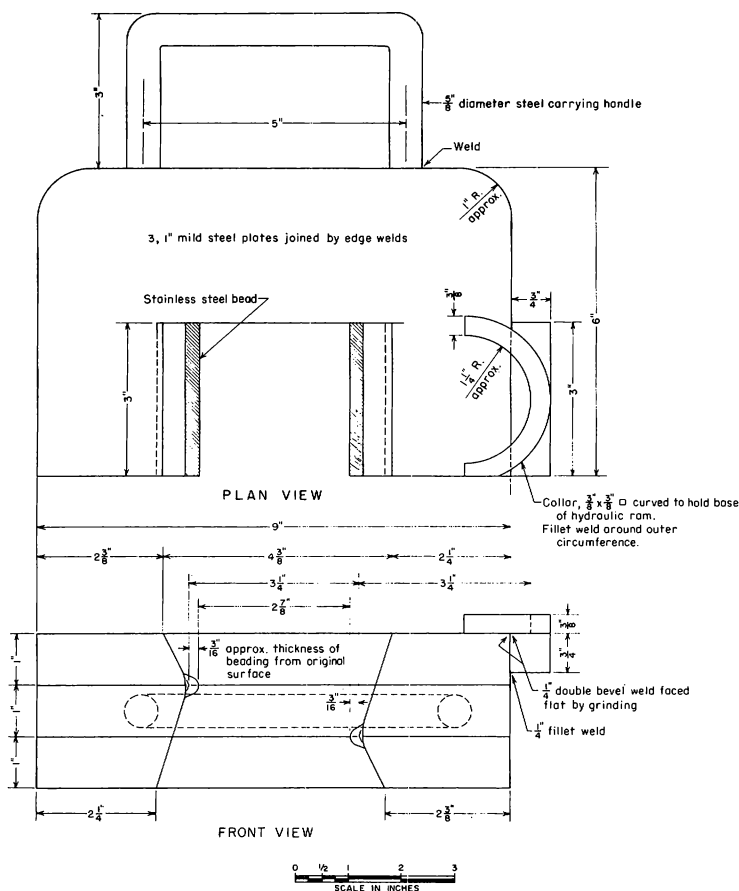
a supported back. The position of the hammerman is not a problem when tightening a wedge because the props are set under supported ground between the face and those in the back row. In a few cases where the ground is heavy, temporary support is gained with expendable wooden props during prop and crib removal.

Caving usually follows the withdrawal of each prop. Normally, it occurs gradually and by the time the last prop is pulled, the cave usually is broken all along the new back row. The back seldom falls immediately after a prop is collapsed, but when falls do occur the 20-hp. slusher hoists are able to pull the props free. Moreover, a cantilever of rock will seldom overhang the last row. If an overhang fails to fall by the next day, cutoff holes are drilled, lightly charged, and shot to induce it to fall. The thicker sandstone lenses tend to overhang; mudstone and siltstone lenses break gradually and fall in small pieces.

The props are set in hitches that are easily chopped in the soft sandstone and mudstone bottom with an axe. After the lower shaft is set in the hitch, the upper shaft is telescoped vertically to the back. A 2-1/2 by 3-1/2 by 6-in. squeeze block, placed between the four prongs that protrude from the prop head, increases the friction between the back and the prop. The locking wedge is then tapped just tightly enough to hold the prop upright until set. A nearly uniform setting load is placed on each prop by hydraulically forcing the two telescoping shafts apart using a 10-ton ram. Because of this initial compression, the prop better resists the initial downward movement of the back and is better able to withstand the jolts from the dynamite blasts that follow. Because of the unevenness and softness of the rock and the softness of the wood squeeze block, some yield probably occurs immediately after setting. The 10-ton initial load is sufficient to compensate for this yield. A German-made 6-ton Neuhaus load setter did not prove successful at the Radon. This lever-operated ratchet device was unable to place the props under sufficient load.

To use the 10-ton hydraulic ram, 4- by 4-in. steps were welded on one side of the lower shafts immediately beneath the lock assemblies. The ram is placed upright on a step with its 2-1/4-in. diameter base set in a semicircular retaining flange welded to the top of the step. A heavy 3-in. horseshoe-shaped steel saddle, which has a square slot cut through its center to fit three sides of the upper shaft, is slid down to rest on the ram's piston (fig. 13). The piston head fits into a curved flange similar to that on the step. The ram is then loaded with a hand pump. The slight pivoting action of the saddle grips the upper shaft and keeps it from slipping. With the ram in place, the lock assembly is clamped by hammering the rotating wedge tight. The Blackhawk Porto-Power model RC-159 ram has a collapsed height of 11-3/4 in., a piston travel of 6 in., and weighs 10 lb. The hand-operated Porto-Power model P-76 pump can develop 10,000 p.s.i.

Unlike the two rows of props that are leap-frogged one past the other as the face is advanced, the crib row is moved each time. The crib supports are centered between the two rows of props, and as the outer props are moved forward, the cribs must be moved as the shift progresses downward. When the crib



**FIGURE 13. - Removable Saddle for Applying Setting Loads to Props With Hydraulic Ram.**

support is freed from the pressure of the back, the steel base is rolled and pulled 4 ft. forward to its new position. This usually can be done by two men; occasionally, the slusher hoist is used. The cribs are spaced one for every four props on 14- and 16-ft. centers depending on whether the props are on 3-1/2- or 4-ft. centers.

The crib supports are built of two units: A lower one-piece steel base and an upper multipiece hardwood timber crib. The timber cribbing blocks are piled on four steel crib releases that are bolted to the top of the steel base. The bases are 21 in. square, and are made of four 4-in., 13.6-lb. H-beams welded to a 1/2-in. base plate (fig. 14). Two 5-in., 6.7-lb. channels tie the top of the columns together. All members are welded, and strengthened with braces made of 1/2- and 3/8-in. plate. The crib bases range from 30 to 60 in. in height and average 36 in. high. They weigh 295 to 435 lb. Crib supports are made of as much steel as possible to minimize compressive deformation of wood.

The four steel crib releases are bolted to the top of the two channels, one at each corner. The hardwood cribbing--green oak in 24-in. lengths, 8-in. widths, and 1-, 2-, and 6-in. thicknesses--is crisscrossed on top of the releases and wedged to the back. The oak timber is cut in Arkansas and in 1959 cost \$155 a thousand board feet delivered. Hecla Mining Co. purchased 162 steel crib bases from Young's Machine Shop, Monticello, Utah, for \$60 each. The crib releases purchased last were cast by the Coeur d'Alene Hardware and

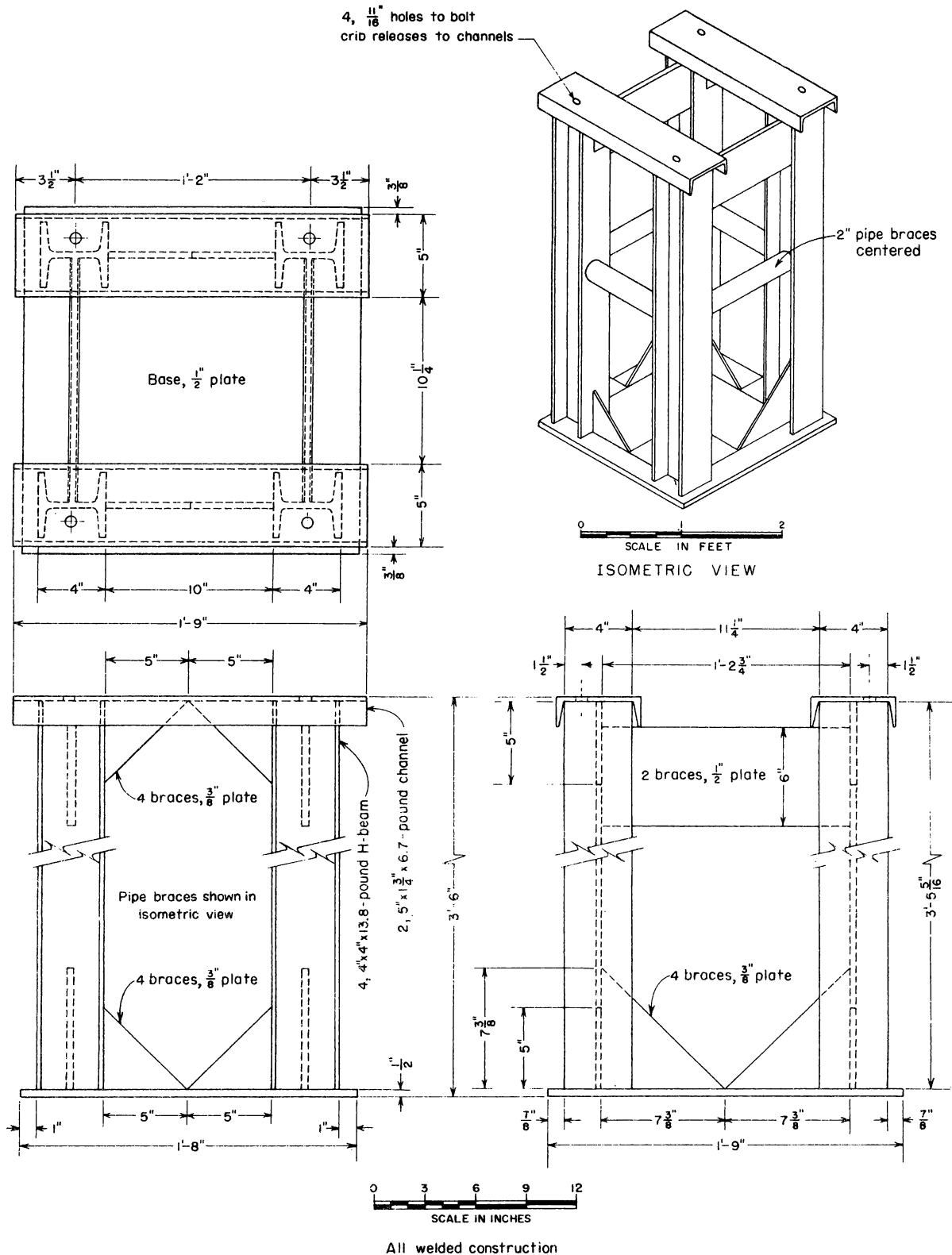


FIGURE 14. - Steel Crib Base, 3-1/2 ft. High.

Foundry Co., Wallace, Idaho. Earlier, crib releases were purchased from Mining Progress Co. and Bethlehem Steel Co. Originally, the simple double-wedge crib release was developed and patented in England and has been used extensively in Europe in longwall mining. The release angle of  $25^{\circ}$  (measured from the horizontal) was not steep enough for easy release at the Radon mine and it was increased to  $34^{\circ}$ . The releases are tripped to spill the cribbing blocks to the floor by striking the hinged T-latch an upward blow. When the steel bases are installed, they are set so that the release latches are parallel with the face. Otherwise it might be difficult to reach the two latches on the cave side. To date, 660 crib releases have been purchased at a cost of \$18 each.

When longwall retreating was started at the northern end of the property, cribs were not used for ground support. The props were installed on 4-ft. centers, but three rows were set instead of two. The same row interval of 4 ft. was used as now. The back was shot down all along the outer row to initiate caving. The third row of props was later eliminated to shorten the supported width from a maximum of 16 ft. to 12 ft., and thereby relieve some of the pressure on the first two rows. Cribs were added to the support system when a major cave occurred that crossed the prop lines to the face. Cribs are now an important part of the roof-support system because they not only add solid support to the break line, but they also increase the miner's confidence. Their large bearing area relative to their height offsets the effect of eccentric loads. Where conditions have indicated that the ground might cave differently, the spacing of props and cribs has been modified to match those conditions.

### Underground Transportation

#### Transportation on the Ore Level

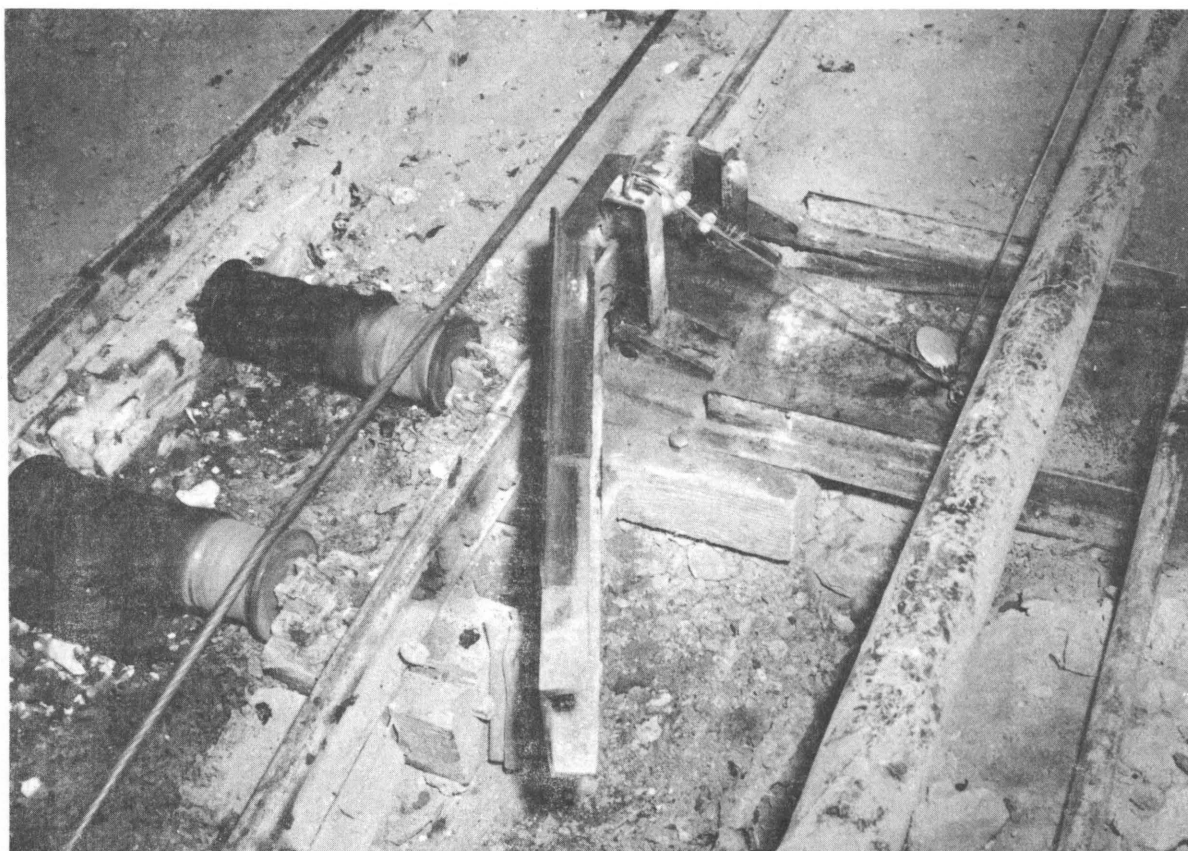
After the ore is scraped into the drifts, it is loaded into 41-cu.-ft.-capacity Granby mine cars with Eimco 12B loaders. Usually, at a longwall face, one contractor mucks and trams the ore while his partner gets the slusher ropes out of the way and loads the next round. Overhead loading is preferred to scraper-slide loading because, car-for-car, the overhead loaders can load faster and can do a better cleanup job. Scraper-slide loading methods were tried but were found cumbersome. Loading from a timber slide built across the track was discontinuous, because only one car could be pushed under the ramp at a time and loading stopped while the cars were switched. Another scraper-slide method tried was to load three cars at a time under a long slide built parallel to the track. The ore was relayed by two scrapers, one moving the ore to the drift and the other moving it to and over the slide. The slide was built some distance from the face so that it would not have to be moved too often. Both systems proved troublesome, and much time was spent in building and dismantling the slides.

With the overhead loaders, only one mine car is used to shuttle between a face and the raise. Due to the short haul, little haulage time, if any, could be saved if strings of cars rather than single cars were used. Various types



of switches and transfers were tried, but one man could load and tram more efficiently using one car than if he had to switch cars. Furthermore, the cost of installing and moving switches or transfers along with the retreat is eliminated. Four 3-ton battery locomotives are used on the working levels. The 41-cu.-ft. Granby cars hold 1.75 tons of ore. To keep the drift height to a minimum, the 12B loaders were equipped with 10-in. rather than 14-in. wheels.

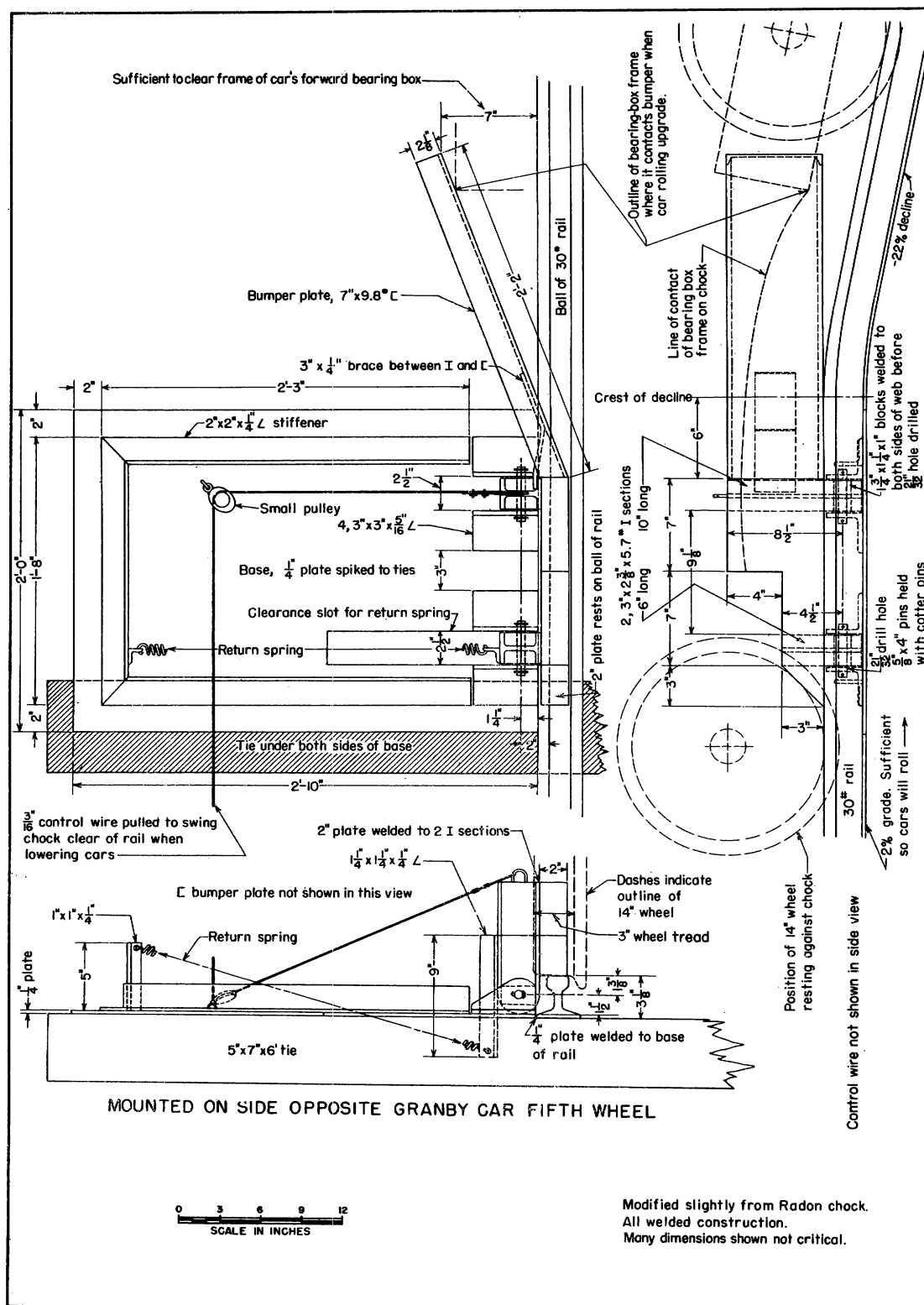
A single-drum 30-hp. Vulcan-Denver hoist at the top of the Far West incline hoists and lowers the Granby cars in trains of three to the Far West chute level. At the crest of the 22-pct. incline a hinged safety chock, its base spiked to the track ties, checks the backward roll of the mine cars down the incline (figs. 15 and 16). The 2-in.-thick steel chock is spring actuated



**FIGURE 15. - Safety Chock at Far West Decline.**

to lie normally on top of one rail. When the cars are being hoisted, the frames of the bearing boxes strike a bumper to pivot the chock to the side and free from the path of the wheel. A return spring pulls it back. When the third car is past the chock, the hoistman lets the cars roll back against it and then unhooks the hoist cable and couples the locomotive for the trip to the ore pass. To lower the empties, the hoistman pivots the chock away from





the rail by a control cable. A 4-ton Mancha locomotive is used on the Far West level. Camel-back dump blocks tip the Granby cars at the raises to empty onto 8-in. grizzlies set 2 ft. lower than the track. The grizzlies are built of 30- and 40-lb. rail sloped away from the track 8 in. in 7-1/2 ft.

#### Transportation on the Haulage Level

On the haulage level ore is moved from the chutes to the station in five 40-cu.-ft.-capacity rocker mine cars pulled by a 1-1/2-ton General Electric battery locomotive over 30-lb. rail laid to 24-in. gage.

The guillotine-type chute gates are air operated and undercut the flow of the ore. The chute stations are timbered with sets of 8- by 8-in. ponderosa pine and have 8 ft. of headroom. The steel gates and the chute mouths are built as integral units before installation, and are bolted to the timbers at a minus 50° angle (figs. 17 and 18). The sides of the chutes are made of 1/4-in. plate, and the bottoms are 3/8-in. abrasive-resistant plate. The gates are actuated by 6-in.-diameter 24-in.-stroke air cylinders controlled by the motorman through four-way solenoid air valves, which are energized by three-pole electric pull switches (fig. 19). The motorman positions each car by a signal light flashed on by a lever-action limit switch in contact with a projection on the side of the car.

At the station, the motorman dumps the cars into the slusher pockets on either side of the shaft. The motorman slushes the ore into two sheet-steel measuring cartridges beneath the pocket with a 36-in. scraper pulled by a 10-hp. two-drum electric hoist. The hoist is mounted above the north end of the pocket with one rope reaved through a 14-in. block hung at the opposite end. The pocket is lined with 30-lb. rail spiked on 12-in. centers to 10- by 10-in. sills and posts. The steel-measuring cartridges are rectangular in section, inclined at minus 50°, and have a capacity of 49-cu. ft., 1 cu. ft. less than the capacity of the skips.

The motorman operates the slusher hoist from an enclosure at the side of the measuring pockets beneath the deck of the station. An expanded-steel-grating cage protects him from the slusher ropes. The pockets are emptied into the skips through guillotine gates. Hecla Mining Co. favored the use of slusher pockets over conventional skip pockets mainly because it was not known just how the mudstone waste would flow.

#### Hoisting

The ore is hoisted to the surface in two 50-cu.-ft., or 2-1/3-ton, Kimberly-type skips suspended below the cages. In full dump position the skip dump rollers are 48-1/2 ft. above the collar of the shaft. Radon ore is dumped into a 200-ton-capacity, hopper-bottom steel bin built inside the legs of the headframe. Hot Rock ore is dumped into a 30-ton-capacity cylindrical, conical-bottom bin behind the main bin. A steel trough is lowered across the top of the main bin by a 5-hp. tugger to direct the Hot Rock ore. The bins are emptied through two guillotine gates. Two electric vibrators welded to the bins are energized through limit switches when the gates are opened.

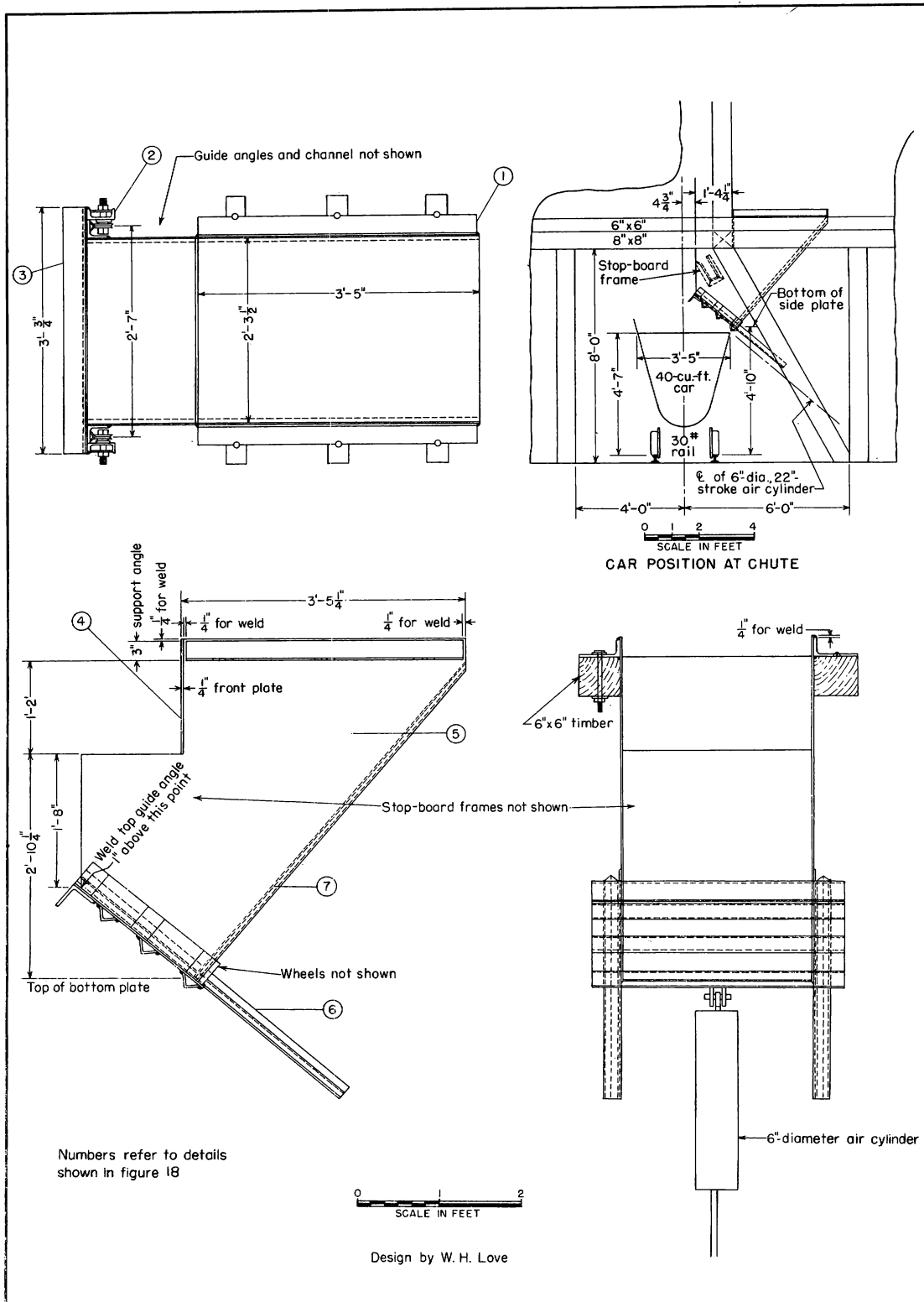


FIGURE 17. - Hecla Chute-Gate Assembly.

**FIGURE 18. - Hecla Chute-Gate Detail.**



**FIGURE 19. - Loading on the Haulage Level.**

The hoist is a 1923, two-drum Vulcan-Denver with a rope speed of 425 ft. a minute (fig. 20). Its 42-in.-diameter drums have 24-in. faces set 4 ft. 5-1/2 in. between centers and are equipped with post brakes and band clutches. Power is applied through a 100-hp. 440-v. induction motor. Lilly Simplex controllers are provided. The Lang-lay ropes are 7/8-in.-diameter. The steel four-post headframe stands 75 ft. 7-1/2 in. high to the centerline of the 48-in. sheaves.

#### Roof Bolting

Roof bolts are used extensively to hold the backs in the development headings. The backs are not heavy, but the mudstone lenses slough almost continuously, unless stopped by a competent sandstone lens or held artificially. First to slough is material partially broken by the blast; the mudstone continues to spall after the surface layer dries and parts from solid ground behind it. The back will arch, but usually the corners of the arch will fall out, and natural stoping will continue upward. The roof bolts are not used to increase the beam strength of the strata over the small openings; rather, their job is to simply compact and reinforce the skin surface. Because arching seldom prevents caving, drift backs are broken flat to provide a better bolting surface for headboards.

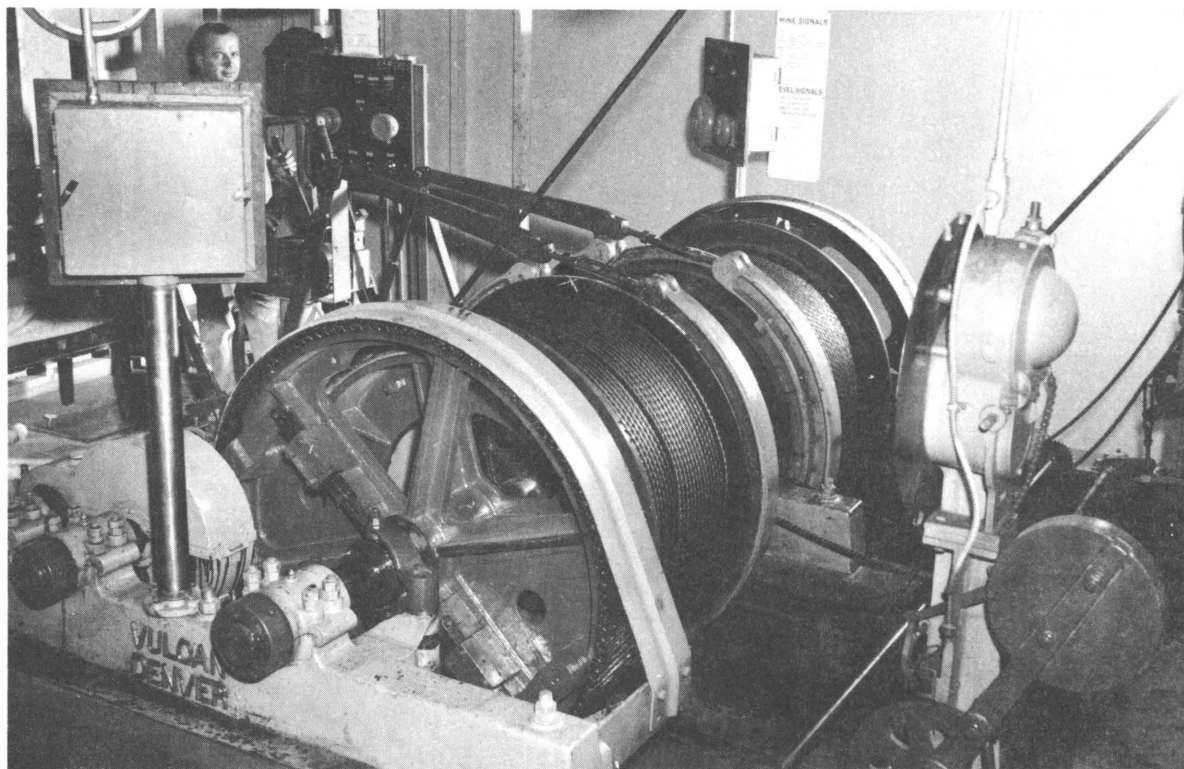


FIGURE 20. - Hoist.

Split-rod-and-wedge-type mild-steel bolts, 4- and 6-ft. long and 1-in. in diameter, triangular steel washers, and pine headboards are used to bolt the backs. In the ore-level drifts mostly 6-ft. bolts have been used, but 4-ft. lengths have performed well in the Cutler formation. In the 1,950-ft. Hot Rock haulageway 4-ft. bolts were used. Roof bolts have not been installed in any set pattern, but are used randomly to match irregularities. The ponderosa pine headboards are 3- by 10-in. timber cut in 24- to 39-in. lengths; most of the headboards are 30 in. long. Throughout the development headings, an average of one bolt has been installed for every 1.2 ft. of lineal advance. About 30 pct. of the backs bolted is lined with wood. Close headboard spacing has prevented sloughing, but occasionally the mudstone has sloughed from behind the boards. The drifts are dry and well ventilated, and the wood should serve its purpose before it rots.

One possible disadvantage of roof bolting the drifts is that frequently they have held almost too well. As the break line of the advancing cave moves along the drift, that part formerly the back of the drift often does not fall evenly behind the break line. The roof bolts will hold the old back for 20 to 40 ft. behind the rear prop line. The lag in the cave, however, has remained nearly constant and has not caught up in a single large fall.

In 1958, a 6-ft. bolt with washer cost \$2.15, while a 4-ft. bolt cost \$1.70. The contract labor cost to install a 6-ft. bolt was \$2 while the cost to install a 4-ft. bolt was \$1.75. Including the cost for the headboard, which cost \$70 per thousand board feet, and other direct and indirect costs such as drill steel and bits, compressed air, and handling, the cost to install a 6-ft. bolt was close to \$4.85. The cost to install a 4-ft. bolt was about \$4.10. Installing a 6-ft. bolt for every 1.2 lin. ft. of drift cost about \$4.04 per foot of drift. Using 4-ft. bolts, the cost was about \$3.42 per foot of drift.

#### Summary of Equipment, Power, and Explosives

The principal items of equipment used at the Radon mine include the following:

- 1 locomotive, 4-ton, battery, Mancha.
- 4 locomotives, 3-ton, battery, Mancha.
- 1 locomotive, 3-ton, battery, General Electric.
- 1 locomotive, 1-1/2-ton, battery, General Electric.
- 5 loaders, overhead, Eimco 12B.
- 15 rock drills, air-leg-mounted, Cleveland H10AL (AL-92 leg) and Ingersoll-Rand JR-38A & BB (AL-93 leg).
- 1 rock drill, air-leg-mounted, dry, Cleveland LLV, with Cleveland LXI dust box and AL-92B leg.
- 2 rock drills, Holman Dryductor.
- 3 rock drills, stoper, Ingersoll-Rand R-58.
- 2 rock drills, stoper, Ingersoll-Rand RP-38.
- 5 paving breakers, Ingersoll-Rand PB-59.
- 4 impacttools, Ingersoll-Rand 534.
- 2 hitch cutters, Ingersoll-Rand 273.
- 20 mine cars, 41-cu.-ft., Granby-type.
- 10 mine cars, 40-cu.-ft., rocker-type.
- 2 scraper hoists, 3-drum, 20-hp., electric, Joy A2F-311 and B2F-311.
- 9 scraper hoists, 2-drum, 20-hp., electric, Joy A2F-211.
- 2 scraper hoists, 2-drum, 10-hp., electric, Joy FF-211.
- 2 scraper hoists, 2-drum, 10-hp., electric, Ingersoll-Rand 10NN-1F.
- 1 scraper hoist, 2-drum, 5-hp., air, Joy S-211.
- 2 tugger hoists, 1-drum, Ingersoll-Rand EU and HU.
- 1 tugger hoist, 1-drum, 500-lb.-pull, air, Joy AW-80.
- 2 pumps, centrifugal, 2-in., 7-1/2-hp., Fairbanks-Morse 5553B.
- 1 pump, Gardner-Denver, duplex steam, 2-1/2 by 1-1/2 by 3.
- 1 hoist, double-drum, 100-hp., 7,500-lb.-pull, 425-f.p.m., Vulcan.
- 1 hoist, 30-hp., single-drum, Vulcan.
- 3 compressors, Ingersoll-Rand 90B type 40; 75-hp., electric motor.
- 1 swing cutoff saw, 7-1/2-hp.
- 1 timber saw, portable, electric, chain.
- 1 forklift, 3-ton Clark.
- 3 fans, axial-flow, 3-hp., Joy I-19.
- 1 fan, axial-flow, 5-hp., Joy I-21.
- 1 fan, axial-flow, model 21-1/4 - 17-1/2 - 3,450, 15-hp., Joy series 1000.
- 4 fans, axial-flow, model 25-1/4 - 17-1/2 - 3,450, 20-hp., Joy series 1000.

- 1 fan, axial-flow, model 38 - 26-1/2 - 1,750, 20-hp., Joy series 1000.
- 1 fan, axial-flow, model 21-1/4 - 17-1/2 - 3,450, 20-hp., 2-stage Joy series 1000.
- 2 pickup trucks, 1/2-ton.
- 1 panel truck, 1/2-ton.
- 1 water truck, 1-1/2-ton.
- 1 ambulance, station-wagon.

During 1959 an average of 20 kw.-hr. of electricity was consumed to produce a dry ton of ore, and the cost for power averaged \$0.26 per dry ton of ore. The average cost for power during 1959 was 1.31 ct. per kilowatt-hour. Dynamite consumption in 1959 averaged 2.6 lb. per dry ton. The powder and primer cost in 1959 was \$0.79 per dry ton.

#### SURFACE PLANT

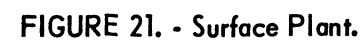
The surface plant was designed to handle 250 tons of ore a day, but provisions were made so facilities could be expanded to handle 400 tons, should conditions require it (fig. 21). The small, compact plant is housed in nine small Armco steel buildings. Most foundations were poured on sandstone bedrock. While most of the mineyard is at shaft-collar elevation, three buildings were constructed on a bench 10 ft. higher, which was blasted in the sandstone on the upslope side of the collar. The back legs of the headframe are anchored to footings poured on the bench.

The 76- by 28-ft. mine office building is partitioned into a modern dry-room, warehouse, and offices. The shop building is 36 by 24 ft. and was constructed 21 ft. from the shaft collar. A 45-ft.-long monorail built of 10-in. 35-lb. I-beams was installed 12 ft. above the ground between the pad at the shaft collar and the far wall of the shop. A Yale 2-ton electrically driven carrier hoist is used to lift and move machinery along the monorail. The small shop is equipped with standard shop equipment--a forge, drill press, 50-ton hydraulic press, power saw, bit and shop grinders, welding equipment, and other tools.

The hoist and compressor building on the sandstone bench is 24 by 48 ft. with its 19- by 24-ft. compressor room partitioned and insulated from the hoist room. Compressed air is furnished by three Ingersoll-Rand 90B air-cooled two-stage compressors, each driven by a 75-hp. 440-v. motor. Each of the compressors has a displacement of 566 c.f.m. at 885 r.p.m. At the elevation of the mineyard, each furnishes approximately 420 c.f.m. at 100 p.s.i. The air is delivered to the 278-cu.-ft. receiver outside the building, and then down the shaft through 6-in. spiral-welded pipe. The hoist and compressor buildings were combined so that the hoistman could take care of the compressors. An indicator board at his controls reports on compressor functioning.

Next to the hoisthouse on the bench is a general storage building, 26 ft. 8 in. by 20 ft. which was formerly the power-generator and distribution building. Before the installation of lines of Utah Power and Light Co. to the mine in October 1955, the company generated its own power with three 125-kv.-a. Caterpillar D-13000 diesel-electric sets operated in parallel. Behind the

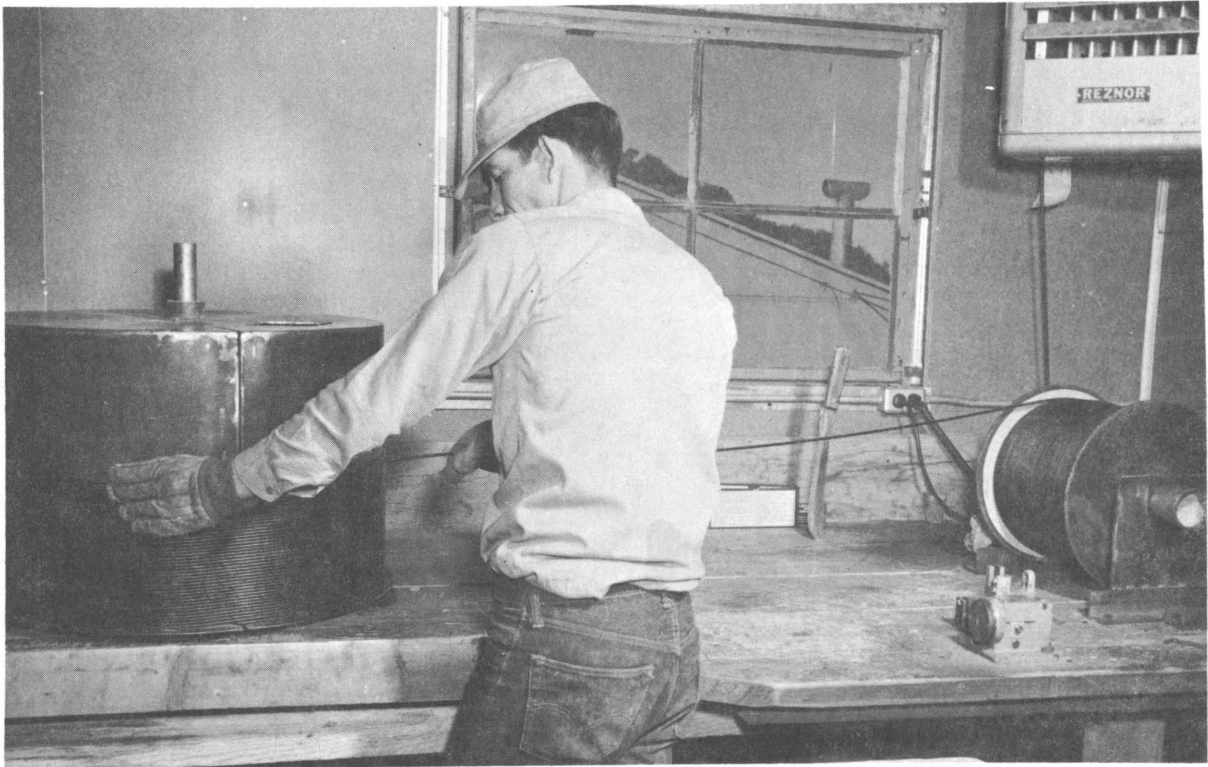






building stands a 300-kv.-a. 12,470-440-v. transformer bank for the surface plant, and a 150-kv.-a. 12,470-2,300-v. bank to supply the underground workings.

A 10- by 10-ft. insulated fuse-capping room is partitioned within the storage building. Its temperature is maintained at a minimum of 65° F. by its own thermostatically controlled propane heating unit. The general-surface man cuts standard lengths of fuse on a company-made measuring device (fig. 22).



**FIGURE 22. - Wrapping Fuse on Measuring Drum.**

The device consists of a 26-3/4-in.-diameter steel drum driven to turn on a vertical axis at 38 r.p.m. by a 1/6-hp., 110-v., 1,140-r.p.m. motor through a system of V-belt-pulley reductions. The drum turns above a work table, the drive rigged below it (fig. 23). The face speed of the drum is about 268 ft. per min. Using a foot switch to turn the apparatus on and off, the operator wraps 87 turns on the drum, pulling the fuse directly from its reel, which turns on a horizontal shaft in its own rack. The wraps are cut into 7.1-ft. lengths by knifing down through a slot in the drum. The operator must slice straight down and cut squarely across the fuse. About 87 fuses are cut in 2-2/3 min. After the caps are crimped to one end and the igniter-cord connectors to the other, the fuses are rolled in bundles of 25 and stored until taken underground. During the first half of 1959, the fuseman averaged 200 completed fuses per hour. The labor cost per completed fuse was 1.74 ct.

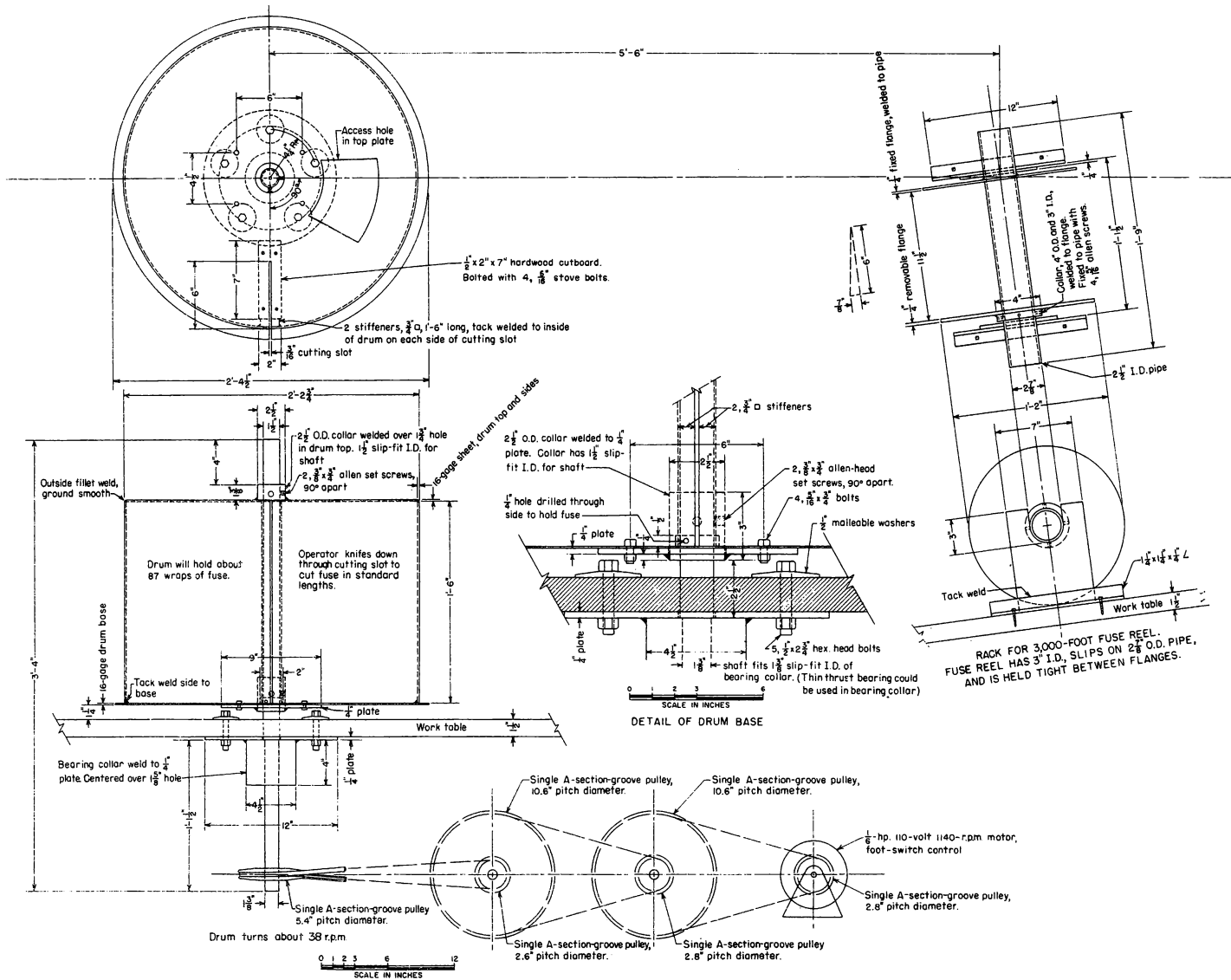


FIGURE 23. - Measuring Drum to Make 7.1-ft. Lengths of Blasting Fuse.

On the opposite side of the hoist and compressor building is the core-storage and sampler's laboratory. This small building is 16 by 20 ft. and contains racks that hold the core that was cut when the property was drilled. There the sampler crushes, pulverizes, dries, splits, and scales his samples.

Water for the mine and plant is stored in two 5,000-gal. tanks mounted on the hillside 90 ft. above the mineyard to provide a head of 39 p.s.i. at the elevation of the yard. The water flows down through a 4-in. surface line and through a bypass in the pumphouse at the edge of the yard. The 10-ft. 8-in. by 8-ft. building houses two 7-1/2-hp., 2-in. centrifugal pumps, which are used either to force the water up to the storage tanks or to increase the pressure in case of fire. From the 4-in. main buried in the mineyard, 2-in. lines feed three 1-1/2-in. fire hydrants. Two of these are spaced on opposite sides of the shaft collar, one near the shop and the other at the warehouse, and a third stands between the pumphouse and dryroom. Two 2-in. lines are hung down the shaft, one for regular underground distribution, which is wrapped with electric heat line, and the other a dry line of the shaft sprinkling system. The water tanks are insulated against freezing by glass wool and corrugated aluminum sheeting. The exposed 4-in. line to the pumphouse is wrapped with thermostatically controlled electric heat lines, blanketed with rock wool batts and covered with corrugated aluminum sheeting. The thermostat control is set in the mineyard. All pipelines that are 4 in. or larger are of spiral-welded construction. Victaulic connections are used on all pipe 2 in. or larger.

Water consumption averages 48,000 gal. a month. The company has contracted the 5.6-mi. water haul from Rattlesnake Spring near the Rattlesnake Ranch north of the mine. The contractor uses a 3,000-gal. tank truck and charges \$5 per 1,000 gal. delivered. He in turn pays the rancher \$1 for each 1,000 gal. he loads. The water cost was \$0.05 per ton of ore in 1959.

The powder magazine is in a small wash about 300 ft. east of the mine plant. It is a 10-ft. 8-in. by 8-ft. Armco steel building, 8 ft. high, with a 1-in. shiplapped inner wooden wall bolted to the panel ridges of the outer steel wall. Between the two walls, 3 in. of sand was poured. Powder is purchased in 15-ton lots, and it is taken underground in 1-to 5-ton lots. The ambulance garage is 12 by 24 ft. and was constructed at the edge of the mineyard near the road. The cap storeroom is at the rear of the garage. Oil and grease are stored in a locked 6-ft. 8-in. by 8-ft. steel building behind the shop.

Timber is cut to size on a 24-in.-diameter, 7-1/2-hp. circular swing saw centrally located in the mine yard. All roof-bolt headboards, cribbing, blocks, wedges, and other small timber items are packaged with steel straps in bundles that can be handled by one man. They are stored in the mineyard on wooden pallets. Material handling around the yard is done with a 3-ton forklift truck (fig. 24). Most of the supplies taken underground and the smaller items of equipment that frequently move between the mine and the repair shop are also handled on wooden pallets. Lifted on or off the cages at the collar by the forklift, the pallets are skidded onto the stations by small one-drum air-



**FIGURE 24. - Lifting Pallet of Props on Cage.**

driven hoists so the cages and skips are released quickly for other duty. The pallets are made of fir and have a steel loop on the end for skidding.

Electric power is purchased from Utah Power and Light Co. under a general industrial-service, medium-voltage schedule that applies to buyers requiring 2,300 to 15,000 v., and who will contract for not less than 35 kw. The power-cost rate that was in effect in 1959 was as follows:

**Demand charges:**

\$2.10 per kw. for first 100 kw. of demand  
 2.00 per kw. for next 400 kw. of demand  
 1.85 per kw. for all additional kw. of demand

**Energy charges:**

0.94 ct. per kw.-hr. for first 20,000 kw.-hr.  
 0.63 ct. per kw.-hr. for all additional kw.-hr.

The rate is based on the customer maintaining at the time of maximum use a power factor of 85-pct. lagging or higher as determined by measurement. If the power factor is found to be less than 85-pct. lagging, the demand is increased 0.75 pct. for every 1 pct. that the power factor is less than 85 pct. The minimum monthly power charge is \$73.50.

Ore from the mine is transported 38 miles to the Uranium Reduction Co. mill in Moab by a contract trucker in 22-ton cable end-dump trucks or in 25-ton "pups", which consist of tandem-drive, end-dump trucks pulling four-wheel end-dump trailers. The contract haulage cost in January 1960 was 4.7 ct. a ton-mile.

**PRODUCTION RATES, PERCENTAGE OF RECOVERY, SAMPLING, AND ORE-GRADE CONTROL**

**Production Rates and Percentage of Recovery**

The Radon mine in 1959 produced 69,089 dry tons of ore, an average of 260 tons of ore a day. To date (1960), 100 pct. of the ore has been recovered.

Productivity in 1959 for all underground labor and supervision, which included the foreman and the two shifters, averaged 7.6 dry tons of ore per man-shift. For all labor and supervision at the mine, surface and underground, including the office and engineering staff, 5.6 dry tons of ore were produced per man-shift.

### Sampling and Ore-Grade Control

Indirect control of the stoping limits is the job of the sampler. In general, the Radon ore body has well-defined limits; its outline, however, is fairly irregular and masses of waste occur randomly within its boundaries. The boundaries of the ore are determined by a sharp change in grade; the ore stratum may or may not thin at the boundaries.

After every longwall slice, the sampler scans the faces radiometrically and cuts check samples at the ore boundaries for laboratory analyses to verify the results of his scanning (fig. 25). His job is not to estimate the grade



**FIGURE 25. - Cutting Check Samples at Ore Limit. Note paint dashed to mark thickness of ore.**

of the ore along the advancing faces but more simply to determine the position of the ore and mark it. He has only a short time to accomplish his work at a new face. He cannot begin until after the last ore is slushed clear and must complete the job before the blasting boards are stacked against the face. Moving updip, he radiometrically scans the face on approximately 3- to 5-ft. intervals, using red lumber crayon to mark the upper and lower limits of the ore. Four counters--Babbal models 600-A, 600-BS, and 610--are kept on hand to make sure that at least two are in repair. The counter probe is equipped with a lead shield to minimize the effect of the background count. The sampler corrects for changes in the sensitivity of the counter by calibrating its meter against the known value of a standard sample. He does this immediately before he goes underground, but carries the standard with him to recheck his instrument at the face. Over the crayon marks, the sampler dashes a more lasting and vivid line using a spray can

of red paint. The miners then pattern their drilling so that the ore layer is near the back. The high-grade ore layer pinches and swells but has an average thickness of 2-1/2 to 3 ft.; occasionally, its thickness changes abruptly. Above and below the high-grade layer lies as much as 1 ft. of submarginal rock that graduates into ore within 3- to 6-in. bands.

Generally, at the updip and downdip boundaries of the ore body, the ends of the longwall faces are kept in submarginal ground 5 to 10 ft. beyond the ore limit. The irregular boundaries of the ore body could not be determined precisely by surface drilling, and breaking some submarginal rock is expedient to determine the exact boundaries and not leave any ore unmined. To verify an ore boundary after scanning, the sampler chips two samples; the first where the counter indicates the boundary to be, and another 5 to 10 ft. beyond. If

the last sample is ore grade, the face is lengthened by drifting. Should it show submarginal ore, about 0.10-percent  $U_3O_8$ , the final rows of blast holes are fanned out to lengthen the face.

In the sampling room, the chip samples are crushed in a 4- by 6-in. jaw crusher, pulverized to minus 60- to 80-mesh, and radiometrically assayed on a Nuclear Chicago, model 183, scaling unit.

## VENTILATION

### Development

Quick removal of blasting fumes, dust, and the daughters of radon gas was important for rapid, safe progress of the development work. During the development period the ventilation system was operated to exhaust the foul air from each of the working faces to the surface. Down the manway of the shaft was hung 630 ft. of 30-in.-diameter 18-gage galvanized pipe in 5-ft. lengths. The pipe was connected to the 5- by 6-ft. ventilation incline that was driven up the dip of the beds from the lower to the upper ore level opposite the manway of the shaft. Timber overcasts were built where the incline met with the two intermediate levels, and a timber bulkhead was built at the upper level. The overcasts were gunited and covered with fine rock to prevent leakage. Fan line connections were made at each level, and 10-ft. lengths of 20-in.-diameter, 20-gage ventilation lines were kept extended to the drift faces as they were advanced. A Joy Series 1000 20-hp. model 38 - 26-1/2 - 1750 single-stage Axivane fan in the headframe exhausted from the 30-in. shaft line. Joy Series 1000 20-hp. model 25-1/4 - 17-1/2 - 3450 single-stage Axivane fans were installed as boosters in the 20-in. fan lines at each of the two overcasts and at the bulkhead on the upper level. By operating the system on exhaust, the contract crews could blast at any time during a shift. The system provided 20,000 c.f.m. through the shaft and drifts while the mine was being developed.

### Stoping

As soon as a connection was made with the adjoining Far West mine on the north, double two-piece doors were built as air locks in each of the four drifts near the ventilation incline. The north Radon and Far West stopes were put on positive ventilation with fresh air being forced past the air locks through 20-in.-diameter pipe. The fresh air enters the mine through the shaft, travels up the service incline, and is forced by the 20-hp. fans through each of the north drifts. The air travels along the longwall faces and through the cave and is exhausted through the exhaust system at the adjoining Far West mine. The south Radon stopes are ventilated by the same exhaust system used in the development stage of the mine. The large exhaust fan in the headframe can be reversed by the hoistman. Ventilation in the Hot Rock stope is provided by forcing fresh air through the Hot Rock drift past a ventilation door near the Radon shaft, and exhausting through the exhaust circuit of the adjoining Cord mine to the south. The total air moved is 43,000 c.f.m. The longwall mining faces are easy to ventilate.

The ventilation doors are hung in frames 5 ft. 10 in. wide by 6 ft. 5 in. high made of 10-in. channel iron (fig. 26). The frames are embedded in



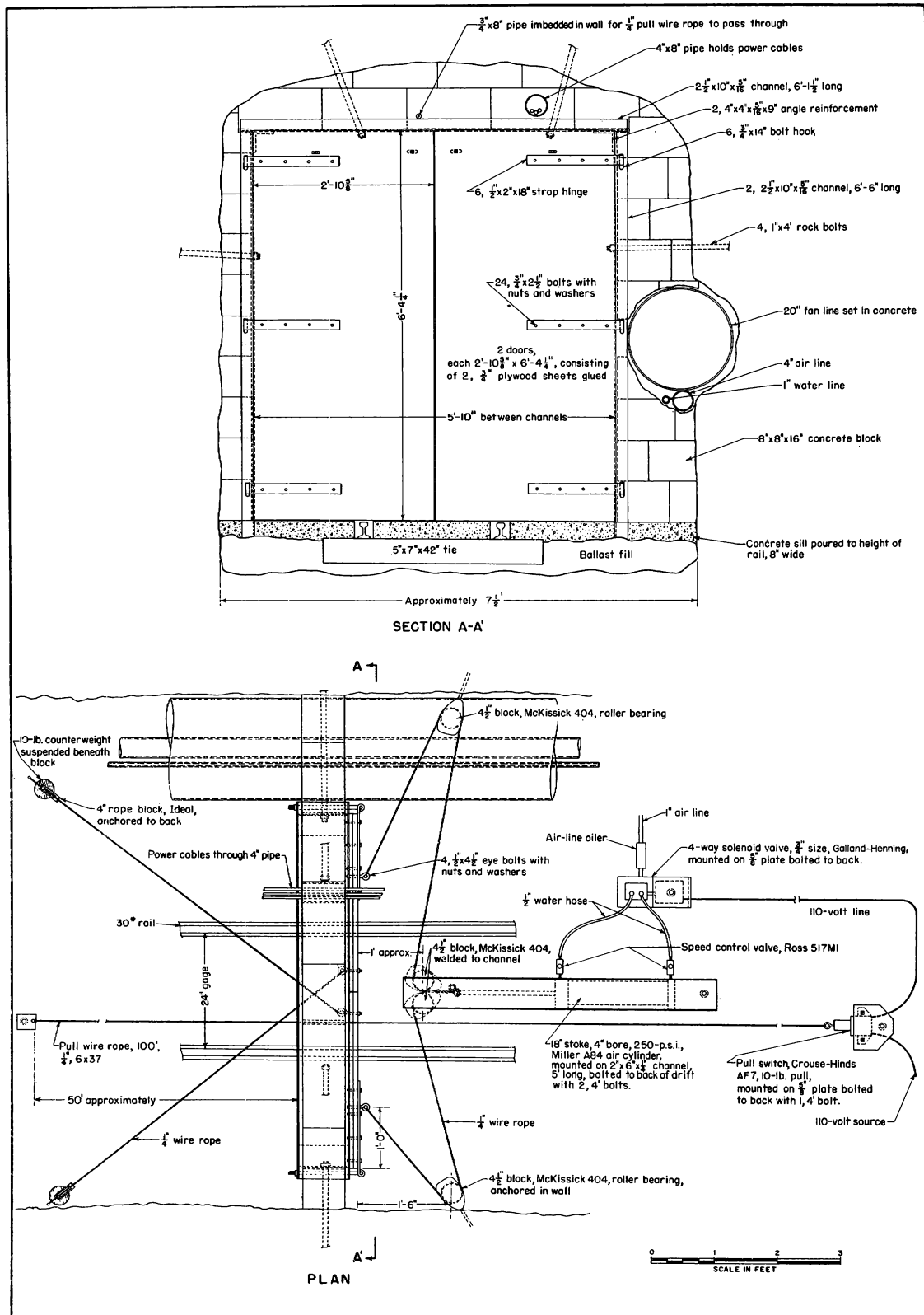


FIGURE 26. - Air Door.

concrete, and the walls are made of 8-in. concrete blocks. The steel frames are welded to rock bolts anchored to the wall rock around their perimeter. The two-piece self-closing doors are built of two thicknesses of 3/4-in. five-ply plywood glued together to form a combined thickness of 1-1/2 in. Each two-piece door is opened by a 4-in.-diameter, 18-in.-stroke air cylinder which is fastened to a piece of channel bolted to the back of the drift. Two cables, fastened to the end of the piston rod, open the doors through a simple system of pulleys. Counterweights close the doors. Air to move the pistons is controlled through a four-way Galland-Henning Nopack 115-v. solenoid air valve. Ross restrictor valves are used at both ends of the air cylinder to control the speed of the piston. Control wires, about 50 ft. long, are stretched along the backs of the drifts on both sides of the doors. About 25,000 c.f.m. at a pressure of 2.4 in. water gage is moved through the four north drifts.

The immediate daughters of radon gas in the working place air are measured monthly with a Juno meter in dust samples collected on filter paper. Air is drawn through the filter paper with a vacuum pump by standard procedure.

#### LABOR CLASSIFICATIONS, WAGES, AND CONTRACT-BONUS SYSTEMS

##### Labor Classifications and Wages

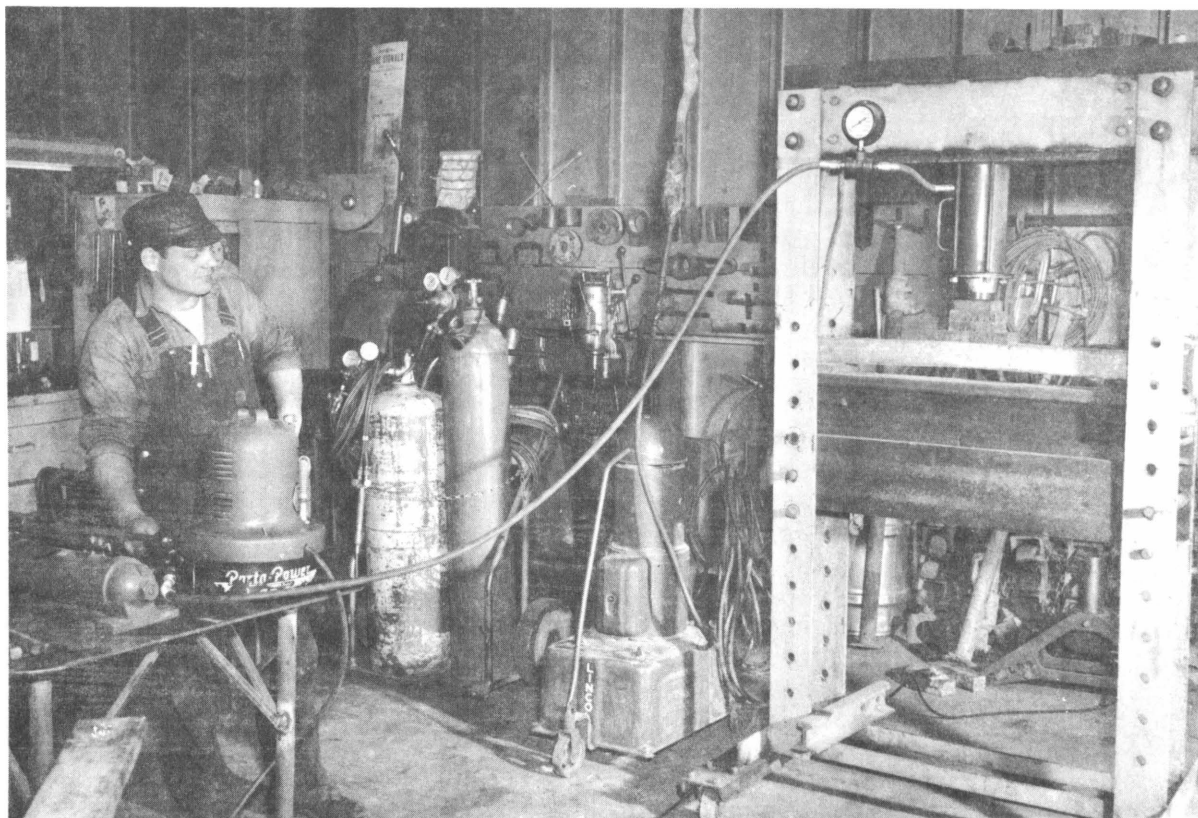
In 1959 the Radon mine operated two shifts a day, 5 days per week. The day shift worked from 7:30 a.m. until 3:30 p.m., and the night shift from 3:30 p.m. until 11:30 p.m. All personnel directly engaged in ore production changed shifts every 2 weeks.

An average of 45 to 48 men are employed (1959) at the Radon mine. This number includes the whole mine force--labor and supervision--both underground and surface. In January 1960, 11 men were employed on day's-pay wages, and 27 men worked under a contract-bonus system. The day's-pay men are listed in table 1 according to their base hourly wage and the number employed per shift.

TABLE 1. - Wage scale

Labor classification	Base rate per hour	Number of men employed	
		Day shift	Night shift
Electrician, underground.....	\$2.675	1	0
Mechanic, surface.....	2.575	1	0
Mechanic, underground.....	2.575	1	0
Hoistman, surface.....	2.575	1	1
Propman, underground.....	2.475	1	0
Cager, underground.....	2.375	1	1
Nipper, underground.....	2.275	1	0
General labor, underground...	2.175	1	0
General labor, surface.....	2.175	1	0
General labor, surface.....	1.975	1	0
		10	2

The work of the day's-pay propman has been important in the conservation of mine props. From October 1956, when stoping began, until December 1957, approximately 185 steel longwall props were lost; in 1958 when one man was assigned the task of caring for the props, only 7 were lost. In 1959 only 3 were lost. Of course, some early prop losses were the result of uncontrolled falls that broke across the prop lines, but many were lost because miners were unwilling to pull partly buried props clear of caved material. Before the mine crews were trained to care about prop loss, the propman often found "lost" props protruding from the cave material. His job is to have damaged props taken to the surface to be straightened (fig. 27) and cleaned, and to deliver them to the stopes where they can be used again.



**FIGURE 27. - Straightening Bent Prop on 50-ton Press.**

The underground mechanic makes all repairs on the scraper hoists, scrapers, trammers, and mine cars, and his counterpart on the surface makes all other repairs and does all fabrication work.

The cagers work on the lower haulage level. They haul the ore from the Radon ore chutes to the slusher pocket and slush it into the loading pockets. The nipper is a tool runner and general underground helper. The man employed as general underground laborer does odd jobs, delivering supplies and timber to the working areas, assisting the propman in gathering and delivering props and cribs, and doing general maintenance and cleanup work in the development

headings. The senior general topman is responsible for care of the bits and steel, timber, steel strapping makeup, and aids the mechanic when needed. The other topman takes care of the mineyard and buildings, saws timber, cuts and caps primer fuses, and does other general tasks.

The supervisory and staff personnel include the superintendent, foreman, two shifters, two engineers, office man, warehouseman-timekeeper, and the sampler.

Vacations are granted to all employees with one or more years of service with the company. For 1 yr. of service 7 days of vacation with 5 shifts' straight day's-pay wages are given to nonsalaried employees. This is increased to 8 days of vacation with 6 shifts' pay for 2 yr. service, 14 days of vacation with 10 shifts' pay for 3 yr. service and increased gradually to a maximum of 21 days of vacation with 15 shifts' pay for 15 yr. service. The employees have six official holidays a year plus a holiday on the individual's birthday. If the employee does not work on a holiday, his holiday pay is his wage for one straight day's-pay shift, which includes the shift premium, is such was being earned. This payment is not construed as a day worked in the computation of overtime for the pay period in which the holiday falls. If he works, he is paid one additional shift's pay at straight time. When holidays fall on Sunday, the following Monday is considered the holiday, birthdays excepted. Exceptions to the official and birthday holidays are that the employee must be employed for one calendar month before the month the holiday falls within, and the employee must work the last scheduled shift before the holiday and the first scheduled shift after the holiday, unless absence is caused by illness, death in the family, or other legitimate reason.

#### Contract-Bonus Systems

The production men at the Radon mine are the contract miners. In 1959, 27 men were being paid under contract-bonus schedules. The four stoping areas--the north and south stopes of the main Radon ore body, the Hot Rock stope, and the Far West stope--were each being worked by a separate contract crew. Except in the Hot Rock stope, where the crew worked only on day shift, each stope crew has its number split equally between the two shifts. The average 1959 contract earnings at the Radon mine were \$39.36 per shift.

There are four contract-bonus rates for work completed in the stopes, based primarily on the heights of the stopes. The four rates are: A cubic-foot rate for drilling and blasting, a cubic-foot rate for slushing and tramming, a unit rate for prop installations, and a unit rate for crib installation. There is a good contract-price incentive to mine low stopes, for mining under low backs at the Radon generally means cleaner mining. To compensate the contractors for the harder work to produce ore under low backs, the unit contract rates increase as the stope heights decrease. One advantage of low stopes is that they are safer places in which to work. The lower the back, the better the choke-off characteristics at the break line, and the less chance there is for rocks to roll down the cave pile into the face area. In lower stopes, shorter props are used; shorter props are stronger. The equiv-

alent cubic-foot rates, which are listed after the four contract rates, are for the reader's interest only and are not the basis for payment. They are the sum of the four based on a cubic foot of stope mined and propped at various stope heights. The cubic-foot conversion cost for prop installation that is included in the equivalent rate is based on a prop spacing of one for every 16 sq. ft. of back. For the cribs, the conversion cost charged to each cubic foot of ore excavated is based on one crib for every 64 sq. ft. of back. Contract stoping rates that went into effect in March 1957 are shown in table 2.

TABLE 2. - Stoping rates

Average height of ore, feet	Contract Rates				Equivalent rate per cubic foot of advance
	Drill and blast, per cubic foot	Slush and tram, per cubic foot	Props, move and set, each	Cribs, move and set, each	
3-1/2	\$0.0875	\$0.101	\$5.42	\$10.84	\$0.334
4	.080	.0925	5.58	11.16	.303
4-1/2	.073	.084	5.74	11.48	.279
5	.0675	.078	5.90	11.80	.256
5-1/2	.0625	.0725	6.06	12.12	.238
6	.059	.068	6.23	12.46	.234
6-1/2	.055	.064	6.40	12.80	.211
7	.053	.061	6.57	13.14	.202
7-1/2 and over	.051	.058	6.74	13.48	.193

Prop and crib installations include moving, setting, lagging, and retightening or resetting if necessary. The contractors have learned that setting props correctly pays for they lose their own time if leaning or fallen props must be reset. Furthermore, when props are knocked down during blasting, additional worktime is lost cleaning up ore shot into the prop area.

Development contract-labor rates were based on lineal advance. Drifting, which included the labor to lay track and hang pipe, was paid for at the rate of \$10 a lineal foot of advance. Inclines, or upgrade excavations, were paid at a rate of \$8.50 a lineal foot for the first 100 ft. of advance increased an additional \$0.50 a foot for each added 50-ft. interval. Declines, or down-grade excavations, were paid at the rate of \$9.50 a foot for the first 50 ft. of advance and increased an additional \$0.50 a foot for each added 50-ft. interval. The beginning rates paid for incline work were lower than those paid for drifting because in incline work the contractors were not required to lay track with the advance, and the rock could be moved quickly from the faces with scrapers. For greater distances, incline work was paid for at rates higher than those for drifting.

The labor cost in 1959 was \$5.65 a ton. This includes all laborers, supervisors, engineers, clerical, and warehouse personnel at the mine.

## SHAFT SINKING DATA

## General information:

Size of rock in cross section.....feet..	9 by 18
Inside timber dimensions of 3 compartments.....do...	4-1/3 by 5-1/2
Total timbered depth.....do...	690

## Work done by contract shaftmen:

Total advance <sup>1/</sup> .....feet..	669.3
Total blast-hole drilling per foot of shaft.....do...	56.9
Average depth of bench-round blasthole.....do...	6.9
Powder loaded per blasthole.....8-in. by 1-1/8-in. cartridges..	11.2
Powder used per foot of shaft.....pounds..	28.2
Rock hoisted per foot of shaft.....30-cu.-ft. buckets..	10.7
Total time worked.....days..	65.5
Total time worked.....man-shifts..	790
Average advance per day.....feet..	10.21
Average advance per man-shift.....do...	.82

## Labor per foot of shaft, in man-hours:

Drilling.....	1.78
Blasting.....	.73
Mucking.....	2.85
Timbering.....	2.01
Hanging pipe.....	.31
General (lunch, etc.).....	1.15
Smoke delay.....	.24
Other delay.....	.64
Total.....	9.71

See footnote at end of table.

## SHAFT SINKING DATA (Con.)

Materials used per 5-ft. 3-in. timber set:

Timber:

Wallplates and endplates, 8- by 8-in.....	board feet..	246
Dividers, 8- by 8-in.....	do.....	64
Posts, 8- by 8-in.....	do.....	213
Blocking.....	do.....	100
Lagging, 2-in. by 4-ft. 6-in, random width.....	do.....	333
Lagging keyboards, 3- by 10-in. by 7-ft.....	do.....	105
Lagging cleats, 2- by 2-in.....	do.....	70
Headboards, blocking.....	do.....	50
Lagging, landing every other set, 2- by 1-in.....	do.....	12
Guide, 4- by 6-in. by 21-ft.....		1
Ladder, staged every other set, 12-ft.....		1/2
Wedges.....		120

Other supplies<sup>2/</sup>:

Nails, 20-d.....	pounds..	2
Nails, 30-d.....	do....	6
Hanger rods, hooked, 7/8-in.....		8
Washers, hanger-rod, malleable-iron, 7/8-in.....		8
Lag bolts, guides, 3/4- by 6-in.....		5
Cut washers, 3/4-in.....		5
Pipe, compressed-air, spiral-weld, 6-in.....	feet..	5-1/4
Pipe, water, 2-in.....	do...	5-1/4
Victaulic coupling, 6-in.....		1/4
Victaulic coupling, 2-in.....		1/4

See footnote at end of table.

## SHAFT SINKING DATA (Con.)

## Other supplies - Continued:

Pipe clamp, hanger.....	1/4
Bell cord.....feet..	10-1/2
Portable cord, 2-wire, No. 10.....do...	5-1/4
Portable cord, 2-wire, No. 16.....do...	5-1/4
Ventilation duct, 20-in.....do...	5-1/4
Lag bolt, 1/2- by 4-in.....	1
Lag bolt, 1/2- by 6-in.....	1
Boiler tubing, casing.....feet..	28

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1/ June 20 - Aug. 1955. 21 ft. sunk and timbered before headframe erected and before contractors began.

2/ 2-in. sprinkler line hung later. 20-in. ventilation duct later changed to 30-in.



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