SOCIETY OF MINING ENGINEERS

10 P

CALLER NO. D, LITTLETON, COLORADO 80127

PREPRINT NUMBER

87-9

MODIFIED VCR STOPING AT THE SIXTEEN-TO-ONE MINE

T. H. Bagan

Sunshine Mining Company
Silver Peak, Nevada

For presentation at the SME Annual Meeting
Denver, Colorado - February 24-27, 1987

Abstract. The Sixteen-to-One Mine was originally designed and operated using Blasthole Stoping with end slicing as the primary mining method. This method changed to Modified VCR Stoping in response to efforts to control dilution. The new method combines Blasthole Stoping with shrinkage methods, pillars, and reduced sublevel intervals to provide the most practical dilution control available for low cost bulk mining.

Project Overview

History and Development

The Sixteen-to-One property lies in the Red Mountain Mining District of Esmeralda County, Nevada, approximately eight miles southwest of the town of Silver Peak. The property's history began in 1935 with the original claim staking, followed by a limited amount of underground development on the vein outcrop by the locator. Subsequent ownership by various mining companies during the '60's and '70's resulted in several diamond drilling programs, exploration drifting, and crosscutting on the vein with no recorded production. Sunshine Mining Company's involvement began in 1964 with the acquisition of a two-thirds interest in the property from Mid-Continent Uranium Corporation, which owned the property at the time. Sunshine's involvement culminated in a decision to mine the orebody in June of 1980.

Mine development started in September 1980 on an underground mine which was to utilize the production capabilities of rubber-tired equipment. Table 1 lists the major pieces of equipment in use at the mine as of June 1986. The development scheme involved the driving of ramps on 15% grade for access to the steeply-dipping vein and ramps driven on 10.5% grade for ore haulage from draw levels to the surface. Sublevels driven on the vein at 1% grade and at full vein width provided the means for production drilling and blasting.

Milling

Mill construction began in April 1981, and by March 1982, the first bullion had been poured from the conventional cyanide facility. The mill utilizes a cyanide leach process, with counter-current decantation for pulp washing and beltpress filtration of tails allowing tailings disposal by conveyor belt to the tails impoundment area, instead of the usual hydraulic placement. The refinery portion of the mill uses Merrill-Crowe precipitation of gold and silver, and the final product is a silver/gold dore' button.

Total mill production for 1985 was 40.7 MT (1.3 million ounces) of silver and 0.26 MT (8500 ounces) of gold from 228,300 MT (251,500 ST) of ore from the Sixteen-to-One Mine. Additional production is also obtained from custom milling ores. Mill throughput capacity presently stands at approximately 726 MT/day (800 stpd).

Geology and Ground Conditions

The geology of the district is composed primarily of Tertiary sedimentary and volcanic rocks. The host rocks for the Sixteen-to-One orebody are late Miocene extrusive volcanic tuffs and breccias with the older volcanics being andesitic in composition and the younger volcanics being rhyolites and latites. The upper elevations of the mine are in rhyolites, while the lower elevations are hosted by the andesites. The andesites are the predominant type.

The vein is a steeply dipping (65°-90°) epithermal fissure vein with a northeasterly strike. Approximate dimensions of the ore shoot are 610m (2000 feet) on strike by 190m (620 feet) vertically. Widths range from 1.5m (5 feet) to 15.2m (50 feet), with the wider portions in the center. The vein tapers toward the surface, with depth, and toward the ends along strike. There is also a significant, mineable, footwall split to the northeast called the Colorado Vein.

The gangue mineralization of the Sixteen-to-One vein is approximately 60% quartz and 40% calcite. The silver mineralization is primarily argentite while the gold occurs microscopically in native form. Wall rock alteration is not well developed, but fracturing sympathetic to the vein walls is common and a cause for wall rock sloughing. Development drifts in waste are driven in tight, blocky ground requiring little if any ground support. Split set rockbolts, wire fencing, and occasional timbering are the preferred ground support with the majority of timber being used in the ore headings and the majority of bolts and wire used in the waste headings.

Uniaxial compressive strength of the wall rock varies from 62.1 MPa (9000 psi) in highly altered rhyolites to 179.3 MPa (26,000 psi) in compact, unaltered, andesite breccia, with an average of approximately 124.1 MPa (18,000 psi). No significant vertical or lateral ground stresses are evident, nor is there any problem with rock strengths per se. The single major ground control problem is the sympathetic fracturing in the wall rock.

Blasthole Stoping

Original Concept

Blasthole stoping was originally selected as the mining method based on the following conditions: (1) an adequate vein width normally over 12 feet, (2) a steeply dipping orebody, (3) a moderately low grade and continuous vein, (4) fairly competent wall rocks and vein, and (5) definite hangingwall and footwall contacts. The information existing at the time of method selection indicated that with a moderate amount of open ground, a dilution factor of 10% would be an appropriate estimate for the Sixteen-to-One orebody. The ultimate contribution of sympathetic wall rock fracturing to the dilution estimate was unforseen and underestimated at the time.

Blasting would be carried out in vertical slashes following the creation at the orebody extremeties of vertical slots or drop raises using VCR raising methods. The vertical slashing would allow visual monitoring of orebody fragmentation between sublevels.

Sublevels would be driven the full length and width of the vein, with centralized access cross-cuts so that mining could proceed in retreat fashion toward the cross-cuts. Sublevel spacing was carefully chosen at 35m (115 feet) sill-to-sill after consideration of such factors as expected drill hole deviation, and the desire to minimize dilution due to changes in vein dip. Figure 1 illustrates the original blasthole stoping concept for the Sixteen-to-One Mine.

The ore, after blasting, falls by gravity to the drawpoint level, where LHD units muck and transfer ore to underground trucks for haulage to the surface. The original concept included draw cones from the draw level to an undercut level to ensure adequate ground support at the drawpoints.

Initial Experience

In mid 1982, the down-the-hole production drill was placed into operation drilling 165mm (6.5 in.) holes. As previously mentioned, the slot raise was drilled first, with a pattern as illustrated in Figure 2.

The center five holes were loaded with high density emulsion explosives, primed with millisecond delay boosters, and shot one lift at a time. When shot, the center five holes cratered downward resulting in an advance of between six and nine feet, depending on ground conditions and powder quantities. This process was repeated until the raise holed through.

After completion of the slot raise, the slashing would begin by loading the production blastholes. These holes were drilled a mininum of three to a row, and deck loaded the full length of the holes. A full row was shot at a time.

Every row of holes was designed by first selecting the proper burden, based on width, for each row. This typically varied from a 1.2m (4 ft.) square pattern to a 3.0m (10 ft.) square pattern, with rectangular pattern variations used as ground conditions dictated. Hole spacings were usually less than the burden in the rectangular patterns. A minimum mining width of 3.7m (12 ft.) was selected to allow for proper muck flow and the rib holes were placed from 0.3m (1 ft.) to 0.7m (2 ft.) inside the footwall and hangingwall contacts.

A geologic section was drawn for each row, with the holes positioned accordingly to minimize dilution, maximize extraction, and maintain desired stoping width. A copy of the section was provided to the driller with collar locations and dips shown for each hole. The driller worked off of survey control and squaring points installed by a surveyor to lay out each row and hole in the stope. Drill cuttings were logged by a geologist for percentage of ore and waste, and this information was used to provide some measure of mining selectivity.

Amounts and types of blasting products were selected by the Mine Superintendent or Mine Foreman as ground conditions warranted. This varied from using high density emulsions to a mixture of ANFO and high-energy emulsion.

As mining progressed, the problem of dilution began to appear and the powder usage in the rib holes was reduced to minimize blasting damage to the stope walls. Also, the lower energy powders were favored in the rib holes, and the higher energy powders were confined to use in the slot raises and the center holes of the slash patterns.

Pillars were added to the mine plan as an added effort to control dilution which was recognized as the sloughing of waste blocks and wedges created by the sympathetic wall rock fracturing. The initial pillar plan was to leave pillars at areas where ground conditions were the worst. Since a new slot raise would have to be drilled and blasted to continue mining the panel, these pillars were preferred to be closer to the sublevel sill and did not extend the full distance between sublevels unless necessary.

Ground conditions had a tendency to worsen as mining progressed to the higher elevations hosted by the highly altered rhyolites, although dilution was a problem in the andesites, also. Eventually, the pillar plan evolved into a regular spacing of pillars, extending the full distance between sublevels, stacked vertically above another.

Modified VCR Stoping

The original stoping concept relied heavily on broken ore in the stope to help support the wall rocks. However, due to the vertical slashing of the ore blocks, muck had to be drawn off to clear the brow for each shot. This left a large amount of open ground in the stope due to rilling of the muck. Under the design conditions of the mine, this would have been satisfactory, but with the dilution problem, the amount of open ground became excessive and had to be reduced.

As previously mentioned, end slicing was preferred over true VCR methods because of the reduced risk of insufficient fragmentation and the ability to visually monitor the stoping results. Bottom slicing, however, afforded the opportunity to decrease the amount of open ground in the stope. Therefore, a compromise was made. A hybrid VCR stoping method would be tried that combined the benefits of both end slicing and bottom slicing, and avoided the risks of true VCR. Termed "Modified VCR Stoping", this method entails blasting a horizontal slice of ore from the block, approximately 3.0m (10 ft.) to 4.6m (15 ft.) thick, completely across the stoping panel between the pillars that were established by the pillar plan. However, instead of only cratering, the rows of holes are loaded for a combination of cratering and slabbing, timed so that they successively break towards a previously established slot raise. Therefore, two free faces are available for each row of holes to break to. Although somewhat expensive, the added cost of the additional free face is felt to be more than compensated by the reduced risk of poor fragmentation and bootlegs possibly going undetected.

The new mining method is expedited in much the same manner as the old. Drilling patterns remain the same, but loading patterns were changed to just a single or double deck of powder as illustrated in Figure 3. The slot raise is holed-through only after the stope back is blasted to within 7.6m (25 ft.) of sill.

After each blast, a tape is dropped down the hanging wall holes for measuring distance to the back and distance to the muck pile to ensure that just enough muck is drawn from the stope to allow room for swell. Ideally, this is approximately a 38% draw, but because of previously mined drifts at lower elevations of the stope, the required percentage is typically less, tying up greater quantities of broken ore inventory for ground control.

Current Progress and Future Activities

Figures 4A and 4B comprise the vertical longitudinal projection of the Sixteen-to-One Mine as of June 1986. Illustrated are the successive zones of mining changes as already implemented, and zones where further changes are planned in the future to augment the mining method change. The future mining area will incorporate approximately a 50% decrease in sublevel interval to decrease dilution caused by changes in vein dip, and to afford greater control over blasting progess. Like wise, maximum heights of stoping panels are being reduced by one-third from approximately 180m (600 ft.) to 120m (400 ft.).

Offsetting the added development costs of these changes is the elimination of the undercut level and drawcones immediately above the draw level, as already partially practiced in the central portion of the main Sixteen-to-One orebody. Drawpoint brows have held up rather well and the drawcones were not contributing greatly to ground stability.

Also offsetting the added costs is the progress made in controlling dilution and arriving closer to projected reserve grades in drawpoint sampling and in mill heads. A qualitative study begun by the mine geologist in June of 1984 and continuing through April of 1986 gives an indication of how the stoping changes have reduced the percentage of observed waste in the drawpoints. As illustrated in Figure 5, as muck haul shifted to stopes where modified methods were used, the percentage of waste decreased significantly.

Calculated reserve grades were correlated with drawpoint sampled grades for the one stope in the mine, the 6890 S.W. Extension, that has been pulled completely empty. Although this stope was mined by end slicing, it lent credence to the calculated reserve grades, because the actual volume mined was measurable. The average grade of the muck hauled from the drawpoints correlated well with the reserve grade after adding in the percentage of dilution. Thus, withthe reserves "performing" well, it could not be argued that dilution was a manifestation of reserve overestimation.

As already seen in Figure 5, 1985 muck haul was almost exculusively from the S.W. VCR and N.E. VCR stopes, which were mined by Modified VCR Stoping. Also in 1985, new production records were set at the mill in ounces produced. This was due, in part, to a mill head grade being approximately 20% higher in silver content in 1985 than in 1984. This increase in grade was related back to the change in mining method and decrease in observed waste rock in the muck haul.

The VCR stopes have not been pulled completely empty, and considerable spillage from the S.W. Stope into the S.W. VCR Stope occurred, which prevents any ore reserve performance study from being made at this time. This study will have to be performed after the S.W. Stope has been pulled empty, because no reliable estimate exists as to the exact volume of ore mined in the VCR stopes. In the future, it is felt that the percentage of dilution calculated for the VCR stopes will be a very reasonable number

The Sixteen-to-One Mine operations were suspended in June 1986 due to low silver prices. When operations resume, the application of Modified VCR Stoping to narrow vein mining in the Colorado Vein, where the minimum mining width is being established at 1.8m (6 ft.), will be made. The Colorado Vein has already been drilled with rows of one and two holes per row, with down-hole diameters of both 165mm (6.5 in.) and 114mm (4.5 in.), and with up-holes of 108mm (4.25 in.) diameter. The results will be interesting and eagerly awaited.

Acknowledgments

Special thanks to Allan Young, Resident Manager, for permission to publish this report, and to Don Earnest, Chief Geologist, and Don Ryan, Mine Geologist, for the collection of data which made this paper possible.

References

Earnest, D.F. 1983, "Geology of the Sixteen-to-One Mine, Esmeralda County, Nevada," Gold and Silver Deposits of the Basin and Range Province, Western U.S.A., Vol XV, 1984, Arizona Geological Society Digest, pp. 101-108.

Taule, J.R., 1986, "Longhole Loading and Blasting Procedures Used by Sunshine Mining Company, 16-to-1 Mine," Sunshine Mining Company, Silver Peak, Nevada.

Young, A.R., 1983, "The 16-to-1 Mine," An In-Depth Study of 5 New Silver and Gold Mines, Northwest Mining Association.

1 - Tamrock Twin Boom Minematic Jumbo w/Tamock HLR-438 Drills. Drills 44.5mm (1.75 in.) Holes.

2 - TRW Mission Track Mounted 6200U Down-the-Hole Hammer Drill Rigs. Drills 165mm (6.5 in.) and 114mm (4.5 in.) Down-Holes, and 108mm (4.25 in.) Up-Holes.

1 - 3.8m^3 (5 yd^3) Wagner ST-5A LHD.

1 - 3.8m^3 (5 yd^3) Wagner ST-5H LHD.

2 - 2.7m^3 (3.5 yd^3) Wagner ST-3½ LHD's.

2 - 22.7 MT (25 ST) Wagner MT 425-30 Underground Trucks.

2 - 12.7 MT (14 ST) Wagner MT 414-30 Underground Trucks.

1 - Getman Exposive Truck.

1 - Getman Service Truck.

1 - 0.8m^3 (1 yd^3) Wagner HST-1A LHD.

1 - Longyear 38 EHS Diamond Drill.

Table 1. Mine Equipment List

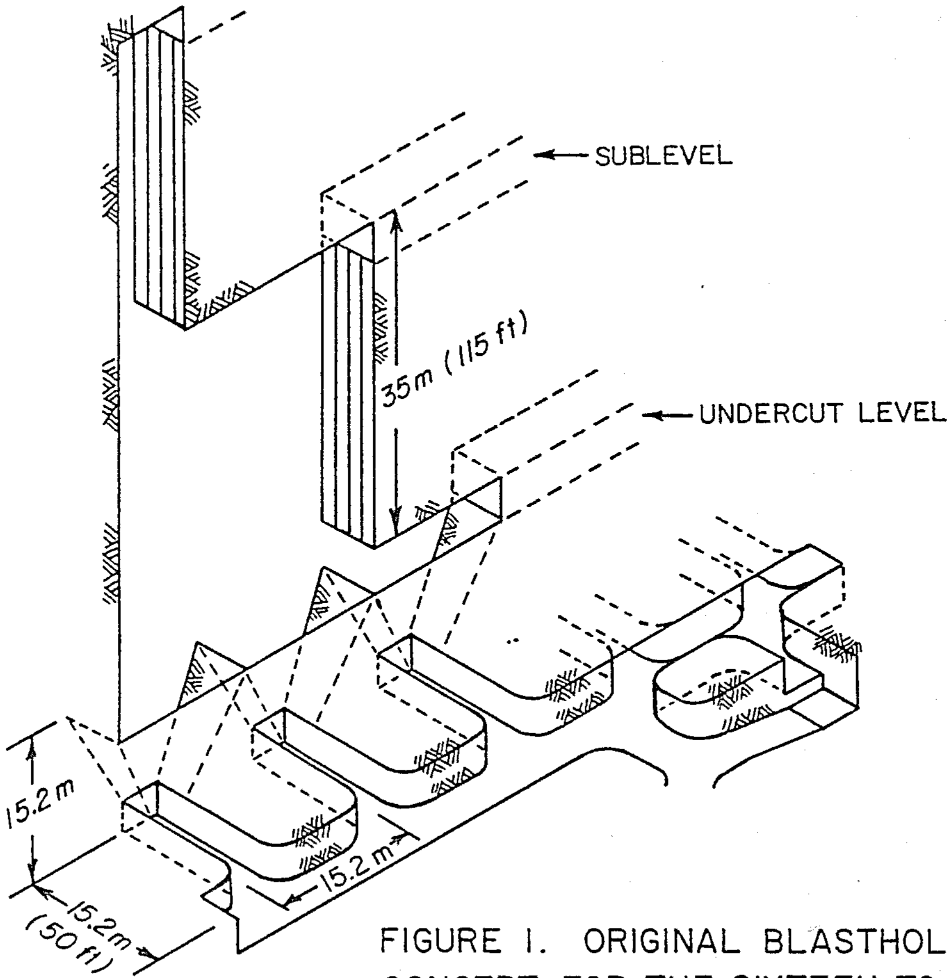

FIGURE 1. ORIGINAL BLASTHOLE STOPING CONCEPT FOR THE SIXTEEN-TO-ONE MINE.

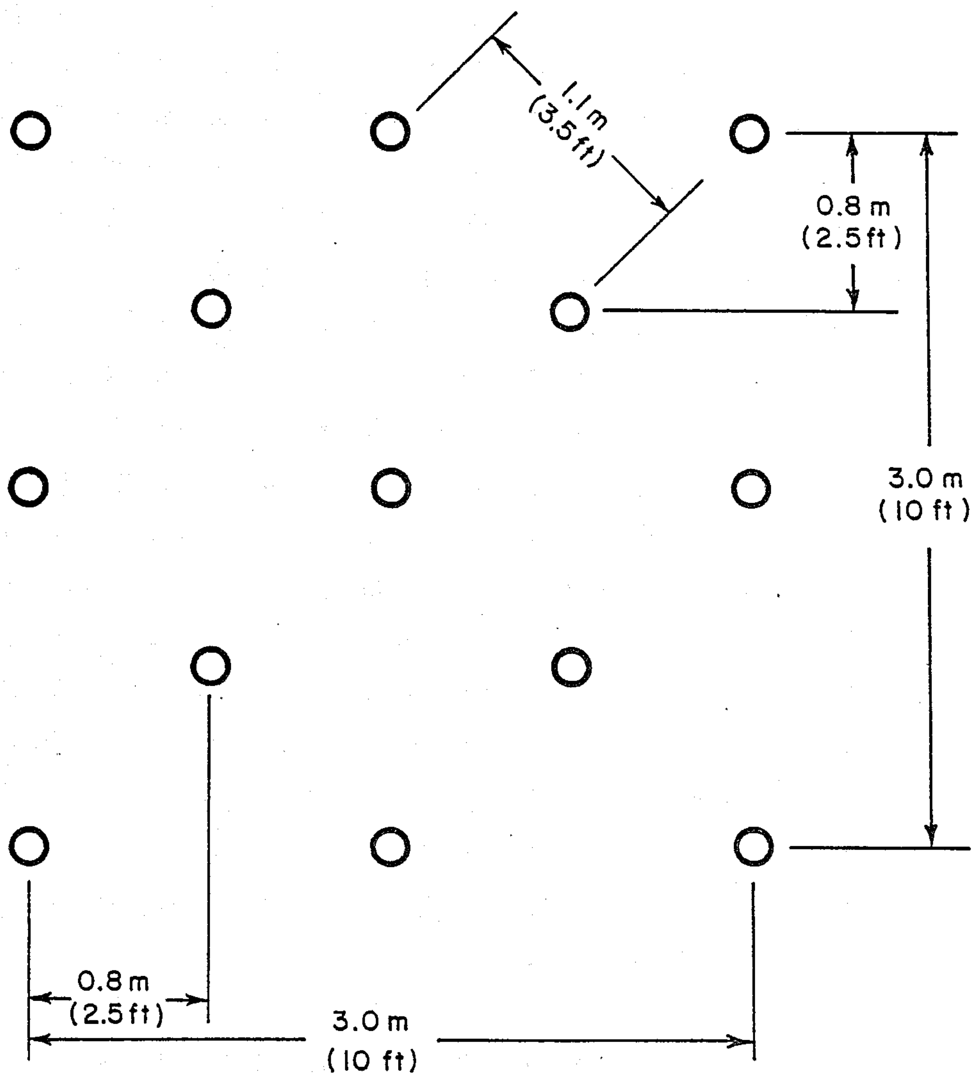

FIGURE 2. TYPICAL SLOT RAISE PATTERN

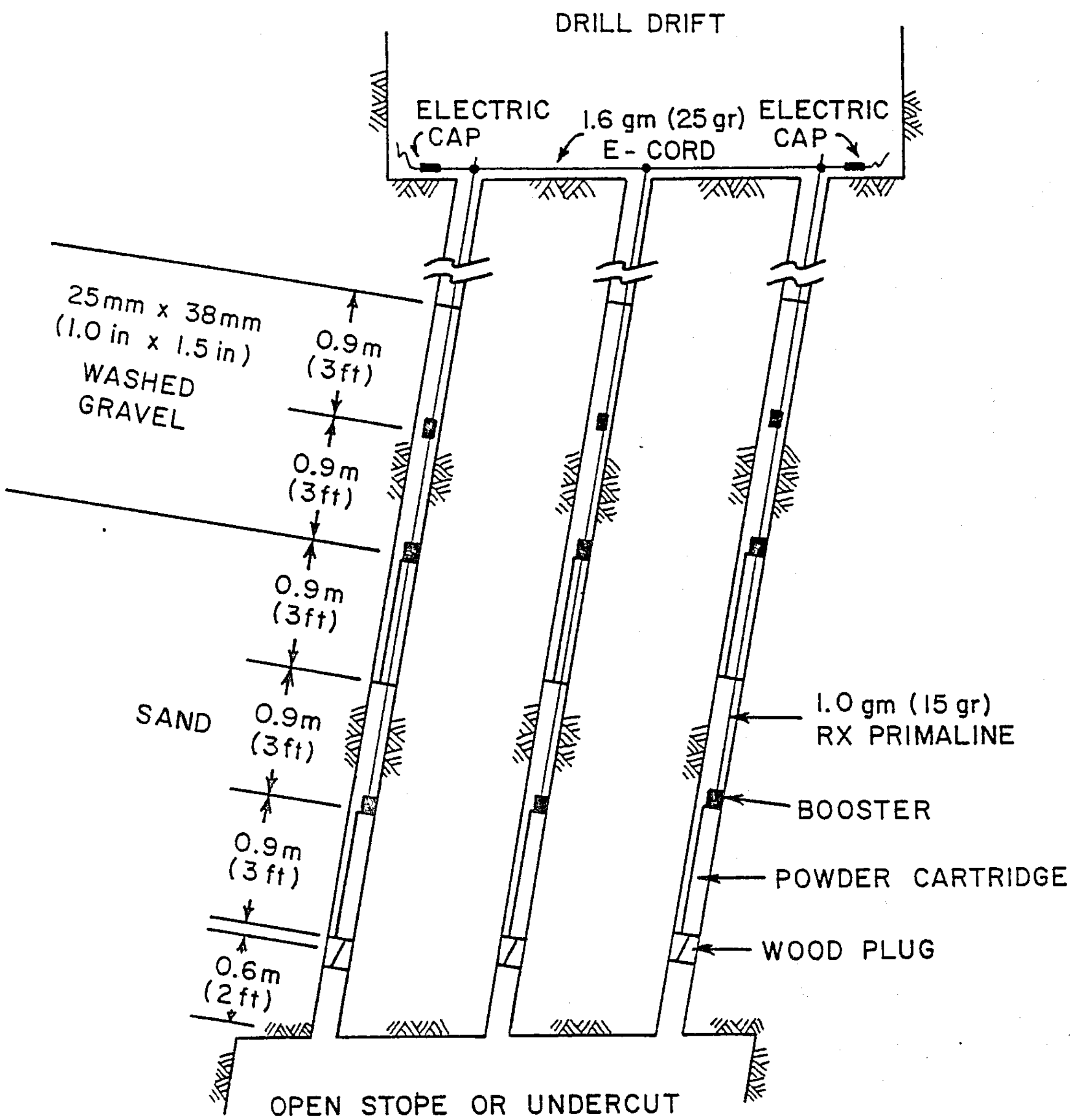

FIGURE 3. TYPICAL MODIFIED VCR LOADING PATTERN

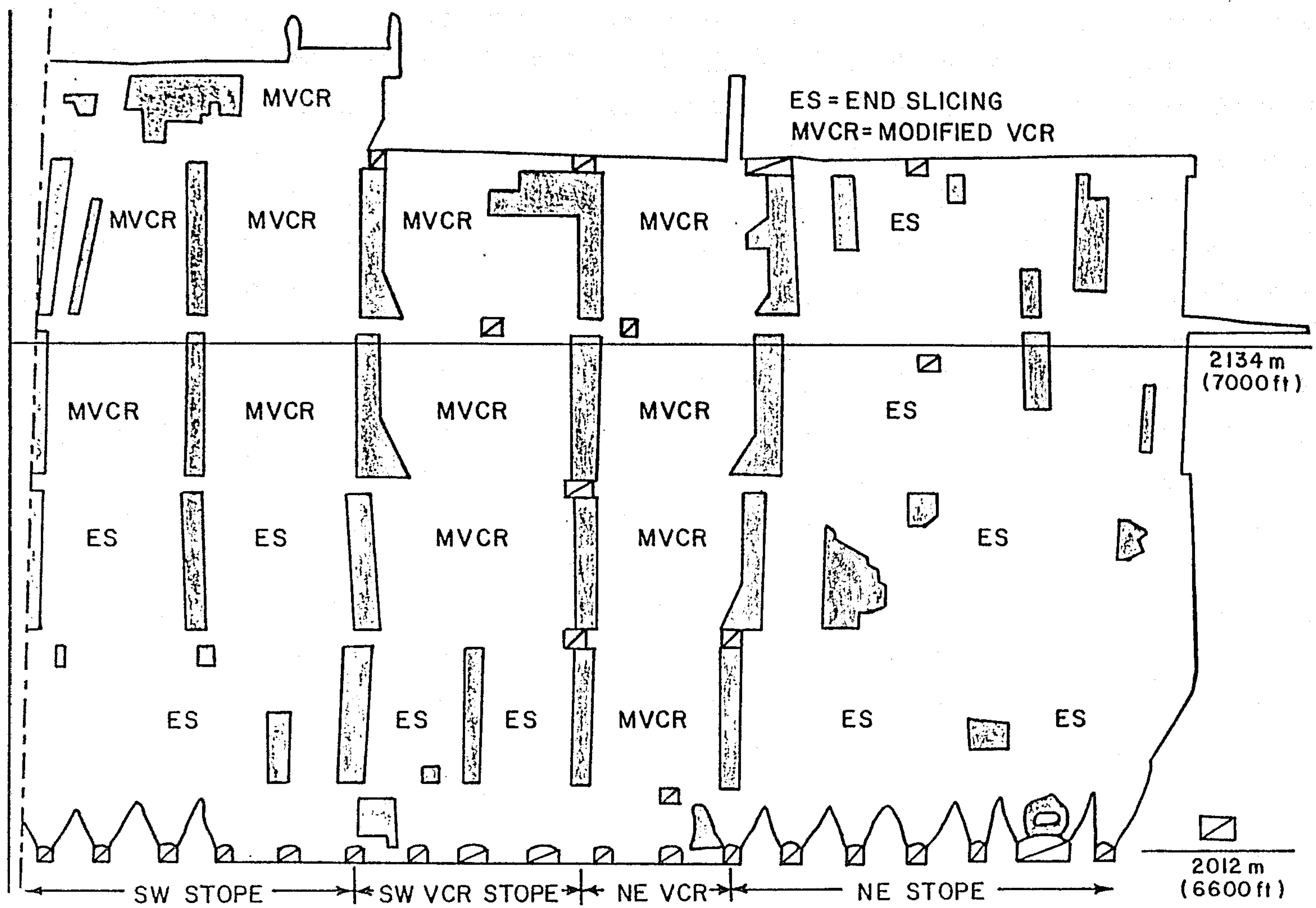

FIGURE 4A. VERTICAL LONGITUDINAL SECTION OF THE SIXTEEN-TO-ONE MINE, NE SECTION

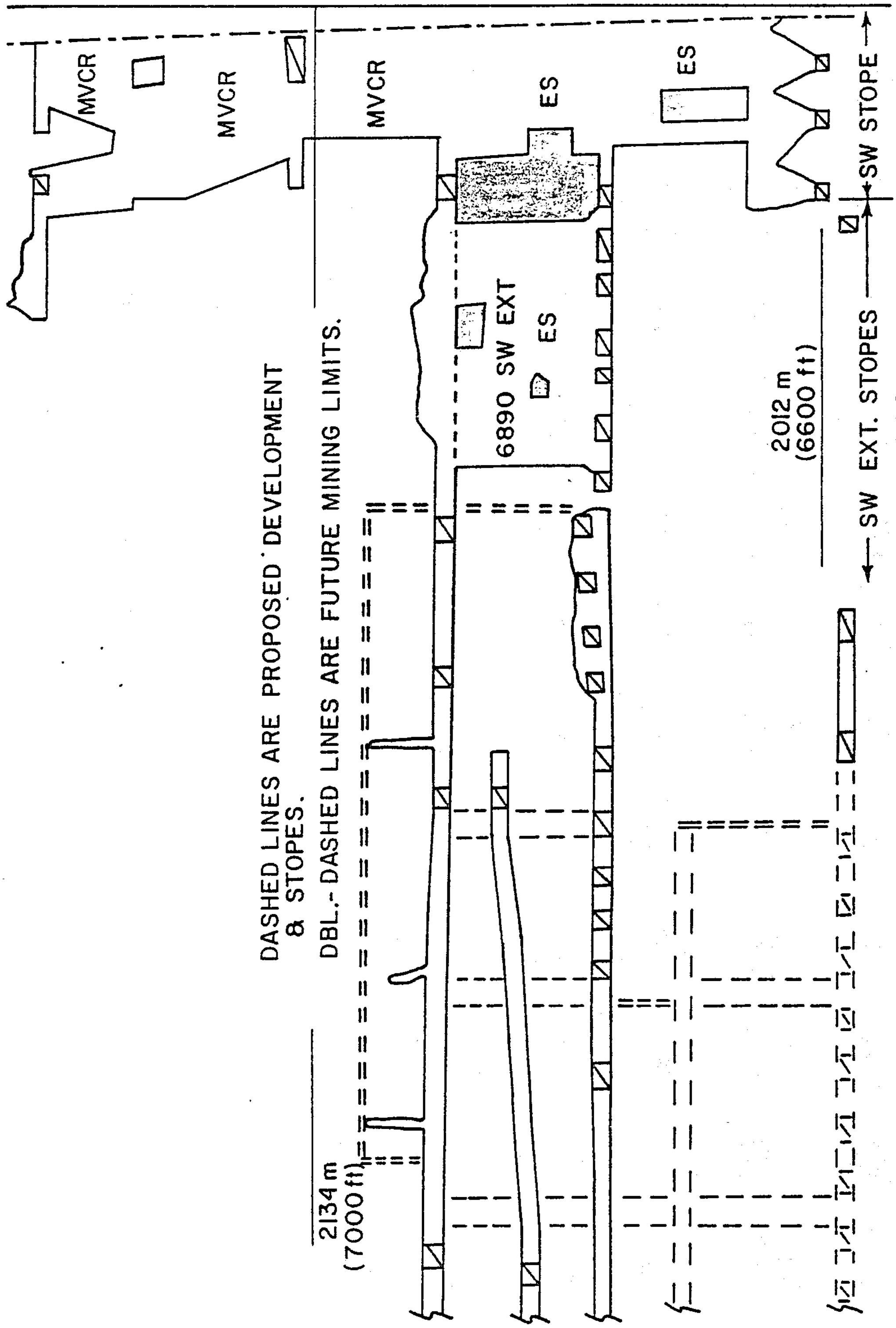

FIGURE 4B. VERTICAL LONGITUDINAL SECTION OF THE SIXTEEN-TO-ONE MINE, SW SECTION

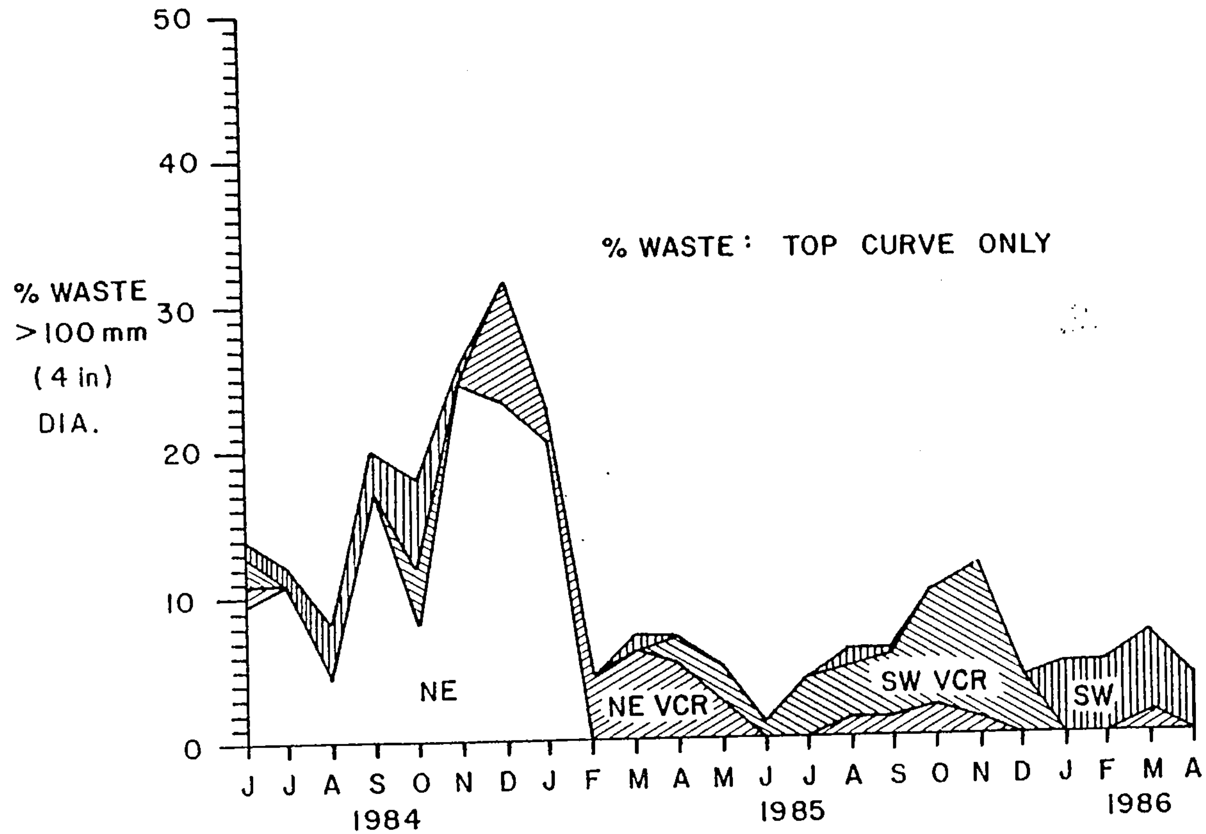

FIGURE 5. STOPE DILUTION OBSERVED AT DRAWPOINTS