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FINAL GEOLOGY REPORT ON VICTORIA MINE

As of 1 September, 1977, the Victoria Operation ended active ore production. The following report is a summation of the major areas of responsibility assigned to the geologic staff at Victoria. The various sections of this report were chosen either for their value to future operations at Victoria or for their value to future operations on other Anaconda projects. There are numerous other reports on such subjects as bismuthinite, swell factors, fan drilling and others in the Victoria file. Included in the Appendix at the end of this report are two sets drilling recommendations for the purpose of expanding the Victoria reserves. The main difference between the two programs is that the earlier uses the old underground drifts; the latter, driftwork that had been proposed for present operations. All plans and sections for both projects are in the Victoria file.

Jerome H. Kaczmarowski

Jerome H. Kaczmarowski
Chief Mine Geologist
Victoria Operations

<u>SECTION INDEX</u>

<u>APPENDIX</u>

GEOLOGY

INTRODUCTION

The Victoria orebody is essentially an arcuate chalcopyrite-pyrite mineralized zone in a breccia pipe surrounded by and composed of Pequop formation sediments of Permian age. Metamorphism has left an overprint on the sediments both within and without the pipe. Structural features such as faults and waste blocks are prevalent and their effect on the location of ore is very important.

GEOLOGIC SETTING

The country rock and the breccia within the pipe are composed of shallow sea sediments. The sedimentary sequence consists of limestone and dolomite with variable sand content, grading into pure sandstone. Most of the sediments outside the pipe are very fine-grained and dense and vary in color from buff to dark gray, with light gray as the predominant color. Fetid limestones are not uncommon outside the pipe; but become less common as one approaches the pipe boundaries and are absent within the pipe.

Two dikes, apparently emplaced along pre-existing fault zones in the Permian sediments, extend into the breccia pipe. Both are extremely altered, with biotite replaced by pyrite and silicate minerals replacing the majority of the feldspars. In good specimens, it is apparent that both dikes are generally equigranular. Though obviously younger than Permian in age, a more precise date for the dikes is not feasible at this time. The exact age relationship to the minerallization is also uncertain at this time; because, although the dikes show extreme alteration, no copper values have been found in either dike.

The bedded sediments surrounding the Victoria orebody have strikes trending from N. 34° E to almost due east. Dips range from 0° to 34° south to southeast, with an average dip of approximately 20° southwest. Numerous minor faults show movements of 0.1-2 meters, but most of these appear to be adjustment faults and of little importance to the overall structural picture.

Figure 1 is a plan view of the 7110' level and shows a summary of all major structural features and the ore outline. The arcuate shape of the orebody in the breccia pipe is evident from the plan, as are the locations of the major faults.

Fault A on the plan has the greatest known displacement of any fault in the immediate mine area. Based on assay information (the displacement of the north ore tail), Fault A has 50 feet of apparent left-slip movement. This fault is important for several reasons:

1. The left-slip movement has displaced the north ore tail to the northwest and all the early drill holes passed over the top of the north tail. DDH-300-23 intersected this tail through a low grade section. A recommendation was made to drill the north ore tail from the 7180' level access drift. This recommendation was not followed because of the need to keep the mill supplied with ore. This low-grade tail is now unminable until the end of planned mining operations.

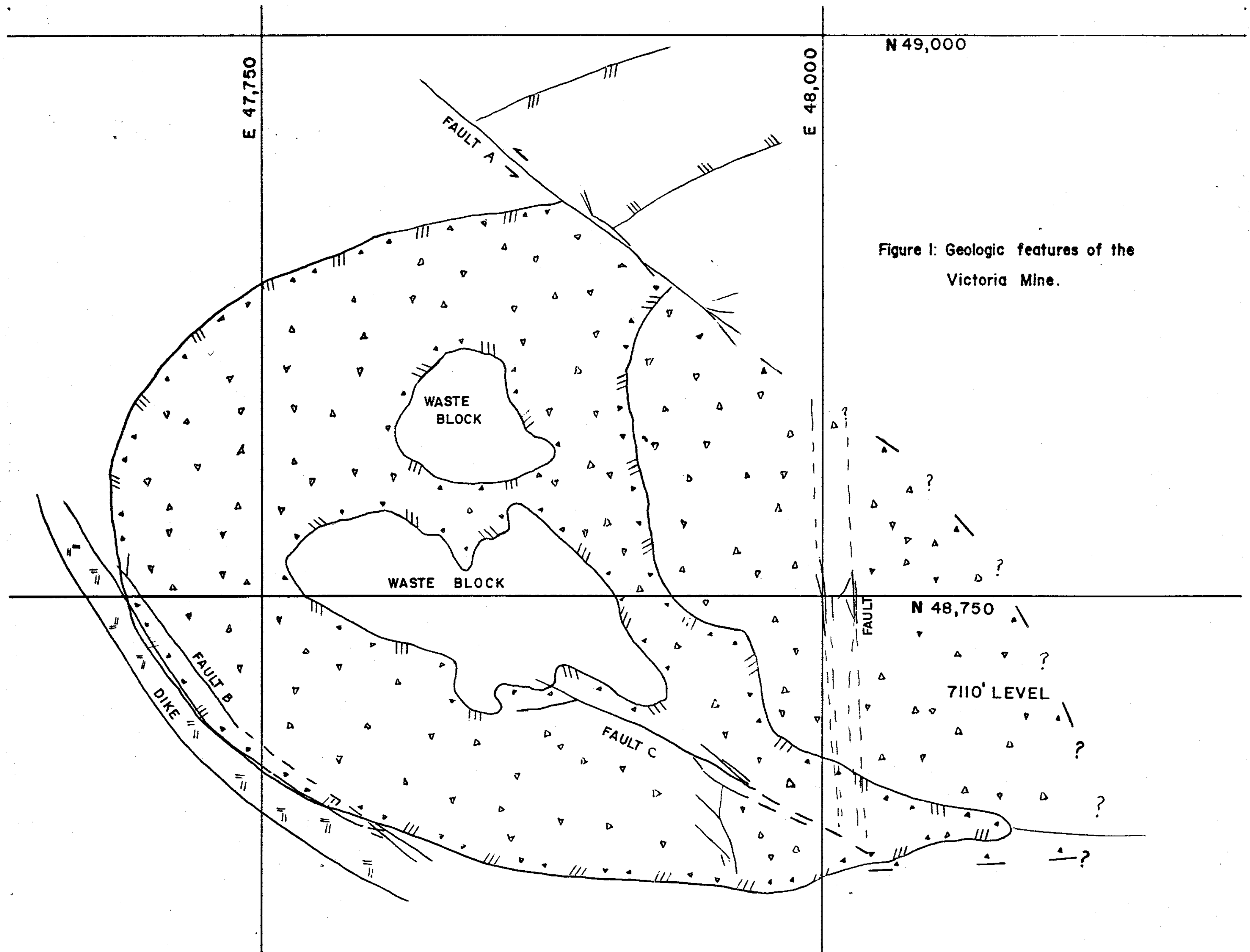
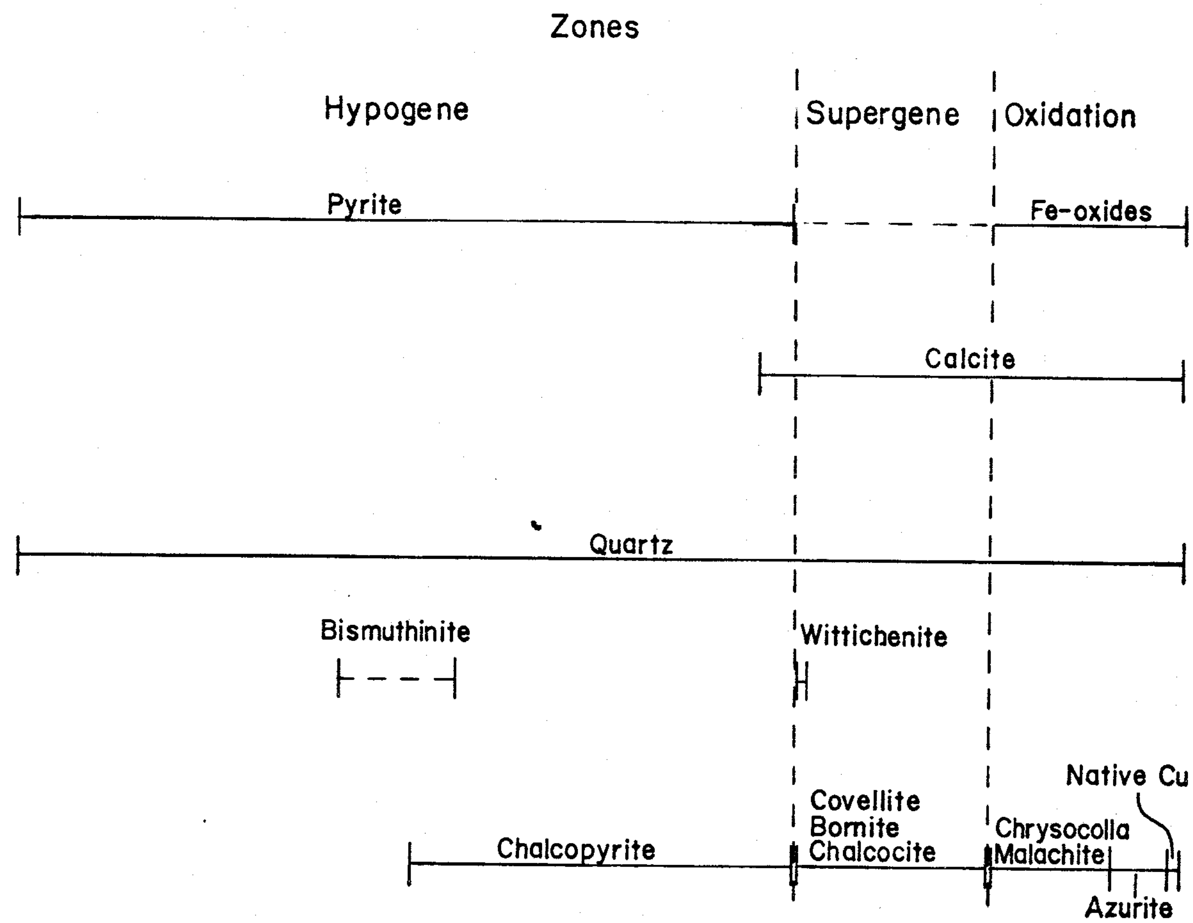


Figure I: Geologic features of the Victoria Mine.

Figure 2: Paragenetic Sequence for Victoria Mine.



2. Fault A has also caused stability problems. During the underground planning stages, curved faults (Fault A falls in this category) were projected to be the worst stability problems. This has indeed been the case at both ends of the access drifts on all completed levels.

Fault B, which is composite of several similar faults, shows similarities to fault A in trends and stability. Movement on this fault is uncertain.

Fault C is an important fault from the standpoint of genesis of the deposit in that its abrupt truncation at the edge of a waste block may indicate a prior system of faults and joints that show the initial plumbing for the Victoria breccia.

The last structural feature of major importance is the breccia itself. Fragments range from essentially clay-sized particles to blocks 170 feet long and +50 feet thick (the waste blocks are described in greater detail below). Most of the finer breccia consists of fragments of limestone and quartz (either vein or quartzite) mixed with montmorillonite-type clays in varying proportions. Diopside is frequently present in breccia fragments of any size. The main mine sections even at the shallow levels that present underground operations have attained seem to show that it might be possible to construct a plumbing system based on the assumption that fine breccia zones are more porous than course breccia zones and therefore more amenable to solution migration. However, more information deeper in the deposit is necessary before this assumption is verified.

Figure 1 includes three of the major waste blocks on the 7110' level. The various waste blocks are important for the following reasons:

1. The low-grade material found in the waste blocks causes problems in the mining operations in that the lack of continuity in ore grade causes problems in planning and also means drilling, blasting, and mucking that does not add to economy of operation. This results from the fact that large zones of unblasted material cannot be left behind intact with the type of mining method used at Victoria (sublevel caving).
2. Although the waste blocks are traceable from drift to drift both vertically and horizontally, drifts driven in them tend to be slabby and dangerous because of clay seams between the bedding planes.

METAMORPHISM

Metamorphic effects in the country rock adjacent to the pipe include an irregular, minor tremolite zone surrounding an inner diopside zone present in the pipe. The diopside zone extends beyond pipe boundaries along favorable beds for at least 500 feet. Saponite, a montmorillonoid clay, is a fairly common alteration product of diopside to at least the 7180' elevation in the pit (average surface elevation prior to the initiation of mining operations was approximately 7475'). Montmorillonoid clays occur along bedding planes and clay-filled joints, and are integral ingredients in most faults. Saponite was not apparent in underground operations below the 7180' elevation. Silicification is also common in the orebody and rocks adjacent to the orebody, at times so pervasive that the original rock types are in doubt. Talc is also a fairly common mineral but is usually mixed with larger amounts of white montmorillonoid clays. Sepiolite was also fairly common along fractures and in some bedding planes in the upper levels of the pit in country rock adjacent to the orebody and was usually related to faulting. Chlorite is a less common mineral in the Victoria orebody and occurs as a minor constituent with some montmorillonite-type clays. Andradite garnet occurs sporadically as a vein-filling between breccia fragments in the upper portions of the orebody, but deep drilling below the deposit indicates massive garnetite below the 6300' level. Diopside occurs as opaque white to pale green separate crystals (0.1-0.5 mm) in limy rocks, as creamy alteration haloes on joints in limy rocks, and as fibrous veinlets up to ¼" wide, intergrown with calcite. Allanite, a hydrated rare earth silicate, occurs as separate, distinct grains (W. Atkinson, early core loggings); but is very locallized and was not usually identified in underground mapping.

ECONOMIC MINERALLIZATION

As mentioned above, the Victoria ore zone is essentially a chalcopyrite-pyrite orebody. The only economic hypogene mineral is chalcopyrite, which occurs as open-space fillings (widths range fron 1.01-2 meters), as the cementing agent in fine breccia zones, and as replacements along bedding planes in larger breccia fragments.

Silver is also present in the deposit and follows copper values almost directly. Assay contouring of at least four pit levels resulted in nearly identical silver and copper contour maps. There is no evidence of any vertical or horizontal zoning in the silver values. A polished section study to determine the silver mineralogy failed to identify silver minerals. Although it is not possible to set up a series of curves relating copper and silver assays, there is a gross correlation as shown below:

COPPER	SILVER
0.0% - 1.50%	< 0.3 oz./ton
1.51%- 2.50%	0.3-0.6 oz./ton
> 2.50%	> .06 oz./ton

A supergene zone occurs at the top of the main sulfide mineralization. This zone is not as well-developed as the zone of oxidation. The upper pit levels (especially the 7250' levels) were appreciably higher in grade than the rest of the deposit, and supergene enrichment may be the explanation. However, the chalcocite-covellite-bornite coatings on chalcopyrite on the high pit levels never exceeded 10% of the total copper sulfides, so I believe a more likely explanation is that a decrease in pressure and/or temperature led to massive precipitation in the upper levels of the deposit.

The effects of supergene enrichment locally extend to at least the 7110' elevation. This occurs in conjunction with a few major joints and faults.

A well-developed zone of oxidation has formed near the top of the Victoria orebody. It is offset 200 feet to the east of the top of the sulfide mineralization. Mining activity from 1872 to 1883 removed most of the "oxide" copper high grade, but present operations have stockpiled approximately 60,000 tons of the acid-soluble portion is in the form of chrysocolla and malachite, (finely-laminated and amorphous), with appreciably lesser amounts of azurite (occurs as individual crystals up to ¼" in length), cuprite (very fine veinlets usually only visible under a binocular microscope), and native copper (found only in one isolated spot in south tail of orebody-see Figure 1). Very minor amounts of sooty chalcocite and trace covellite account for the remaining non-acid soluble portion of the oxide zone.

GANGUE MINERALLIZATION

The main gangue minerals associated with the Victoria orebody in order of decreasing abundance are: pyrite, quartz, calcite, specular hematite, bismuthinite, and wittichenite. As would be expected, iron oxides such as limonite are very abundant in the upper levels of the deposit down to approximately the 7275' level. Pyrite in this zone is frequently oxidized and at times only remnant pyrite remains. There is also more quartz veining and jasperiodal quartz in this part of the deposit than in levels below.

From the 7275' elevation to approximately the 7100' elevation, the amount of oxidized pyrite and iron oxide decrease until both are essentially only local occurrences near faults and/or large-scale continuous joints. There is also slightly more specular hematite in this zone. Quartz veining also decreases slightly and calcite veining becomes more apparent, although the quartz veining is still more abundant than the calcite veining. Jasperoidal quartz is not present.

Below the 7100' elevation, old underground information indicates conditions and occurrences to be similar to those found on the 7100' elevation.

Bismuthinite is found in large crystals (some up to 20 cm. in length) in patches throughout the ore deposit and apparently shows no real zoning. Wittichenite was observed only in polished sections made from heavy mineral separates of blast hole samples high in bismuth, and occurred as veinlets 40 to 50 microns wide in bismuthinite.

PARAGENESIS

Figure 2 shows the paragenetic sequence which summarized the mineralogical information described above. The majority of the data is from my mapping and hand specimen study.

GENESIS

Based on the evidence seen in both pit and underground mapping, I believe the Victoria orebody to be the result of solution collapse with no explosive activity involved in the process. The large waste blocks would seem to indicate quiet, slow collapse, because explosive activity should have broken them apart. Because of the number of faults and their location (the breccia pipe is completely surrounded by faults and where rotation or waste blocks have not cut them off, they can be traced through breccia zones), it is my opinion that the faults and associated jointing were the initial plumbing system for the early solutions that dissolved rock and brought about further brecciation. This resulted in a good host for copper mineralization as is witnessed by both large, open joint fillings and finer veinlets in shattered, larger blocks. The lack of mineralization in larger competent waste blocks also supports this idea, because the beds in the blocks are generally composed of fairly fresh, aphanitic to very fine-grained limestones which would inhibit solution migration and therefore copper minerallization.

The actual source of the copper mineralization remains uncertain. Nothing in my mine mapping has shown any evidence for or against the monzonite stock northwest of Victoria Mine being the source, nor has any other source presented itself.

MAPS

All geologic plans and sections have been moved out of the Victoria offices.

RESERVES

A summary of remaining underground ore reserves for the Victoria Mine is listed below. Reserves contained in the 7143'-7179' level block consist of developed ore, most of which has been fan drilled. Ore in the 7107'-7143' block has the necessary development drifwork, but has little or no fan drilling.

UNDERGROUND RESERVE SUMMARY
- BY LEVEL -

	PROVEN		PROBABLE	
LEVEL	TONS	GRADE (%Cu)	TONS	GRADE (%Cu)
7143-7179	8,000 t	2.20		
7107-7143	180,060 t	2.07		
7071-7107	175,734 t	2.38		
7035-7071	150,374 t	2.42	6,219 t	1.97
6999-7035	128,336 t	2.31	15,769 t	2.09
6963-6999	113,022 t	2.13	45,254 t	3.42
6927-6963	138,208 t	2.47	16,342 t	3.42
6891-6927	156,442 t	2.57		
6855-6891	168,439 t	2.45		
6819-6855	176,456 t	2.32		
6786-6819	96,128 t	2.29	64,799 t	1.81
G. TOTAL	1,491,200 t	2.34	148,383 t	2.51

This summary does not include 60,000+ stockpiled tons of 0.94% soluble copper or the 30,000 tons of 0.5% copper in stockpile 1 (this stockpile also contains 0.34% acid soluble copper).

TONNAGE AND GRADE CONTROL

PIT

As of February, 1976, the Geology Department was given control of tonnage and grade for the Victoria Mine. At that time it was recognized that a problem existed in both tonnage and grade reports from the mine, since neither was close to weightometer and assay data supplied by the concentrator.

The first step in correcting these problems was to compare past mine tonnages with concentrator tonnages. The result of this comparison indicated that concentrator tallies were only 81% of mine tallies. Mine tallies were then multiplied by 0.81 and the remaining pit ore tallies- from February, 1976 to July, 1976- fit within 2% of this concentrator tally.

The next step was to determine the source of the grade difference. A check of the blast hole sampling procedures indicated that operators and samplers did not understand either the method or the importance of correct sampling. Having corrected these faults, assays were rejected which were obviously too high (any assay that was greater than 8% copper) while discarding an equal number of the lowest assays in each ore block. This resulted in agreement between mine and concentrator grades. The scientific validity for this approach was never totally proven, but it was successful and therefore of use from an operational standpoint.

In March 1976, it was recommended that the tonnages and grades in stockpiles 1 through 4 be downgraded. The basis for the tonnage decrease was the above-mentioned historical figures. The grade decrease basis was a close observation of mill assays when particular stockpiles were run through the concentrator without any blended additional ore from other sources. The recommendation involved a 20% decrease in tons and the following grade cuts:

STOCKPILE #	PRIOR GRADE	SUGGESTED GRADE
1	0.92% Cu	0.5% Cu
2	1.86% Cu	1.3% Cu
4*	3.46% Cu	3.13%Cu

* Stockpile #3 was essentially finished by then.

At that time, the recommendation was not followed.

In July, 1976, the remaining tons in the stockpiles were so depleted that it became visually obvious that a correction in the tonnages carried on the books was mandatory. At that time, 94,634 tons at a grade of 4.81% copper were dropped from the records. The need for this course of action was viewed with some disfavor and in September, 1976, M. A. McKinnon requested that I try to discover what had happened to cause the problem.

The results of this check are as follows:

1. Cubic foot per tons factors were regularly changed to reconcile survey tonnages and truck tally tonnages. Thus waste factors between 11 and 12 cubic feet per ton and ore factors between 10 and 11 cubic feet per ton were commonly used, when in reality, 11 cubic feet per ton should have consistently been used. The lack of consistency described above in the tonnage factor, of course, leads to more high grade tons, and that is probably the main reason for the high grade loss.

2. No attempt had been made to use historical figures as a reference, even though this is a simple expedient.
3. Supervision of the various aspects of the total project was faulty.

A more detailed, explanatory final report on the pit tonnages and grades is contained in the 28 October, 1976, report entitled, "Reported Vs Actual Tonnages and Grades." A copy of that report is included in Appendix of the report.

UNDERGROUND

In January 1976, the Victoria Staff began underground operations. By June, 1976, the first development ore was reached. From June through the end of July, the writer carried out sampling. At the end of July, one sampler (W. Egbert) was hired and in October another (J. Gamble). Both did a superlative job.

During the period that I was the only sampler, all sampling had to be muckpile grab samples. When the samplers were hired, they continued the muckpile samples and began chip wall samples. Both methods of sampling turned out to be satisfactory when compared with concentrator feed grade assays (variance was 0.30% or less for both methods).

During this period, reported tonnages also were done by the Geology Department. Drift (development) tonnages were determined by direct measurement. Production tonnages were based on truck tallies (15 tons for Caterpillar scrapers; 13 tons, for Elmac haul trucks).

Table 1 shows a comparison by month between received concentrator tonnages and grades and reported mine tonnages and grades.

TABLE 1. Monthly Concentrates vs Mine Reported Tonnages and Grades.

MONTH	CONCENTRATOR		MINE	
	Tons	% Cu	Tons	% Cu
NOVEMBER	20,993	2.02	19,728	2.09
DECEMBER	15,000	2.4	14,924	2.4
JANUARY	23,326	2.18	23,326	2.18
FEBRUARY	26,062	2.32	26,342	2.24
MARCH	25,613	1.61	26,112	1.69
APRIL	18,499	1.93	18,958	1.93
MAY	21,387	1.96	22,928	1.96
JUNE	24,394	2.13	22,994	2.13
JULY	31,282	1.86	31,882	1.87
TOTALS	206,556	2.027	207,194	2.032

PERCENT DEVIATION CONCENTRATOR VS MINE

1. tonnages = 0.31%
2. grades = 0.25%

Fan drill blast hole sampling was also tried, but lack of support from operations precluded any valid analysis of this type of sampling. A sample collector and splitter was designed and placed on at least one fan drill, but never received a reasonable test during actual operations.

PLANNED VS. PRODUCED TONNAGES AND GRADES

INTRODUCTION

This section explains the obvious differences between planned tonnages and grades. Table 1 shows tonnage extraction rates for the three completed levels. (The 7140' level is not totally mined, but is essentially done.)

TABLE 1. Tonnage extraction rates for underground levels already completed.

LEVEL	PRODUCTION ORE EXTRACTION RATE	PROD. & DEV. ORE EXTRACTION RATE
7220'	26.54%	45.26%
7180'	40.42%	50.49%
7140'	67.03%	71.58%

Table 2 shows the grade comparisons for the four developed levels.

TABLE 2. Grade Comparison Showing Projected vs. Produced Grades by Level.

LEVEL	PROJECTED GRADE*	PRODUCED GRADE
7220'	2.75% Cu**	2.55% Cu
7180'	2.75% Cu**	2.21% Cu
7140'	1.83% Cu	2.05% Cu
7110'	1.81% Cu	1.89% Cu***

*These grades have been decreased by 16% according to the mine plan (result of dilution) for production ore only.
**This grade was changed from the original 3.80% to 3.15% in July, 1976.
***This is not a completed level and the final grade should be appreciably higher.

A copy of the planned reserves for the underground operation is included on the following page for easy reference. Factors such as over-drawing of fans, tons of waste removed or left on the ore pad, drift hang-ups, the method of opening up drifts,and drifts incorrectly driven (badly off centerline) create problems in evaluating where losses actually occurred in the mining operations.

OVER-DRAWING OF FANS

This item is probably the most important factor in the reported low monthly mill heads. Samplers indicated to operations personnel that a drift had gone to waste and mucking should be stopped. On a number of occasions, these instructions were not followed. However, a more important factor in the low grade feed supplied was the fact that the Geology Department was allowed to hire only two samplers to cover 21 man-shifts per week. This resulted in numerous instances of over-mucking of production drifts, especially from 2:30 a.m. until 7:00 a.m. every morning and on weekends.

WASTE ON THE ORE PAD

The over-mucking mentioned above resulted in many thousands of tons of waste being dumped on the ore pad. Waste was routinely removed from the ore pad, but it was never possible to remove all of it. Operations personnel directly involved were informed repeatedly concerning this problem, but it was never corrected.

DRIFT HANG-UPS

Smooth caving is important in supplying consistent grade to the concentrator. A hang-up in a drift is a situation in which the ore fails to cave evenly- or not at all- at any point in the mucking cycle. Reasons for hang-ups are as follows:

1. Insufficient weight above the block being mined. This sort of hang-up was especially prevalent on the 7220' and 7180' levels. Fans would shoot quite well, but would form an arch very early in the drawing of ore. In many cases the only solution was to continue to shoot fans until the void formed by arching could not stand. This, however, resulted in many tons of ore being left in an unmineable position.
2. Insufficient number of previously loaded fans. When this situation occurred, the shock of close-proximity blasting (within 5 feet of the fan to be loaded) usually caved the holes in the fan to be loaded and prevented a clean blast on that fan.
3. Moisture-induced decomposition of explosive in loaded holes. If holes in wet fans were allowed to sit for more than a day, hang-ups could occur. This happened on the 7180' level at least 3 or 4 times.
4. Tying into the wrong ring. This happened in drifts 2 and 4 on the 7140' level. In the latter case, we lost 55 to 60 feet of good grade ore (5,000 tons at +2.0% copper) in drift 4 from resulting bootlegs (fans will adequately pull 5 feet of burden but will not pull 10 feet of burden).

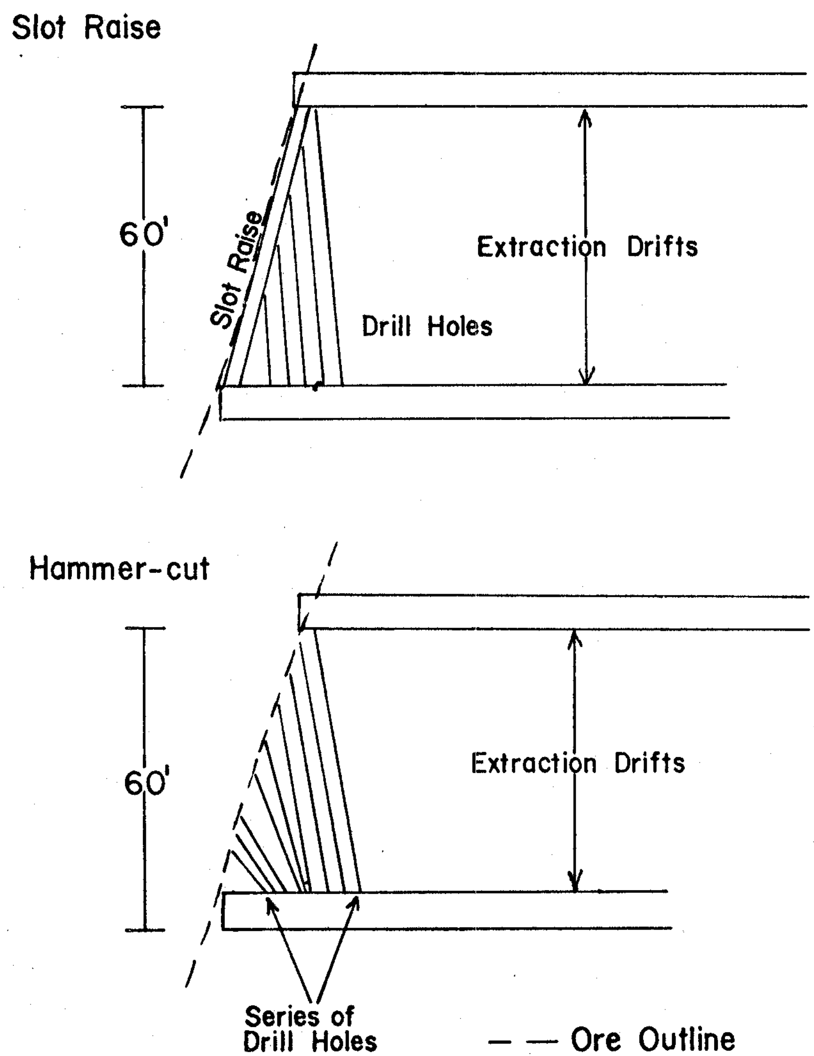
METHODS OF INDUCING CAVING IN PRODUCTION DRIFTS

Caving was induced in production drifts using two methods: slot raises and hammer cuts. Both methods have good and bad points. Figure 3 shows schematic representations of the two methods.

Slot raises are the most efficient from the standpoint of ore extraction. They can be placed very accurately on the ore-waste contact and little or no ore is left behind in an unmineable position. The main problem with their use is that they are time-consuming and more expensive than hammer cuts. Very few of these were used in the Victoria Mine, because of the constant need for ore to keep the concentrator running.

Hammer cuts were generally successful within their design limits, they are quicker than slot raises, and easier and safer to do when bad ground

Figure 3: Sections showing the two methods of initiating underground ore production.



is encountered. The main drawback with hammer cuts is that they result in trapped ore at the ends of drifts. On the 7140' level, this trapped ore was between 15% and 20% of the total level reserves. Judging from ore trapped on the 7180' level, recovery of trapped ore on the level below usually results in such strong dilution with pit waste that recovery is not economic.

MISPLACED DRIFTS

There are two examples of this type of error. On the 7220' level, drift 1 was placed to the north of its planned location, even though sections showed that the ore contact would be close to the edge of a correctly-located drift. This misplacement resulted in drift 1 being unmineable.

On the 7110' level, drift 8 was driven too far to the south of its proper, planned location and is outside of the ore boundary. It is therefore useless for mining purposes.

CONCLUSION

It is my opinion that the Victoria ore projections have been very accurate. All levels have proved to contain more available tons than projected and slightly better overall grade than expected. The one major change I instituted was the downgrading of DDH-56, which is the hole that pushed the northwestern block copper grade to an unrealistic 3.80% copper. Comparison with blast hole data in levels mined by the pit indicated that DDH-56 was 1.10% to 1.35% above what was actually found in the block, because of an intersection in a localized high grade area only 10 feet across. This change first occurs in the Underground Reserve Report of 21 July, 1976; however, it was discussed previously in February, 1976.

SLOPE STABILITY

EARLIER WORK

Prior to October, 1974, Anaconda's Mining Research Department studied the Victoria stability problem, but a cutback in that month ended the stability design program. In January, 1975, Seegmiller Associates made ad new Stability study. A summary of their conclusions is shown below:

1. The overall stability of the pit should be good.
2. The north slope area from the 7300' to 7425' levels is likely to have a plane failure along bedding planes. Additionally, toppling failure is possible.
3. Plane and/or toppling failures are possible both east and west of the Victoria Shaft. The North, Southeast, and Southwest areas in the pit are possibly sites of minor toppling failure.

In June, 1975, a fatality resulted from a combination toppling/plane failure on the North 7575' bench. The following day, the Victoria Geology department was given responsibility for the pit slope stability program.

JULY 1975 TO JULY 1976

Following the fatal accident, Seegmiller Associates instructed mine personnel on the use and maintenance of various slope monitoring devices such as quadrilaterals, blinkers, and the HP3800 distancemeter system. The location of the various monitoring devices is shown on Figure 1, Stability Plan Map (this plan is in the Victoria Structural File).* Shortly thereafter, E. O. MacAlister recommended that seismic holes be drilled at Victoria. The writer located four holes as far from known faults as possible, in order to obtain mineralogical and seismic information as unaffected by locallized conditions as possible. Velocity scans from the holes were plotted next to a log of each hole (see Figure 2, Slope Stability Section, also in the Victoria Structural File) and zones of weakness were inferred from correlation of clay layers and low-v elocity sections in the various holes. As indicated on Figure 2, there were six major and two minor zones of possible failure.

On 30, December, 1975, a minor failure occurred in the northwest corner of the pit during which a portion of the 7375' bench slid into the pit. The block that failed was probably pushed by the entire north slope above it. The bottom of the failure was also the weakest bedding plane in Major Failure Zone A. At that time the writer indicated to M. A. McKinnon that a major failure was possible during the months of February and April for the following reasons:

1. During February, daily thawing and freezing of the snow cover lead to daily addition of water. This water flowed down into cracks and acted as additional lubrication on the already clay-lubricated failure plane.
2. Rain and the final melting of remaining snow cover in the month of April have the same affect as the melting.

* Figure numbers in this section are not consecutive in order that the main slope stability report may be used with greater ease.

In addition, small, apparently isolated cracks began appearing in various places in the pit. These led to a picture of greater and greater strain in the northeast corner of the pit on the northeast corner of the pit on the 7375' level.

Velocity measurements from 20 February, 1976, to 25 February, 1976, indicated an increase in the north pit slope speed, with the slope of the velocity curve becoming vertical (failure) between the 26th and 29th of February, 1976.

On the 26th, radios were issued to the loader operator in the bottom of the pit and to an observer in the northeast corner of the pit. At approximately 10:30 a.m., the pit was vacated. From 11:30 a.m. until 4:30 p.m., a major failure occurred in which the north slope moved a minimum of 34 feet.

This failure was a classic bedding plane failure. The strike of the beds in the north slope was such that the moving block formed a miniature thrust fault (labelled "Pushed-up Area" or Figure 5, Stability Diagram) in the northeast corner of the north pit wall. As soon as the block had pulled away from the northwest-trending dike (see Stability Diagram), the buttressing effect provided by the dike was lost; and the entire north slope began to rotate into the pit. This continued until enough of the block was broken up to provide additional friction at the toe of the slide.

Another failure occured 16, April, 1976. This failure was probably entirely a result of additional lubrication from run-off into the cracks of the north slope. Operations again were halted safely with no loss of life or damage to equipment. Minimum movement during this failure was 32 feet.

A more detailed explanation of the slope evaluation is found in the report of 19 September, 1975, entitled, "Slope Stability Information of the North and Northwest Pit Areas." A copy of this report is included in the appendix at the end of this report. All originals for this section of the Victoria Final Report are located in the Victoria Structural File.

THE ANACONDA COMPANY
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Victoria Mine
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INTER-COMPANY CORRESPONDENCE

To: M. A. McKINNON Date: OCTOBER 28, 1976

From: J. H. KACZMAROWSKI Subject: REPORTED VS ACTUAL TONNAGES AND GRADES

As requested, I have reviewed all available data to determine where tonnage and grade estimates differ from actual tonnages and grades as determined by weightometer and assay information supplied by Steve Stillar, Victoria Mine Concentrator Superintendent. I also re-planimetered every pit bench to determine how much ore should have been reported. Table 1 shows a comparison between the two methods of obtaining total pit ore tonnage.

Table 1: Total pit ore tonnage.

Mill Estimate	Planimetry Estimate
606,756 Tons (From start-up until 10/18/76)	627,155 Tons (7375 level to 7188 level)
+54,000 Tons (Stockpile survey on 10/18/76)	-15,000 Tons (Rough estimates of boulders thrown away)
-34,541 Tons (Underground ore through mill up to 10/18/76)	+21,960 Tons *(Old stockpiles)
626,215 Tons (Equals total tonnage)	634,115 Tons (Equals total tonnage)

Difference = 7900 Tons (1.26% of total tonnage.)

* Actual tonnage = 27,111 tons downgraded by 19% (factor obtained historical records) to obtain 21,960 tons.

Table 2 indicates tonnages reported in monthly reports.

Table 2.

Month	1974	1975	1976
January	-	90,157 T	9,800 T
February	-	55,931 T	11,165 T
March	-	3,385 T	32,626 T
April	-	122,460 T	35,239 T
May	18,350 T	14,257 T	41,270 T
June	300 T	76,000 T	3,400 T
July	-	3,221 T	Pit Finished
August	-	48,633 T	-
September	14,930 T	11,120 T	-
October	21,200 T	-	-
November	87,055 T	-	-
December	4,065 T	-	-
Yearly Totals	145,900 T	425,164 T	133,500 T
Overall Total	704,564 T		

A comparison of the mill estimate tonnage from Table 1 and the total reported tonnage shows the reported tonnage to be 78,349 tons too high. The remainder of this report is an attempt to locate the difference.

A review of all monthly survey sheets indicates a survey total of 665,564 tons. An additional 26,665 tons should be added to this total, in that a comparison of the survey tonnage reported for the 7350' level indicates only 14,000 tons. My planimetry estimate is 34,426 tons. This results in a difference of 20,426 tons. As indicated in Table 2, reported tonnage for the month of November, 1974, was 87,005 tons. Total survey tonnage for the month was only 60,390 tons. The difference between the reported monthly tonnage and the survey tonnage is the 26,665 tons mentioned above. This, in my opinion, should logically be added to the reported survey tonnage to give a total of 692,229 tons. One additional point that should be mentioned is that truck tallies were used to determine monthly tonnages from May through December of 1974. Total reported tonnage for that period was 145,900 tons. My evaluation of historical records for the year of 1975 in February, 1976, (weighometer vs truck tally vs loader tally) indicated that truck tallies were actually only 81% of total truck tonnage. That results in a possible excess reported tonnage of 27,721 tons.

Table 3 shows a comparison between the survey tonnage and my estimate overall tonnage. The possible 1974 truck tally error is also subtracted.

Table 3:

Survey tonnage	692,229 tons
My estimate	- 634,115 tons
	58,114 tons
1974 Error (?)	- 27,721 tons
Remainder	30,393 tons

The remaining 30,393 tons is probably to be found in the specific volume factors used in the survey reports. A factor of 10 cubic feet/ton was used when the ore grade was approximately greater than 2% sulfide copper. This specific volume figure is probably too low because the waste rock in high grade zones is slightly more porous than non-altered, non-mineralized country rock. The use of 11 cubic feet/ton seems to give much better agreement with mill weights (i.e., my estimate). If one converts survey tonnages where possible to a consistent 11 cubic feet/ton, an additional 31,264 ton excess is indicated. This explains the remaining 30,393 ton difference between the surveyed tonnage and my estimate.

The remaining tonnage difference between the reported monthly tonnage and the survey difference is 12,335 tons. 891 tons can possibly be explained as double-reporting, in that the old 150 level drift cut through the pit and was not subtracted. I can not explain the remaining 11,444 tons.

The grade difference between the reported monthly tonnages and grades and the mill tonnage and grades is again partly traceable to the use of a 10 cubic feet/ton specific volume for higher grade zones. This would have a tendency to report more high grade ore than is actually present. The 31,264 ton excess has to be subtracted at least a 3.36% grade.

Jerome H. Kaczmarowski

Jerome H. Kaczmarowski
Geologist

JHK:sk

Report on

UNDERGROUND DIAMOND DRILLING RECOMMENDATIONS

VICTORIA MINE

Elko County, Nevada

J. H. Kaczmarowski

December 1975

SUMMARY

Recommended diamond drilling in the Victoria ore body can be categorized as follows:

1) Drilling within the low-grade breccia zone between the 7000' level and the 6900' level to insure that the entire zone is indeed low-grade;

2) Drilling within the block located between the 6750' level and the 6550' level to prove up the reserves in that block; and,

3) Exploration drilling of the projected north and south limbs of the Victoria ore body below the 6550' level to determine the amount of ore present. If substantial tonnages are found, mining operations in the 6750' level to 6550' level block may need to be re-examined.

The cost of a program to accomplish these aims is summarized below.

Table 1

740' of Drifting on the 300' Level	$ 148,000
Drill Stations	10,500
Diamond Drilling	233,640
	$ 392,140

It is suggested that the entire drilling program be initiated as soon as the 300' level is intersected by underground mining operations. This is to insure that there will be ample time to complete the entire drilling

program (projected time for the whole program is from Oct. 1, 1976 to March 4, 1977) and modify mining operations if the exploratory drilling proves successful. Any exploratory hole in ore should be extended as long as the potential for ore exists.

A summary of the pertinent information on each recommended hole is shown in Table 2.

Table 2

Tabular Summary - Underground Drilling Recommendations
VICTORIA MINE
Elko County, Nevada

Rec.	Objective	Bearing	Inclination	Length DD	Location*	Estimated Cost
1	Low-Grade Block	N.90°E.	-69°	495	Drill Station 1	$ 9,900
2	Low-Grade Block	-	-90°	330'	Drill Station 1	6,600
3	6800' - 6750' Block 6750' - 6550' Block	-	-90°	557'	Drill Station 2	11,140
4	6750' - 6550' Block	-	-90°	753'	Drill Station 3	15,060
5	Exploration below 6550' 6750' - 6550' Block	-	-90°	672'	Drill Station 4	13,440
6	6750' - 6550' Block	-	-90°	560'	Drill Station 5	11,200
7	6750' - 6550' Block	N.90°E.	-77°	549'	Drill Station 5	10,980
8	6750' - 6550' Block	N.90°E.	-65°	481'	Drill Station 5	9,620
9	Exploratory 6750' - 6550' Block	-	-90°	418'	Drill Station 6	8,360
10	Exploratory 6750' - 6550' Block	N.90°E.	-80°	584'	Drill Station 6	11,680
11	Exploratory 6750' - 6550'	N.90°E.	-68°	533'	Drill Station 6	10,660
12	6750' - 6550' Block -6850' Exploration	Due N.	-63°	611'	Drill Station 7	12,220
13	6750' - 6550' Block -6550' Exploration	Due N.	-46°	705'	Drill Station 7	14,100
14	Exploration 6750' - 6550' Block	N.11°E.	-63°	623'	Drill Station 7	12,460
15	Exploration 6750' - 6550' Block	N.6°30'E.	-44°30'	682'	Drill Station 7	13,640
16	Exploration below Center of Ore Body	N.10°E.	-54°	571'	Drill Station 7	11,420
17	Exploration 6750' - 6550' Block	N.31°E.	-57°	592'	Drill Station 7	11,840
18	Exploration 6750' - 6550' Block	N.17°E.	-36°30'	684'	Drill Station 7	13,680
19	Exploration 6750' - 6550' Block	N.28°E.	-48°	589'	Drill Station 7	11,780
20	Exploration	N.90°W.	-80°	693'	Drill Station 4	13,860
	TOTALS:			11,682'		$ 233,640

* See Figure 1, Plan of 300' Level, including proposed drifting and drill stations, for drill hole locations.

INTRODUCTION

The following is a series of recommendations for underground drilling in the Victoria ore body. The report is preliminary in nature because some of the holes recommended may need to be extended or may require minor directional changes as more information is acquired.

New crosscutting has been kept to a bare minimum to attain a reasonable intercept angle with the ore blocks to be checked. This angle is generally greater than 45^{o}; but, in several cases, the angle is nearer to 40^{o} if too much additional drifting is required to attain the 45^{o} angle.

The holes are laid out to accomplish a dual purpose where possible. Therefore, there are only two holes classed as purely exploration. Holes are recommended in the order indicated for two reasons:

1) As crosscutting progresses, drilling should be started as soon as the appropriate drill station is reached. This insures that no holes will be lost for lack of time. The mining operation will eventually deny access to the 300' level.
2) The holes that should prove up the most accessible and largest tonnages of ore are the first to be drilled. This approach enables the mine planners to examine the mining plan as early as possible to determine if changes are necessary.

The program should be viewed as a total package in that access to the area to be examined will be greatly limited if additional drilling becomes necessary at a later date. Also, if the program is cut short after the cross-cutting is finished, but before all the recommended holes are completed, valuable information will be lost and any holes already completed become extremely expensive.

RECOMMENDATIONS

A. New work (see Appendix 1, 300' Level Plan).

Drifting is to begin at the end of the rehabilitated 300-2 Drift west (Point A) and will extend the 300-2 Drift west 330' to Point B. Cross-cut B C is finished next in order to begin Holes 6, 7 and 8 to allow drilling to be continued until Crosscut B C is finished. Then Crosscut B D is to be completed and Holes 9,10, and 11 begun. Crosscut D E is the last working to be driven.

B. Drill Stations (see Appendix 1, 300' Level Plan).

Drill Stations 1 through 7 should be completed as soon as drifting and crosscutting have reached the proper locations. Drill Station 1, 2, 3, 4 and 6 are set off to the side to afford clearance for any operations in the main workings. Drill Stations 5 and 7 offer no clearance problems and are placed at the end of headings.

C. Drilling

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
1	Low-Grade Block	N. 90^{o}E.	-69^{o}	495'	Drill St. 1	$ 9,900
2	Low-Grade Block	-	-90^{o}	330'	Drill St. 1	6,600

Both Hole 1 and 2 are to be drilled to check the low-grade block indicated on Assay Section N 48750 (Appendix 2). Hole 1 is also used to check the east ore intercept at the 6890 level. If additional ore is indicated, this hole should be extended. Hole 2 is extended as shown to check the center of the ore body from the 6870' (500' level) to the 6750' level.

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
3	6800-6750' Block 6750-6550' Block	-	-90^{o}	557'	Drill Sta. 2	$ 11,140

Hole 3 is to be drilled to check the east end of the 6750-6550' ore block. This was the potential of proving up 74,091 tons of ore. Additionally, the hole also checks the west end of the ore down to the 6750' level (see Appendix 2, Assay Section N 48,750).

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
4	6750-6550' Block	-	-90°	753'	Drill Sta. 3	$ 15,060

This hole is recommended to prove up about 62,727 tons of ore in the 6750-6550' ore block (see Appendix 2, Assay Section N 48,750).

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
5	Exploration below 6550' 6750-6550' Block	-	-90°	672'	Drill Sta. 4	$ 13,440

Hole 5 is recommended to prove up about 12,600 tons of ore in the 6750-6550' Block. Additionally, this hole is to be extended to at least the 6400' level to determine if economic mineralization continues in the same trend as seen in the upper levels.

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
6	6750-6550' Block	-	-90°	560'	Drill Sta. 5	$ 11,200
7	6750-6550' Block	N.90° E.	-77°	549'	Drill Sta. 5	10,980
8	6750-6550' Block	N.90° E.	-65°	481'	Drill Sta. 5	9,620

Holes 6 through 8 (see Appendix 2, Section N 48,800) are recommended to prove up various portions of the northern part of the ore body. Ore potential is as follows:

Hole 6	37,455 Tons
Hole 7	40,773 Tons
Hole 8	43,098 Tons

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
9	Exploratory 6750-6550' Block	-	-90^{o}	418'	Drill Sta. 6	$ 8,360
10	Exploratory 6750-6550' Block	N. 90^{o} E.	-80^{o}	584'	Drill Sta. 6	$ 11,680
11	Exploratory 6750-6550' Block	N. 90^{o}E.	-68^{o}	533'	Drill Sta. 6	$ 10,660

Holes 9 through 11 (see Appendix 2, Section N 48,700) are recommended to prove up the southern limb of the ore body. All three should be extended as long as a geologically sound reason exists (such as ore or likely-looking breccia). The following table gives potential ore tonnages for each hole.

Hole 9	34,218 Tons
Hole 10	32,800 "
Hole 11	36,339 "

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
12	6750-6550' Block -6550' Exploration	Due N.	-63^{o}	611'	Drill Sta. 7	$ 12,220
13	6750-6550' Block -6550' Exploration	Due N.	-46^{o}	705'	Drill Sta. 7	$ 14,100

The main purpose leading to the recommendation of Holes 12 and 13 (see Appendix 3, Assay Section E 47,300) is exploration of the south and north limbs, respectively, of the ore body. A secondary purpose is indicated by the potential tonnages listed below.

Hole 12	5,972 Tons
Hole 13	13,095 "

Both sould be extended as long as the possibility for more ore exists.

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
14	Exploration 6750-6550' Block	N.11^{o} E.	-63^{o}	623'	Drill Sta. 7	$ 12,460
15	Exploration 6750-7550' Block	N. 6^{o}E.	-45^{o}	682'	Drill Sta. 7	$ 13,640
16	Exploration below center of ore body	N.10^{o}E.	-54^{o}	571'	Drill Sta. 7	$ 11,420

Holes 14 and 15 (see Appendix 3, Assay Section E. 47,350) are recommended prior to Hole 16 because they prove up the ore tonnages listed below; and, at the same time, explore the south and north limbs (respectively) of the Victoria ore body. Hole 16, while aiding in providing up ore in the south limb of the ore body, is mainly to explore the area below the center of the ore body.

Hole 14	9,136 Tons
Hole 15	10,118 "

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
17	Exploration 6750-6550' Block	N.31°E.	-57°	592'	Drill Sta. 7	$ 11,840
18	Exploration	N.17°E.	-36°30'	684'	Drill Sta. 7	$ 13,680
19	Exploration 6750-6550' Block	N.28°E.	-48°	589'	Drill Sta. 7	$ 11,780

Holes 17, 18, and 19 (see Appendix 3, Assay Section E. 47,450), while aiding in establishing grades and tonnages, are essentially exploration holes. They should be extended until the possibility for additional ore no longer exists.

Rec.	Objective	Bearing	Inclination	Length	Location	Estimated Cost
20	Exploration	N.90°W.	-80°	693'	Drill Sta. 4	$ 13,860

This hole is recommended primarily as a deep exploration hole. If it is successful,then additional drilling may be indicated to delineate another ore block below the 6550' level.

COSTS

Cost calculations are based on the estimates indicated in Table 2.

Table 2

Drifting	$ 200/foot
Drill Stations	$ 1500/station
Diamond Drilling	$ 20/foot

PRIORITIES

The above recommended priorities are based on ore tonnage potentials and are set up with a view toward maintaining continuous drifting and drilling operations. The information will aid in future mine planning and will give a much clearer idea of deeper reserves. As mentioned above, however, this program should be viewed as a complete package. Any loss of holes after crosscutting is completed makes information already obtained very expensive. The plan is nevertheless set up with a view toward stopping the program if poor ore potentialities are encountered.

Respectfully submitted,

J. H. Kaczmarowski

J. H. Kaczmarowski

JHK:lek

THE ANACONDA COMPANY
General Mining Division
Victoria Mine
P.O. Box 65
Wendover, Utah 84083

INTER-COMPANY CORRESPONDENCE

To: M. A. McKinnon Date: March 10, 1976

From: J. H. Kaczmarowski Subject: Underground Ore Reserves

THIS REPORT UPDATES THE VICTORIA MINE UNDERGROUND RESERVES. INCLUDED INFORMATION IS BASED ON PIT DATA, DRILL HOLES, UNDERGROUND LEVEL ASSAYS, AND EARLIER RESERVE REPORTS.

UNDERGROUND RESERVE SUMMARY
- BY LEVEL -

	PROVEN		PROBABLE	
LEVEL	TONS	GRADE (% CU)	TONS	GRADE (%CU)
7179-7275	209,886 t	3.80		
7143-7179	198,245 t	2.09		
7107-7143	215,697 t	2.07		
7071-7107	175,734 t	2.38		
7035-7071	150,374 t	2.42	6,219 t	1.97
6999-7035	128,336 t	2.31	15,769 t	2.09
6963-6999	113,022 t	2.13	45,254 t	3.42
6927-6963	138,208 t	2.47	16,342 t	3.42
6891-6927	156,442 t	2.57		
6855-6891	168,439 t	2.45		
6819-6855	176,456 t	2.32		
6786-6819	96,128 t	2.29	64,799 t	1.81
G. TOTAL	1,926,967 t	2.47	148,383 t	2.51

J. H. Kaczmarowski
J. H. KACZMAROWSKI

REPORT ON

UNDERGROUND DIAMOND DRILLING RECOMMENDATIONS

VICTORIA MINE

ELKO COUNTY, NEVADA

J.H. KACZMAROWSKI OCTOBER 1976

Recommended diamond drilling in the Victoria ore body from the 7071'level can be categorized as follows:

1) Drilling within the block located between the 6750' level and the 6550' level to delineate the reserves in that block.
2) Exploration drilling of the projected north and south limbs of the Victoria ore body below the 6550' level to determine the amount of ore present. If substantial tonnages are found, mining operations in the 6750' level to 6550' level block may need to be re-examined.

The cost of a program to accomplish these aims is summarized in Table 1.

Table 1. DRILLING PROGRAM COSTS

753' OF DRIFTING ON 300' LEVEL	$ 150,600
5 DRILL STATIONS	7,500
12,356' DIAMOND DRILLING	247,120
	$ 405,220

Assuming that all contacts are approximately as visualized and the grade is economic, the potential exists to gain 450,000+ tons of ore (This is an increase in reserves of 21.4%).

It is suggested that the entire drilling program be initiated as soon as possible to insure that there will be ample time to complete the entire drilling program (projected time for the whole program is from March, 1977 to October, 1977.)

The choice of using the 7071' level as the drilling level is based on the following:

1.) If the drilling is delayed until a lower level is reached, and the reserves should appreciably change, mining options are reduced.

2.) As mining operations enter the present upper levels, any additional drifting reduces the mine's ability to reach a 1000 tons/day production rate at the earliest possible time.

The location of the access drift is determined on the following parameters. The drift -

A. Must allow the majority of the drill holes to intercept the probable ore contacts at angles greater than 45^{o}; and,

B. Must be clear of any possible caving resulting from mining operations.

Plan 1, appendix, indicates the location of the access drift. It should be noted that a possibility exists that extraction drift # 10 could be used as part of the access drift. This would save 97 feet of drifting. Production requirements at the time will determine if this approach is feasible.

The drill stations are also shown on Plan 1, appendix. The 50' interval between drill stations with the drill holes fanning north is recommended to allow for the grading of ore blocks with a maximum distance to any hole of 25' in the East-West direction and 50' maximum to any hole in the North-South plane.

All drilling is to be NX-sized core. This will facilitate mapping and analysis. Table 2 gives the pertinent information on each recommended hole. Hole numbers indicate order of priority.

Table 2. RECOMMENDED SUBSURFACE DRILLING AT VICTORIA.

HOLE #	INCLINATION	DIRECTION	LENGTH	LOCATION *	OBJECTIVE	COST
1	-36.5°	Due North	612'	N48,410 E47,450	Northern Orebody	12,240
2	-47.25°	"	600'	"	Center of Orebody	12,000
3	-54.75°	"	620'	"	South Limb	12,400
4	-62.5°	"	578'	"	South Limb	11,560
5	-40°	"	643'	N48,400 E47,400	Northern Orebody	12,860
6	-53°	"	664'	"	Southcentral Orebody	13,280
7	-66.5°	"	625'	"	South Limb	12,500
8	-46°	"	696'	"	Northern Orebody	13,920
9	-59.5°	"	648'	"	South Limb	12,960
10	-45°	"	682'	N48,390 E47,350	Northern Orebody	13,640
11	-55°	"	700'	"	Southcentral Orebody	14,000
12	-63.5°	"	645'	"	South Limb	12,900
13	-40°	"	604'	"	Low Grade Zone-North	12,080

HOLE #	INCLINATION	DIRECTION	LENGTH	LOCATION *	OBJECTIVE	COST
14	-55.5°	Due North	714'	N48,395 E47,300	Central Orebody	14,280
15	-46.5°	"	706'	"	"	14,120
16	-63°	"	618'	"	South Limb	12,360
17	-53.5°	"	749'	N48,395 E47,250	Central Orebody	14,980
18	-58.5°	"	697'	"	Southcentral Orebody	13,940
19	-58	Due West	555	N48,630 E47,942	Eastern South Limb	11,100
TOTAL			12,356			247,120

* Location of drill stations and recommended holes are indicated on Plan 1, appendix A.

The holes that penetrate the most accessible and largest tonnages are generally the first to be drilled. This approach enables the mine planners to examine the mining plan as early as possible to determine if changes are necessary.

Table 3 indicates tonnages projected to be intercepted by hole, assuming the contacts are as presently projected.

Table 3. PROJECTED TONNAGES BY HOLE.

HOLE #	TONNAGE
1	31,100
2	53,800
3	19,100
4	8,700
5	25,200
6	31,700
7	14,500

Table 3 cont.

HOLE #	TONNAGE
8	40,000
9	18,800
10	48,500
11	30,000
12	14,300
13	LOW GRADE ZONE CHECK 0 TONNAGE.
14	21,600
15	34,500
16	10,900
17	8,400
18	41,000
19	0 TONNAGE *

* THE PURPOSE OF HOLE # 19 IS TO OBTAIN GRADE INFORMATION. PREVIOUS WORK HAS DETERMINED THE CONTACTS.

Sections - both North and East - and level plans are included in the appendix to show location and intercepts of all holes.

Table 4 indicates the costs used to estimate the total cost of the recommended drilling program.

Table 4.

Drift	753'	@ $200/foot (This is in line with present mining costs)	$ 150,600
Drill Stations	5	@ $1,500/Station	$ 7,500
Drilling	12,356'	@ $20/Foot (This per foot estimate includes set-ups, drill mud, etc.)	$ 247,120

J.H. Kaczmarowski

J.H. Kaczmarowski
Geologist

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THE ANACONDA COMPANY

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INTER-COMPANY CORRESPONDENCE

To: M. A. McKinnon

Date: September 19, 1975

From: J. H. Kaczmarowski

Subject: Slope Stability Information Of The North & Northwest Pit Areas

OBJECTIVE:

To correlate geological and geophysical information in order to indicate major and minor zones of weakness and the extent of said zones. Additionally, to indicate amounts of continuing movement and velocity in the "NNW Block" below the failure which led to a fatality.

CONCLUSIONS:

1. Using the SS-2 hole as a reference line, major zones of weakness occur at the following depths:

a. 50' - 65'
b. + 95'
c. 105' - 110'
d. 135'
e. 155'
f. 195' - 210'

2. Minor zones in the same hole occur at the following depths:

a. 170'
b. 210' - 220'

3. The "NNW Block" maintains a constantly decreasing total movement and velocity which may or may not eventually cause stability problems in the pit bottom.

TYPES OF INFORMATION AVAILABLE:

1. Six rotary or hammer drill holes with downhole geophysical data.

2. The cuttings of three of the above holes were logged to obtain stratigraphic columns.

3. Total displacement between holes was obtained from pit mapping.

4. Quadrilaterals and blinkers are used to monitor the NNW corner of the pit and the areas above and behind it.

WORK PLANNED: No further work is planned at this time other than a daily monitoring of the quadrilaterals and blinkers.

J. H. Kaczmarowski
J. H. Kaczmarowski

JHK:dsd
cc: R. Thomas
Enclosures

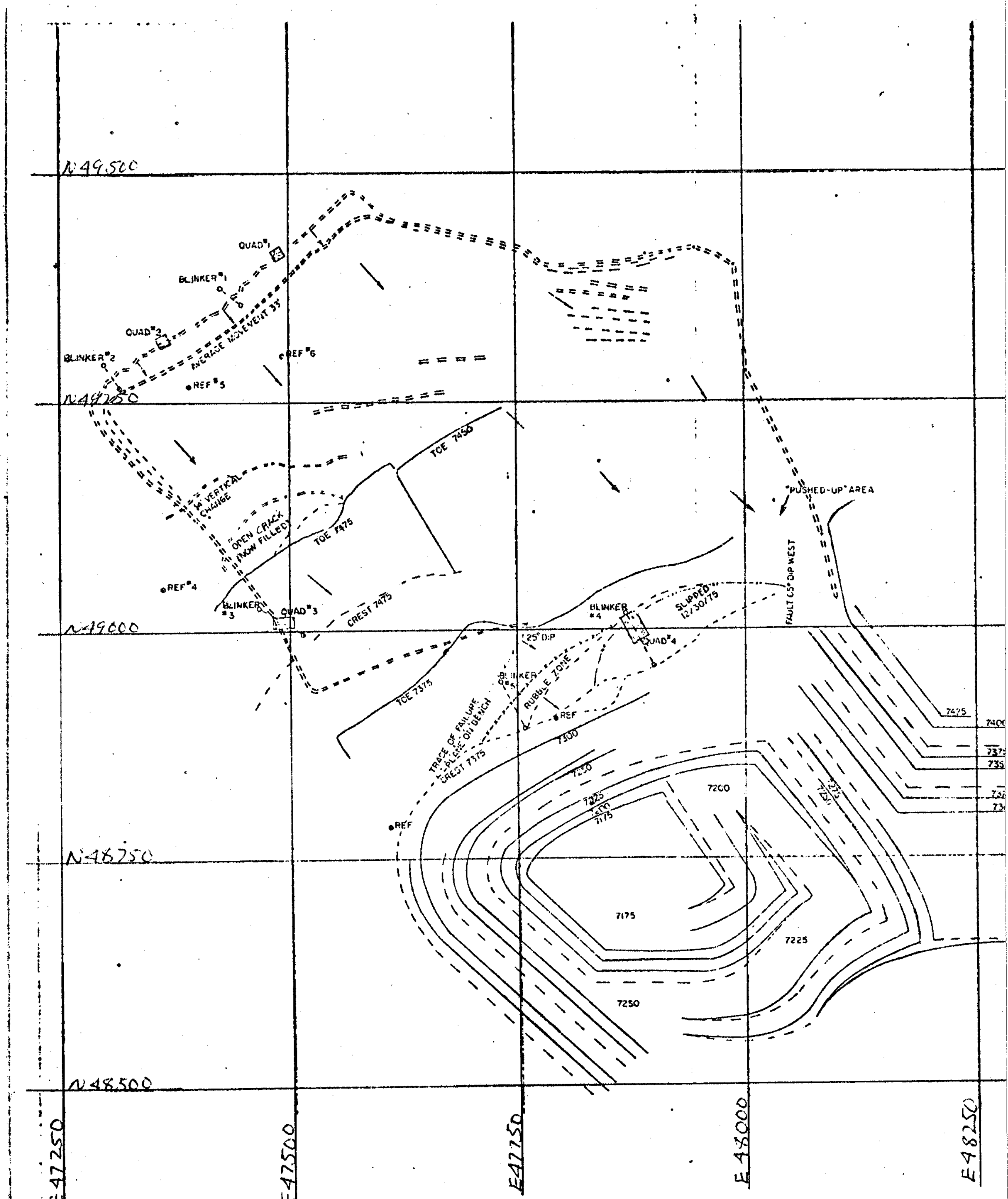


Figure 1: Stability Plan Map

Slope Stability Report 1975
J. Kaczmarowski
September 19, 1975

PREVIOUS WORK:

Previous work by Seegmiller Associates indicated that the most probable slope failure zone would be in the first 190 feet of the Phase II final slopes. One such failure has and is occurring.

OBJECTIVE:

It is now possible to delineate possible failure planes more precisely. These will be indicated. The total movement and velocity of the "NNW Block" failure and the implications of this movement are shown in Figure 4.

CORRELATION:

A plan map indicating the 7475' bench outline and the location of the six pertinant holes is shown in Figure 1. The holes essentially ring the NW area above the pit. RDH-A and RDH-B were the first holes drilled and utilized to obtain seismic velocities used to indicate possible failure zones. The present locations of monitoring equipment are also shown on Figure 1.

Holes SS-1 through SS-4 were drilled during the period from July 22, 1975 to August 5, 1975. The cuttings from holes SS-2 through SS-4 were logged and the stratigraphic columns on Figure 2 constructed. The P-wave velocities obtained from each of the SS-2 through SS-4 holes are shown schematically next to the geological columns. Velocities from holes SS-1, RDH-A, and RDH-B are shown in their correct locations on the section in Figure 2. In all cases, the various holes have their collar elevations corrected in relation to the section.

The major and minor classifications are arbitrary divisions based on the amount of information available to delineate each zone. Table 1 indicates the basis for each of the major failure zones.

Table 1 Information Used To Delineate Each Major Failure Zone.

Major Zone	Seismics In	Geology In
A	SS-1, RDH-A, RDH-B	SS-2, SS-3, SS-4
B	SS-1, SS-2, RDH-A, RDH-B (?)	SS-2, SS-3, SS-4 (?)
C	SS-1, SS-2, SS-3, RDH-A, RDH-B	SS-2, SS-3
D	SS-3, RDH-B	SS-3, SS-4
E	SS-2, SS-3	SS-3, SS-4
F	SS-2, SS-3, RDH-B	SS-2, SS-3

Figure 2. Indicates information used to delineate minor failure zones.

Table 2. Basis for delineation of minor failure zones.

Minor Zone	Seismics In	Geology In
A	SS-3	SS-2, SS-3
B	SS-2, SS-3, RDH-B	SS-2, SS-3

Note: Minor zone B is downgraded because information is confined to the center half of the section. It is likely that it is really a major weakness zone.

The block below - and probably including - the fatality failure area (See Figure 3) has been moving continuously since the failure. Figure 4 shows a plot of the resultant movement and a plot of the velocity as indicated by a quadrilateral located across the western crack on the 7475' bench locating the edge of the block.

Both curves are constantly decreasing. The main problem is that the monitored crack continues over the edge of the 7475' bench and nearly reaches the 7375' bench. A pile of rubble obscures the lower edge of the crack. The main support for the 7375' bench is mineralized breccia, and the possibility exists that the bench may fail if too much pressure is placed upon it from the above moving block. The 7375' bench is monitored by 2 reflectors, 2 blinkers, 1 qualdrilateral, and constant visual observation when personnel are in the pit bottom.

CONCLUSIONS:

1. Using the SS-2 hole as a reference line, major zones of weakness occur at the following depths:

a. 50' - 65'
b. $\pm$95'
c. $\pm$105' - 110'
d. 135'
e. 155'
f. 195' - 210'

2. Minor zones in the same hole occur at the following depths:

a. 170'
b. 210' - 220'

3. The "NNW Block" maintains a constantly decreasing total movement and velocity which may or may not eventually cause stability problems in the pit bottom.

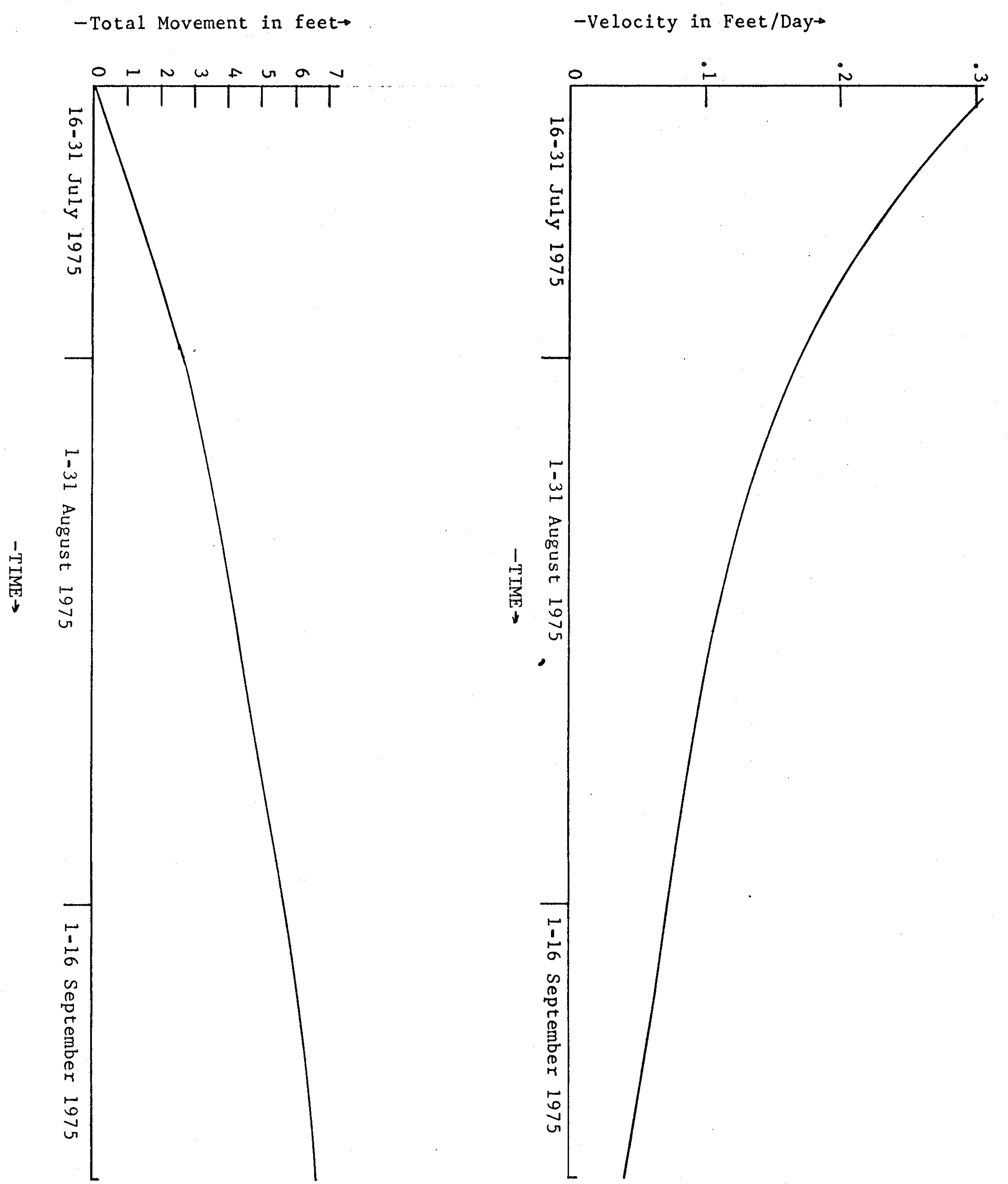


Figure 4: Velocity and total movement readings based on 7475' bench quadrilateral.

FIGURE:3

Section Illustrating Collapse in Northwest Face, Victoria Mine. Line of Section N50°W. Looking N.40° E.

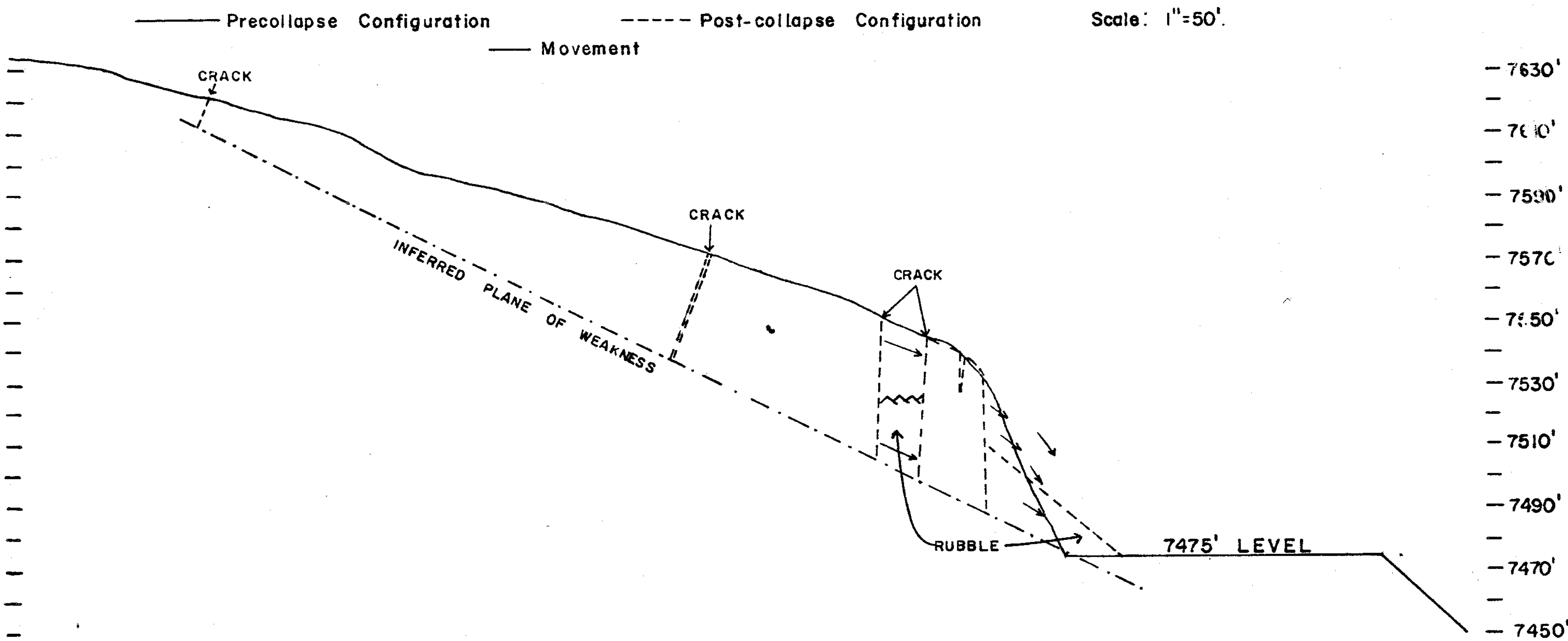
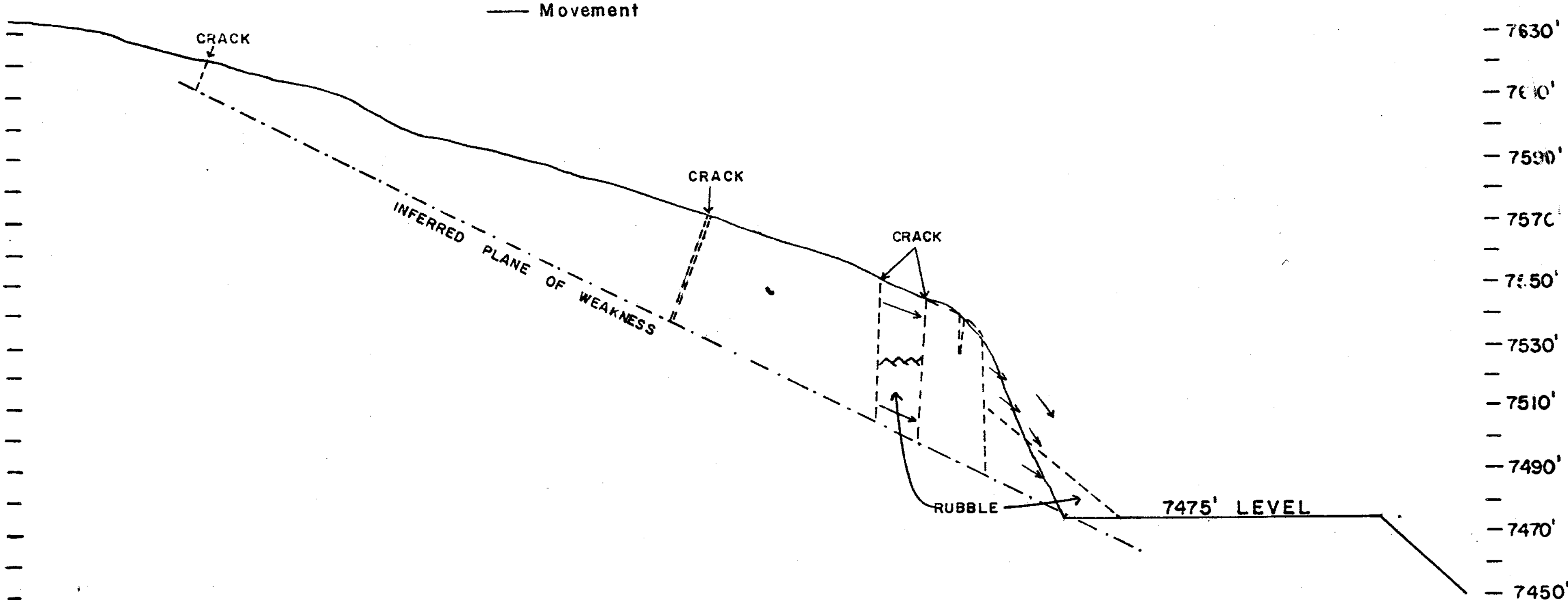


FIGURE:3

Section Illustrating Collapse in Northwest Face, Victoria Mine. Line of Section N50°W. Looking N.40° E.

——— Precollapse Configuration ----- Post-collapse Configuration Scale: 1"=50'.

—— Movement



3

House 16,000
Well 4,000
Lot 16,000
Founda 6000
Sept 2000
Fuit 4000
Roof 2000

50,000
80 000
30 000

1
15
300
35

50,000
16000
34,000

3
16,000
4 000
7 000
6 000
2 000
4 000
2 000
41 000

41,000
7.000
34 000

300