CONTACT INFORMATION
Mining Records Curator
Arizona Geological Survey
3550 N. Central Ave, 2nd floor
Phoenix, AZ, 85012
602-771-1601
http://www.azgs.az.gov
inquiries@azgs.az.gov

PERRY, KNOX, KAUFMAN, INC.
MINERAL EXPLORATION AND DEVELOPMENT

OFFICES:
TUCSON, ARIZONA (BUSINESS)
2345 E. BROADWAY, SUITE 206
P. O. BOX 12754, ZIP 85732
TELEPHONE (602) 622-0582

SPOKANE, WASHINGTON
NORTH 20 PINES ROAD, SUITE 21
P. O. BOX 14336, ZIP 99214
TELEPHONE (509) - WA 4-0878

Spokane, Washington
January 27, 1975

To: J. B. Imswiler
International Minerals and Chemical Corporation

From: Perry, Knox, Kaufman, Inc.

Subject: DeSoto Mine Evaluation, Yavapai County, Arizona

I. Summary and Recommendation

The DeSoto property is a Precambrian, volcanogenic, stratabound massive sulfide prospect from which approximately 290,000 tons of 3.3% copper * ore, with nuinor gold, silver, and zinc values, have been mined. Two parallel zones of copper mineralization in altered siliceous, tuffaceous, rhyolitic schists have been defined by surface and subsurface exploration over strike lengths up to 1,500 feet (west zone) and to depths of 6-900 feet. Substantial widths of 0.5-1.0% copper mineralization have been shown to exist to the depths tested. It is not unreasonable to assume that other lenses of ore might exist at greater depth along this favorable rhyolitic horizon. Two to three million tons of economic grade copper ore, with associated gold and silver values, would support a very profitable operation.

It is recommended that an attempt be made to option the property on a reasonable basis; renegotiation of an underlying agreement will be required to obtain an acceptable payment schedule during the initial exploration period. Minor geologic mapping and a limited Turan E. M. Survey will be required. Three diamond drill holes are proposed to test the two zones at greater depth, approximately 3,000 feet of drilling. Estimated cost of this program is on the order of $53,000, exclusive of property payments.

* USGS Bull 782 page 163 states 180,000 tons of 3.75% Cu. c last ore mined averaging 2.25% Cu.

II. General

The DeSoto property is situated in the Bradshaw Mountains (Prescott National Forest) approximately 8 1/2 miles S. SW. of Mayer and 3 miles west of Cleator in Sections 31 and 32, T. 11N, R. 1 E., Yavapai County, Arizona. Local topography is rather rugged, though not extreme, and poses no serious problem to exploration or possible development. The elevation in the immediate prospect area is approximately 5,500 feet. Current access is by several miles of rather second-rate road from the Mayer-Crown King road.

Past production is reportedly on the order of 290,000 tons of 3.3% copper, .05 oz./T gold, 1.2 oz/T silver, and less than 1% zinc. Part of this production undoubtedly was derived from higher grade semi-massive sulfide lenses that also contained better zinc values.

III. Geology

Some geological information on the general area can be found in several U.S.G.S. publications and maps:

U.S.G.S. Bulletin 782
U.S.G.S. Bulletin 1345
U.S.G.S. Geologic Map GQ 996
U.S.G.S. Geologic Map GQ 997
U.S.G.S. Air Mag Map GP 758

Of interest, also, are a report on the Iron King Mine in the A.I.M.E. volume, Ore Deposits of the U.S., and a general massive sulfide review by Sangster in Geological Survey of Canada Paper 72-22.

A review of the recent U.S.G.S. geologic mapping for the Mt. Union and Mayer quadrangles, which is at a scale of 1" = 1 mile and rather generalized, shows the DeSoto area to be underlain by Iron King Volcanics (Precambrian Archean Yavapai Schist) which consist mainly of andesitic and basaltic flows and rhyolitic flows and tuffs. The rhyolitic rocks exposed at the DeSoto are rather localized and are not shown on the U.S.G.S. map. The U.S.G.S. mapping, however, does show the DeSoto to be situated on the east flank of a west-dipping, overturned syncline; this indicates that the steeply-dipping bedding at the DeSoto is stratigraphically right side up.

The rocks exposed at surface and intersected by drilling at The DeSoto are believed to be mainly biotitic, chloritic, feldspar schists derived from andesitic tuffs and graywackes; quartz, sericite and quartz, chlorite, sericite schists derived from rhyolite pyroclastics, and very low iron, partially tuffaceous chert. The rhyolite-chert units apparently form a lenticular band at least 850 feet thick in the vicinity of the copper mineralization that extends along strike over lesser widths for perhaps 2,000 feet. At surface, the rhyolitic schists appear to possibly grade southward into the chert; their relationship at depth is unknown. Foliation, which probably conforms very closely to bedding, strikes approximately N 15-20° E. and dips 70-75° west.

The copper mineralization observed occurs mainly in two siliceous, schistose lenses up to 250 feet wide (east zone) separated by approximately 500 feet of chert and relatively barren schist. The better grades of mineralization exposed at surface and by underground workings extend perhaps 5-600 feet along strike, though lower grade mineralization can be traced for 1,500 feet along the west zone. Within the east zone primary sulfide mineralization, principally pyrite and chalcopyrite, occurs as streaks, disemenations and small semi-massive lenses, roughly parallel to foliation, in economic, lenticular shoots to 40 feet in width, 150 feet in strike length, and several hundred feet in plunge length; these shoots have been followed in a zone down-dip approximately 900 feet. Low grade (less than 1% copper) pyrite-chalcopyrite mineralization in disseminations and streaks also occurs between the main shoots ~~on~~ in the east zone; mineralization in the west zone consists largely of this type also.

Within the higher grade zones of copper, black chloritic schist is associated with the quartz, sericite, chlorite schists. The black chlorite and nature of the sulfide mineralization is somewhat suggestive of stringer-type ore, though no cross-cutting relationships of the mineralization and bedding can be demonstrated; shearing, of course, can produce such a situation in stringer-type ore.

IV. Potential

The DeSoto property is a fairly typical Precambrian, volcanogenic, massive sulfide-type occurrence, although the percentage of sulfides present, for the most part, is insufficient to qualify the ore as massive sulfide. Mineralization is associated with acid volcanic pyroclastics and chert, the chert represenging a volcanic exhalite situated within an intermediate to basic volcanic and volcanic-derived metasediment pile. Ore bodies in this type of environment often occur in clusters, usually near the top of the rhyolitic units. Exploration for this type of occurence on a strategraphic basis has proven successful in many instances in the Canadian Shield.

In the Bradshaw Mountain area, at least two other ore bodies have been found in similar geologic settings in the Iron King Volcanics. At the Blue Bell Mine 4 miles northeast, the situation is identical to that at the DeSoto with the exception that the thick lens of chert or weak sulfide iron formation is lacking; the Blue Bell has produced 1 1/2 million tons of copper ore from steeply plunging lenses extending to a depth of 1,500 feet; the grade is comparable to The DeSoto production. At the Iron King Mine 16 miles to the north, 5 million tons of 13-16% zinc-lead ore was mined; at the Iron King Mine, the early production was derived from the "footwall series," a series of short, small, massive sulfide lenses in andesite -- exploration in the hanging wall (stratigraphic foot wall) finally found the main "I Series" that extended to a depth of 3,000 feet; the I Series was poorly exposed at surface, and where exposed, gave little indication of the magnitude of the ore zone below.

It is not unreasonable to assume that additional ore grade mineralization might occur at depth or along strike near the upper contact of the siliceous, schistose, rhyolitic rocks at the DeSoto. If present, this ore could be expected to be at least of comparable grade to that mined in the past; a larger lens or series of lenses might very possibly contain better grade. The zones of mineralization are still on the order of 100 feet wide and well mineralized at the deepest points explored.

Although it is believed that we are dealing primarily with stratabound, volcanogic-type mineralization occurring along a favorable rhyolitic horizon, there is the possibility that this is footwall zone, stringer-type mineralization. This is not particularly significant, however, since a number of very profitable orebodies consist entirely or in part of stringer ore.

The dimensions of a zone of mineralized rock required to constitute an attractive orebody are not great. A zone 40 feet in width, 750 feet long, and 1,000 feet in plunge length would provide a reserve of 2 1/2 million tons.

(Using 12 FT^3/T)
Low

Although of relatively minor importance, the near-surface (open pit), acid soluble copper potential should be considered. Although incompletely evaluated, the presence of approximately 2 1/2 million tons of 0.9% copper mineralization, part of which is leachable, can be demonstrated. A rough economic analysis of this potential indicates that it might be possible to realize a $500,000/year after tax profit over a 7 year life; royalties might reduce ~~their~~ this return considerably.

V. Past Exploration Efforts

Prior to the work of Cutlass Explorations Ltd., all exploration efforts were apparently confined to an evaluation of the acid soluble copper potential and to work immediately adjacent to the old workings; this exploration consisted of a number of short percussion and diamond drill holes.

Cutlass Explorations devoted most of its effort again to a study of the leaching possibilities; a series of percussion, rotary, and diamond drill holes was completed. The last stage of Cutlass' work apparently involved an attempt to test the down-plunge potential of the mineralized zones as shown on the enclosed cross sections. As will be noted, this work failed to reach any greater depth than the old workings in the east zone and only tested the upper 700 foot plunge length of the west zone (400 feet below surface due to topographic configuration). It should be noted that diamond drill holes in this area tend to be deflected perpindicular to foliation; it is probable that the Cutlass holes flattened somewhat and therefore tested less depth potential than indicated by the cross sections.

No geophysical or geochemical surveys have ever been run in the DeSoto area.

VI. Land Status

The DeSoto property consists of 19 patented and 24 unpatented claims. Although a claim map has not yet been provided us, it is assumed that these form a contiguous block with the exception of several patents situated in the valley bottom to the southeast. Sherwood Owens of Tucson is the owner of this ground.

Cutlass Explorations Limited of Vancouver, B.C., holds an option on this property from Owens. The terms of this agreement are not favorable. These terms are summarized as follows:

1. Advance payment schedule (as of January, 1975)
   (All payments due on 22nd of each month)
   $2,500/month - Jan-March, 1975
   30,000 - April, 1975
   2,500/month - May - July, 1975
   32,500 - August, 1975
   2,500/month minimum payments thereafter

2. 10% NSR royalty on production to $1.7 million (all payments applicable), then 5% NSR or $2,500/month minimum payments.

3. Any claims staked within a 3 mile perimeter area are included in this agreement.

Steve Radvak, Consultant for Cutlass, has been dealing with Owens for some time and feels that these terms can be renegotiated to some degree. The following amended schedule is suggested if IMC is interested in pursuing this matter:

1. 5% NSR royalty on production to a $1.5 million end price. (Radvak feels that Owens will not settle for an end price, and a 2% NSR royalty may be required after payments total $1.5 million.)

2. The advance payment schedule must obviously give us adequate exploration time at a reasonable price. At least six months time is needed, after which substantial payments might be justified if economic mineralization has been found.

   $2,500/month - 1st 6 months
   30,000 at end 6 months
   2,000/month thereafter.

3. Work commitments might be considered to make the proposed amendments more acceptable to Owens.

With respect to a deal with Cutlass, it seems obvious they would accept a 10% net profits arrangement. It is suggested that an attempt be made to acquire their interest for 7 1/2-8% net profits, all costs to be recovered on a proportional basis prior to any distribution of net profits. Cutlass claims to have expended approximately $350,000 to date including $125,000 in payments to Owens.

VII. Economic Considerations

Orebodies of the type that might occur on the DeSoto property provide a very attractive return on investment. Although it is not possible to accurately predict the economics of what might be found due to the variables involved, a rough approximation can be made based on reasonable assumptions. A briefly summarized evaluation is as follows:

Reserves: 2 million tons @ 3.5% copper, .05 oz/T gold, 1.0 oz./T silver.

Production Rate: 1,000 TPD; 6 years life

Capital Requirements:

$1.3 million - 1,000 foot, 2 compartment shaft and related facilities
0.7 million - preproduction expense (exploration, etc.)
1.0 million - mining equipment, misc. surface installations
4.0 million - 1,000 TPD flotation mill
$7.0 million - Total Capital

Estimated Operating Costs:

$8.00/T - Mining (Sublevel Stoping)
4.50/T - Milling
1.00/T - Overhead
3.50/T - depreciation and amortization
$17.00/T - Total

Mill Recovery: 90%

Concentrate Grade: 24%

Metal Values: 60¢ copper, $150 gold, $4.00 silver.

NSR Value per ton ore: $36.00/T
Pre-Tax Profit per ton: $19.00/T
Total Pre-Tax Profit after estimated royalty: $36 ~~$3.6~~ million

Estimated Total Net after Tax: $22 million
Estimated Annual Net: $3.6 million
Estimated Annual Cash Flow: $4.7 million
Payback Period: 1 1/2 years

Variations in metal prices, tax rates, depletion, and smelter schedules can obviously affect these numbers appreciably.

VIII Proposed Exploration Program

Exploration of the Desoto property would, at least initially, be quite simple. The down plunge extensions of the known mineralization and the favorable rhyolitic units should be tested at depth. The following program is proposed.

1. Geologic mapping - several days only.

2. Turam E. M. Survey - Although it is doubtful that penetration would exceed 600 feet, this is an inexpensive item (estimate $3,000 for 5 line miles) and should be run on the chance that an otherwise unknown target might be defined.

3. Diamond drilling - 3 holes, 3,000 feet; the initial hole would test the down plunge extension of both mineralized zones; the other two holes would test only the western most zone, probably the zone with the highest potential for deep economic mineralization. These holes would be drilled from the valley to the west, several hundred feet below the mineralized outcrops.

Estimated exploration costs are itemized below:

$40,000	Diamond drilling
2,000	Road and drill site construction
3,000	Assays
3,000	Turam Survey
1,000	Legal
4,000	Supervision
$53,000	Total

As you will note, no provision has been made for property payments.

Consideration must be given to the fact that since any agreement entered into on The DeSoto will require substantial monthly payments, any significant encouragement in the above described program will necessitate immediate subsequent efforts.

James A. Knox

James A. Know
Perry, Knox, Kaufman, Inc.

SWASTIKA MINE

A mile east of the Peck along the road to Peck siding are the Black Warrior and Silver Prince veins, now owned by Frank W. Giroux, of Mayer, under the name of the Swastika Silver & Copper Co.

The Silver Prince is mentioned in Raymond's report of 1877 with the statement that the cost of packing the ore to Prescott was $50 a ton. The Mint report for 1883 mentions both veins, stating that the Black Warrior was 2 to 3 feet wide, that $40,000 in silver had been extracted so far, and that 8 tons a day was milled in a 4-stamp mill for a yield of 113 ounces of silver to the ton. About 1885 the mine was considered exhausted, and it was idle until reopened by F. W. Woods in 1910. From 1910 to 1915 the mine produced 600,000 ounces of silver. The total production is stated to be about 1,000,000 ounces. Since 1915 the mine has been in intermittent operation. Mr. Woods states that from 1875 to 1908 the Silver Prince had yielded $480,000 and the Black Warrior $385,000. The later production came wholly from the Silver Prince.

The country rock consists of Yavapai schist, mostly fissile and sericitic, with lenses of quartzite, but the outcrops are deeply oxidized. The two parallel veins strike due north and dip 60° W., with the schist. The Prince lies 300 feet west of the Black Warrior. Between the two there is a 50-foot dike of light-colored porphyry.

The Silver Prince is developed by tunnels and a 400-foot shaft about 600 feet to the north. The vein is at most a few feet wide and carries dark-brown limonitic ore. There is a little quartz, but the principal gangue mineral is a sideritic carbonate, with native silver, chloride, and some sulphides. The sulphides consist of a partly decomposed tetrahedrite rich in silver and a little chalcopyrite. The ore, which contains a little lead, was sold to El Paso and the lead smelter at Needles in 1914 and later shipped to Salt Lake City.

The shaft on the Black Warrior is said to be only 125 feet deep. The lowest levels were not visited, but it is evident that the ore on them is poorer. Here, too, the conditions are similar to those at the Peck, namely, an extraordinary concentration in the oxidized zone and impoverishment below. Considering the history of this mine, it would be rash to say that it is exhausted. More comments on the concentration in the oxidized zone of these deposits are found on page 49.

DE SOTO MINE

The outcrops of the De Soto copper mine lie 2 miles northeast of the Peck mine, on the summit of the high ridge separating Peck Canyon from Crazy Basin. The altitude is about 5,800 feet. The main tunnel is 600 feet below the outcrop, and from it an incline leads down to Middleton station on the Crown King branch road. The property is owned by the same interests that control the Humboldt smelter (Southwest Metals Co.), to which the ore has been shipped. Work was discontinued in 1922, the ore bodies being considered exhausted. Most of the information given below was obtained from Mr. J. L. White, of the staff of the smelter.

The Yavapai chloritic schists strike N. 23° E. at the mine and dip 70° NW. The ore bodies, which carry pyrite-chalcopyrite ore and are contained in a chloritic schist, have yielded a total of 180,000 tons, averaging about 3.75 per cent of copper with 1 ounce of silver and 0.02 ounce of gold to the ton. The last ore treated contained 2.25 per cent of copper. There is less pyrite than at the Blue Bell

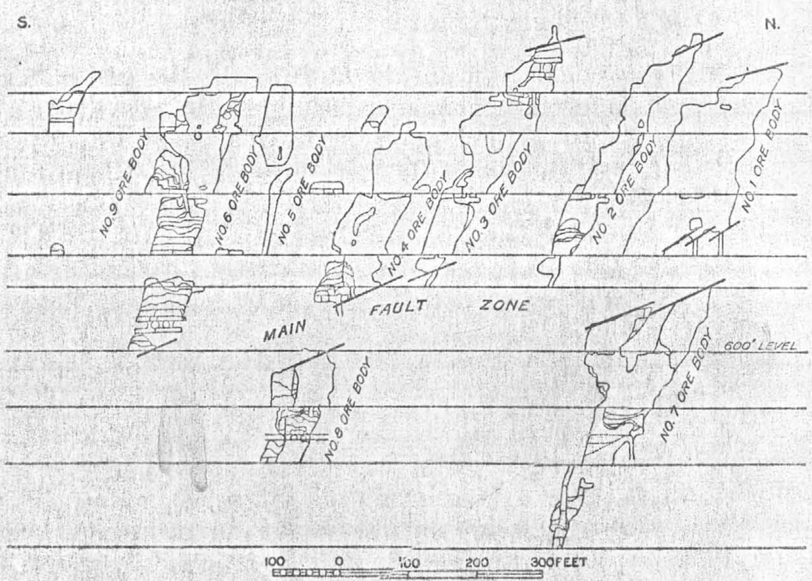


FIGURE 10.—Longitudinal section of ore lenses in the De Soto mine

mine, some sphalerite and galena, and occasional specimens of tetrahedrite. (See pl. 17, *A*.) A few prisms of arsenopyrite were observed.

The gangue is fine-grained quartz. There are also lenses of coarser quartz, much of it crushed and showing undulous extinction. Gangue and sulphides replace the schist, which is mainly chloritic with a little biotite.

The ore occurs in overlapping lenses. (See fig. 10.) On the upper levels there were seven such lenses close together, with an individual width of as much as 50 feet and a greatest length of 250 feet. Exploration extended to a depth of 300 feet below the main tunnel level, and at this depth only one small lens persisted. The total length of the ore zone is 350 feet; the total width 200 feet.

The ore bodies are said to have been cut off in depth by a flat fault 250 feet below the surface. The small bodies found below this depth are believed to represent the continuation of the ore below the fault. The ore bodies are shown in Figure 10, each separately, in a longitudinal projection, for they overlap so that they can not be indicated in their correct relative position.

OTHER PROPERTIES

The veins of the Gold King group, in the southern part of the district, are said to be the extension of the Gladiator vein, which is in the Pine Grove district. Near by is the Blue Bird vein. Both these deposits are in Yavapai schist.

PINE GROVE DISTRICT

The Pine Grove district lies in the heart of the Bradshaw Mountains, in a well-timbered region, at an altitude of 6,000 to 7,500 feet. (See pl. 21, *B*.) Its highest point is the diorite mass of Towers Mountain. Most of the claims lie in a basin-like depression on the east side of the ridge. The district is reached by an automobile road from Prescott, 40 miles long, and by a branch railroad from Mayer, which ascends Crazy Basin and Poland Creek in a series of switchbacks. It is an old mining region, and many of the veins were very rich near the surface. The earliest properties worked were the Del Pasco, Gladiator, and War Eagle. The ores carry silver and gold.

Most of the mines are situated in granodiorite (quartz diorite, according to Jaggar and Palache), which forms a rounded mass 3 to 4 miles in diameter, intruded into Bradshaw granite and still earlier Yavapai schist. The granodiorite is cut by a series of dikes which trend north-northeast across the center of the area. In part these dikes are rhyolite porphyry, in part granite porphyry. There are also some light-colored granitic dikes which seem to be affiliated with the granodiorite; the others just mentioned appear to represent a distinctly later intrusion.

There are three prominent vein systems, which trend north-northeast and generally dip about 60° WSW. The shoots have a tendency to pitch northward. They occur mostly in the granodiorite but continue also to the north in Yavapai schist, diorite, and mixed areas (Wildflower mine), though these harder rocks are as a rule less favorable. Few of the veins are more than 5 feet in width, and they contain a filling of predominant quartz, with some ankerite and calcite.

Much of the quartz is drusy and contains more or less pyrite, chalcopyrite, zinc blende, and galena, with some tetrahedrite. In places free gold occurs in the primary ore. Most of the ore extracted

Preliminary Production Evaluation

of the

DESOTO PROPERTY, YAVAPAI COUNTY, ARIZONA

for

CUTLASS EXPLORATION LIMITED
1606 - 1055 West Georgia Street
Vancouver, B.C.
V6E 3P3, Canada

by

C. M. Armstrong, P.Eng.

Consulting Engineer

4085 West 29th Avenue
Vancouver, B.C.
V6S 1V4, Canada

(604)224-7678

September 18, 1974

CONTENTS

INTRODUCTION

On behalf of Cutlass Exploration Limited, the writer examined the DeSoto property on August 7, 1974, under the guidance of J.Simpson who supervised much of the recent work on the property.

The principal purpose of the examination was to assess the production potential of the oxidized portion of the East Zone, considering in situ leaching, or mining, preferably by open pit techniques, and processing by some other form of acid leaching.

The surface exposure of the East Zone and readily accessible workings on the 200 Level (5160 elevation) were examined. A small slide accumulation of silt and debris at the collar of the 600 Level adit dammed over 1 foot of water in the tunnel, preventing examination of that level. The West Zone was examined on surface outcrop and in the 5260 elevation adit (100 Level?).

Copies of plans and sections, assay details for some of the drill holes, and a recent report (May 17/74) by R.H.Seraphim, P.Eng. were made available to the writer to assist in the evaluation.

Details regarding physiography, vegetation, rock exposure, soil, water, power, climate, property and claims, history, regional geology, local geology and mineralization, etc., most of which have been documented in earlier reports, are not repeated here; but data of particular importance to this evaluation are presented in summary form.

GENERAL

As shown on Figure 1 the DeSoto property is approximately 68 road-miles north of Phoenix: 53 miles on paved Highway 17 to Bumble Bee, 10 miles on gravel road to the mine turn-off (Forestry road 259B) 1 mile west of Cleator, and 5 miles on a very rugged, circuitous "cat" road to the workings.

Phoenix is a major rail and highway terminus and distribution center.

Topography in the immediate mine area is moderately rugged, rising about 1500 feet from 4000 feet ASL on the Crown King/ Cleator road 1 mile to the southeast to 5500 feet immediately west of the surface exposure of the East Zone. Selection of drill sites is limited by topography.

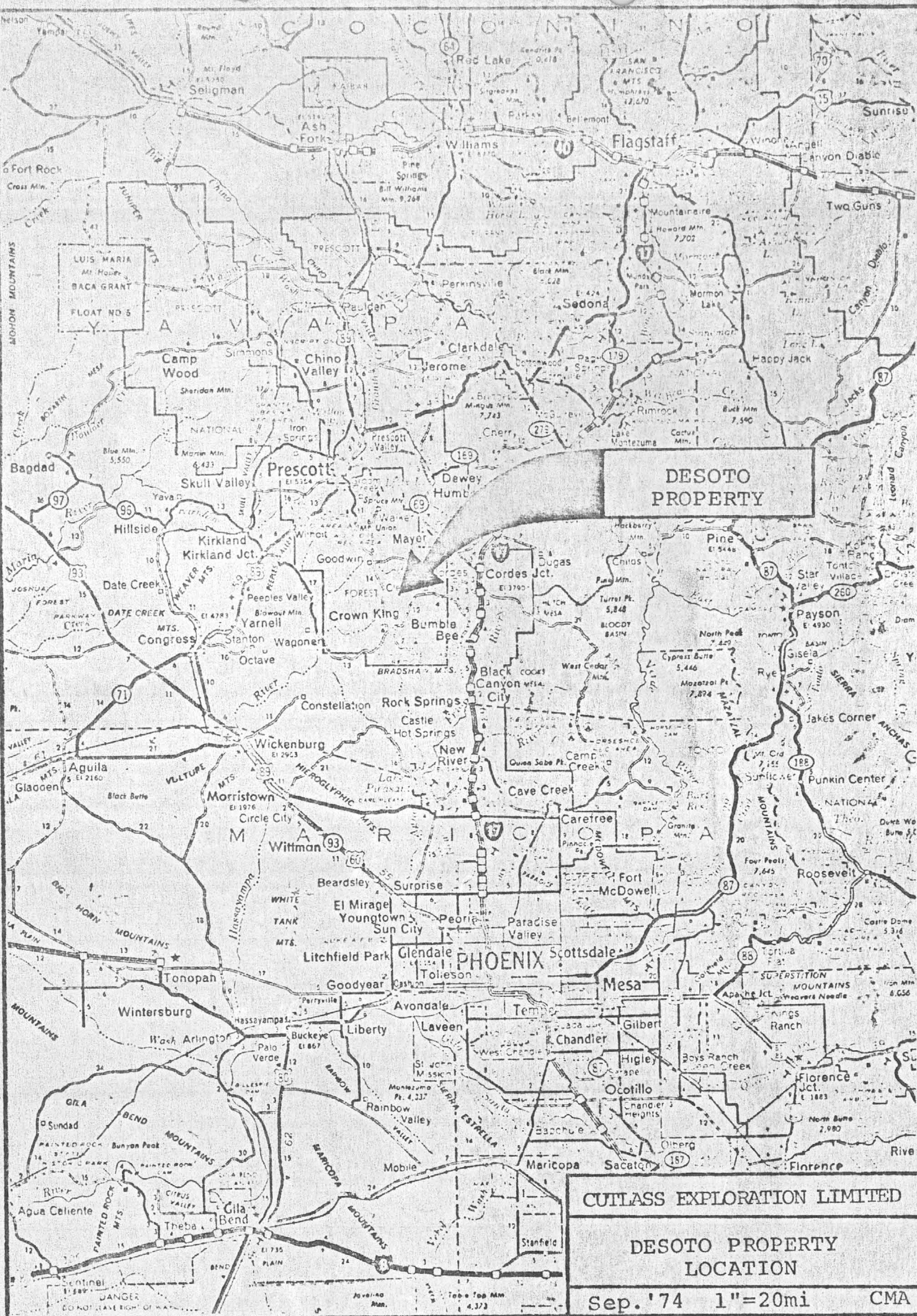
The mine area is underlain by Precambrian metavolcanic rocks referred to as the Yavapai Schist, striking about N30°E and dipping 60° to 70° westerly. Rhyolite, quartz-sericite schist, quartz-chlorite schist, and quartz-biotite schist predominate. The strongly foliated rocks are highly silicous and competent, and provide unusually stable hangingwalls for the open stopes which were mined 40 to 80 years ago.

In the East Zone primary sulphide mineralization is principally pyrite and chalcopyrite, which occurs as small, semi-massive lenses, streaks and disseminations within fairly high grade (over 1% Cu) lenticular zones or shoots to 40 feet in width, 150 feet in strike length, and several hundred feet in plunge length. Quartz-chlorite schist is mapped as the principal host rock. Low grade pyrite/chalcopyrite mineralization in disseminations and streaks of uncertain continuity also occur in areas between the major shoots. Abundant fresh sulphides were observed on the 200 Level.

At surface, the copper mineralization has been oxidized principally to malachite with lesser azurite, chrysocolla, and other secondary copper minerals. High grade shoots are readily demarcated by sharp colour contrast. Leaching, downward migration, and redeposition of the copper values probably enlarged the primary shoots, and both diamond and percussion drilling indicate that low grade mineralization occurs between the major highgrade zones.

In the West Zone crosscut (5260 elevation), much of the mineralized zone containing both oxide and sulphide copper mineralization appeared to be quite intensely fractured with platey fractures at ½-inch spacings roughly paralleling the regional trend.

Plans, cross sections, and longitudinal projections at a scale of 1"=100' (reduced from 1"=50'), Figures 2 to 17, accompany this report.



(from Arizona
Road Map 1974)

Fig. 1

RESERVES

East Zone

A preliminary calculation of mineral reserves in the East Zone from surface to the 200 Level (5160 elevation) was made by the writer based on cross sections G to O at 50-foot intervals. Detailed assay results were available for only 8 of the 25 vertical percussion holes within the zone, varying in length from 30' to 220' (total 3380'), and 2 of the 4 vertical diamond drill holes, varying in length from 170' to 510' (total 1240'). Reason for the short percussion holes, most of which stopped in good grade mineralization, is uncertain. Based on stope outlines, dip of the shoots was taken to be 70° westerly on cross section, and plunge 75° to 80° southerly on longitudinal projection. Strike continuity of the intersections was substantiated by constructing a plan at the 5300 elevation. Since only intersection averages were available for all of the holes, a true cut-off grade could not be employed; however, the lowest grade intersection within the zone was 0.28% Cu over 60 feet, and the next lowest 0.41% Cu over 50 feet. Basis for the mineral/mineable reserve calculation is as follows:

Category		Description	Tonnage	%Cu_T*	
Drill proven	I	Drill intersections projected down dip only through the vertical length of the intersection, and 25 feet on strike in two directions.	426,000	0.94	
Drill probable	II	Drill intersections projected 100 feet vertically down dip, and a maximum of 50 feet on strike in one direction (25 to 75 feet from the intersection).	317,000	0.97	
Drill possible	III	Drill intersections projected a further maximum 100 feet vertically down dip (to the 200 Level), and a maximum of 50 feet on strike (25 to 75 feet from the intersection).	66,000	1.57	
Subtotal I+II+III			809,000	1.00	
Possible	IV	Internal area untested because of inaccessibility (topography).	545,000	0.90	(i)
	V	Area between sections G and J (150') untested because of inaccessibility (topography and stoping).	143,000	0.65	(ii)
	VI	Mineralization below the 200 Level requiring no further waste removal when the entire mineral reserve is mined by open pit to the 200 Level.	184,000	0.96	(iii)

* Cu_T = total copper.

Category		Description	Tonnage	%Cu_T*
Subtotal IV+V+VI			872,000	0.88
Total I+II+III+IV+V+VI			1,681,000	0.93
Less stoped tonnage (iv)			116,000	
Net mineable reserve			1,565,000	0.93
Waste	VII	Internal - less than 0.20% Cu_T. Interpretation of detailed assay data could indicate other internal segregateable waste zones.	73,000	
	VIII	External - material removed when the entire mineral reserve (I to V, above) is mined by open pitting (steep, 55° final pit slope justified by the shallow depth of the pit and competent wall rocks).	1,722,000 (v)	
Total VII+VIII			1,795,000	

Waste/Ore ratio $\frac{1,795,000}{1,565,000} = \underline{1.2/1}$

Based on section L, a 50° final pit slope would result in an approximate 15% increase in external waste:

$$\frac{2,053,000}{1,565,000} = \underline{1.3/1}$$

(i) The internal untested area, IV, occurs in the highest grade, partially stoped zone. The faces and walls of the stopes examined show that high grade oxidized copper mineralization remains behind. Two older cross sections in the stoped area (appendix i) show assays for 4 diamond drill holes, one of which (DDH#1, -15° easterly) crosses most of the above area, averaging 0.65% Cu over 59 feet; and underground sampling at the 100 Level on the same section shows 1.32% Cu over 47' horizontally (weighted horizontal average 0.93% Cu). This grade agrees with the 0.94% Cu in the proven category, I, above, and it is reasonable to apply a grade of 0.90% Cu to this block.

(ii) Grade of the undefined mineralization southerly from section I, area V, should approximate the average 0.67% Cu on section I. Accordingly, it is reasonable to apply a grade of 0.65% Cu to this material.

(iii) For mineralization below the 200 Level within the confines of the trial pit, block VI, the grade should be similar to the average of categories I to IV, inclusive, namely 0.96% Cu.

(iv) Based on stope outlines on the cross sections, approximately 105,000 tons of ore appear to have been extracted by past producers from the above reserve. Mieritz reported a total production of approximately 290,000 tons averaging 3.34% Cu, of which the above tonnage represents 36%. This proportion is visually consistent with stope outlines above the 900 Level. Allow a total reserve reduction of 40% or 116,000 tons, with no change in resultant grade.

(v) Calculations on the trial pit are approximate and preliminary, only, and were undertaken to provide a rough measure of the stripping ratio involved in extracting the total mineral reserve from surface to the 200 Level, so that preliminary calculations on production costs and profitibility could be made. Detailed assay data, superior topographic and survey control, and additional fill-in drilling both for reserve and stope location purposes will be required for more advanced pit design. While the presence of underground stopes complicates the mining system, in the writer's opinion, it does not detract from the obvious merit of extracting the oxidized portion of the East Zone by open pit mining.

Additional reserves of sulphide copper unquestionably occur below the previously calculated pit: from section H to O (350'), no drill holes cross the zone; and from section A to H (350'), 16 intersections in 3 surface and 9 underground holes in the partially stoped area from the 600 Level to the 900 Level average 1.79% Cu_T over an average core length of 17 feet. Indicated true horizontal width of the intersections varies from 6 to 35 feet, averaging 19 feet, and at least 3 parallel high grade zones are indicated. Inadequate assay data is available on the inter-zone material to permit estimation of bulk grades over aggregate horizontal widths to 160 feet; however, one surface hole (DDH 11) averaged 0.78% Cu_T over 130 feet near the 900 Level. Much additional, costly, surface and/or underground diamond drilling, the latter requiring substantial underground development for drill sites, is required before a meaningful reserve can be calculated for the East Zone below the 200 Level. Certainly, it is both unrealistic and unsupported to project the 250-foot wide oxidized zone below the level.

West Zone

Because reserves in the West Zone could influence the selection of a production system, some preliminary calculations were made to provide an indication of the zone's reserve potential. On sections D to K, the weighted average grade of 16 drill holes (5 vertical percussion, 6 inclined hammer percussion, and 5 diamond) was 0.63% Cu_T; however, it is the writer's opinion that the true grade may be significantly higher. Average of the 5 percussion holes, only one of which cut the entire zone (at the south extremity), was 0.39% Cu_T; average of the 6 inclined hammer percussion holes, most of which cut the entire zone, was 0.61% Cu_T; and average of the 5 diamond drill holes, most of which cut the entire zone, was 0.86% Cu_T. I believe the latter grade, which also must include some sludge losses, is probably more representative of the true grade of the West Zone. Any further drill testing of this zone should be with coring equipment using mud as the circulating medium.

On the same basis as the East Zone, mineable reserves in all categories to the 5160 elevation, plus that amount below the 5160 requiring no further stripping, is about 900,000 tons. With steep, 55° final pit slopes, the waste/ore ratio is slightly less than 1.4/1; and, based on section H, a 50° final pit slope would result in an approximate 25% increase in external waste removal, yielding a waste/ore ratio of 1.7/1. To the deepest intersection at about elevation 4800, a further 1,100,000 tons is indicated, and the zone is open to depth. Total reserve for the West Zone is approximately 2,000,000 tons averaging 0.6 to 0.9% Cu_T.

OXIDE/SULPHIDE RATIOS

East Zone

As previously mentioned, 4 diamond drill holes and 25 percussion holes were used in the reserve calculation. Although the writer has not examined sample rejects from the core or percussion holes, it appears from examination of fresh surface mineralization and underground specimens that, because of the competent nature of the host rock and the occurrence of oxidized copper minerals, chiefly malachite, as fine permeations rather than coarse fracture fillings, losses in the circulating medium (mud) with diamond drilling and in the dust in percussion drilling have not been excessive. Nevertheless, some loss of readily slimed or dusted oxidized copper, possibly 10%, is inevitable; and on that basis the reserve grade would be closer to 1.0% Cu_T. Similarly, based on comparison of the variation of individual sample grades with the two types of drilling, contamination of successive samples in the percussion drilling does not appear to be significant. Assay breakdowns were available for 4 older diamond drill holes (appendix i), below, which were not used in the reserve calculations since their location with respect to the current grid was not established.

Hole	Dip	Intersection ft	$\%Cu_T$	$\%Cu_{ox}$*	Cu_{ox}/Cu_T %
DDH#1	-15°	157	0.57	-	-
SW 1	-90°	247	1.21	0.94	78
SW 2	-90°	126	0.95	0.56	59
SW 3	-90°	172	1.50	0.84	56
Average			1.09		
			1.24	0.82	66

The overall average of 1.09% Cu_T is somewhat higher than the reserve average of 0.93% Cu_T, or the upgraded 1.0% Cu_T (due to losses).

Composite pulp samples from the recent vertical core and percussion drilling yielded the following results (appendix ii):

Composite	Depth	$\%Cu_T$	$\%Cu_{ox}$	Cu_{ox}/Cu_T %
DDH 1-4	0-50	0.34	0.30	88
	50-100	0.91	0.79	87
	100,plus	0.87	0.75	86
	Average	0.71	0.61	86
P 1-18, 29, 33, 34	0-50	0.42	0.37	88
	50-100	0.43	0.24	56
	100,plus	0.74	0.25	34
	Average	0.53	0.29	55

		$\%Cu_T$	$\%Cu_{ox}$*	Cu_{ox}/Cu_T %
Average		0.62	0.45	73
Average	Old & recent	0.93	0.64	69

There are obvious anomalies in all of the above data, for which a multitude of explanations are possible; however, in the absence of definitive check data, it is reasonable for preliminary calculations to assume that 70% of the copper is acid soluble.

West Zone

Composite samples from 2 vertical percussion holes and 2 inclined hammer percussion holes above the 5160 elevation, averaged 0.46% Cu_T and 0.28% Cu_{OX}, equivalent to 61% "oxide" copper. An exceptionally long, 170-foot intersection in hole H-2 which extended well into the footwall of the main zone immediately above the 5160 elevation, contained only 27% "oxide" copper. This hole is not believed by the writer to be representative.

As previously discussed and demonstrated, it also is the writer's opinion that the significantly lower average values in the vertical percussion holes (0.39% Cu_T) and inclined hammer percussion holes (0.61% Cu_T), compared to the diamond drill holes (0.86% Cu_T), represent, at least partly, losses of oxidized copper in the highly fractured host rock (½" fracture spacings); and there is a reasonable possibility that the proportion of "oxide" copper will approximate that of the East Zone, namely 70%.

* Cu_{OX} = "oxide" copper (this value should be referred to more correctly as acid soluble copper, Cu_{AS}, the magnitude of which is dependent, in part, on the analytical technique employed - limiting mesh size, acid type, acid concentration, pulp density, and time).

ASSOCIATED VALUES

Values in gold, silver, and zinc accompany the primary copper mineralization in both the East and West Zones. While considerable recent analytical work has been done on intersections of sulphide copper, none has been done on copper mineralization from the shallow oxidized zones. Insoluble precious metal values (gold and silver) could be concentrated somewhat in the oxide zone, and readily soluble zinc values could be depleted. A summary of all recorded data (appendix iii) follows:

Zone	Metal	Source	Samples No.	Samples Total length (ft)	Analyses Cu_T %	Analyses Metal	Ratio Metal/Cu_T
East	Au	Production (Mieritz)	290,000 tons		3.0	0.05 ozAu/T	0.017
		Underground drilling					
		Individual samples	19	190	1.41	0.059	0.042
		Composite samples	4	184	1.38	0.052	0.038
		Average	23	374	1.39	0.056	0.040
	Ag	Production (Mieritz)	290,000 tons		3.0	1.20 ozAg/T	0.40
		Underground drilling					
		Individual samples	19	190	1.41	0.52	0.37
		Composite samples	4	184	1.38	0.54	0.39
		Average	23	374	1.39	0.53	0.38
	Zn	Underground drilling					
		Individual samples	23	233	1.12	0.49 % Zn	0.43
West		Surface drilling					
	Au	Composites	15	1000	0.70	0.042 ozAu/T	0.060
	Ag		15	1000	0.70	0.25 ozAg/T	0.34
	Zn		12	926	0.67	0.26 % Zn	0.39

In the East Zone there is a substantial difference between the Au/Cu ratio from production statistics (0.017) and recent assaying (0.040); however, the writer is uncertain whether the production figure represents a"head" grade or a "recovered" or "paid for" grade, and about 40% of the production tonnage was from the oxidized zone above the 200 Level. Individual assay ratios in the sulphide zone varied from a low of 0.004 to a high of 0.254, and 0.03 is a reasonable average figure. It is not known whether this ratio will change significantly in the oxide zone. Good agreement was obtained for Ag/Cu ratios from production statistics and recent assaying, at about 0.4; and the Zn/Cu ratio, for which there is no production record, also is 0.4.

East Zone metal ratios:

1.0% Cu_T, 0.03 ozAu/T, 0.4 ozAg/T, 0.4% Zn.

Since the reserve grade is approximately 1.0% Cu_T (including provision for a 10%, actually 7.5%, loss of copper values in the percussion drilling) the above Au, Ag, and Zn values may be representative.

No comparative production data is available for the West Zone, and, again, it is uncertain whether ratios in the primary sulphide zone will be the same as those in the oxide zone. The Au/Cu_T ratio appears to be somewhat higher in the West Zone than in the East, the Ag/Cu_T is marginally lower, and the Zn/Cu_T ratio is the same.

West Zone metal ratios:

1.0% Cu_T, 0.06 ozAu/T, 0.3 ozAg/T, 0.4% Zn.

Since the reserve grade is somewhere between 0.6 and 0.9% Cu_T, expectable associated values are as follows:

0.6% Cu_T, 0.04 ozAu/T, 0.2 ozAg/T, 0.2% Zn.

0.9% Cu_T, 0.05 ozAu/T, 0.3 ozAg/T, 0.4% Zn.

PRODUCTION SYSTEM

Discussion

The most readily accessible, best defined, and highest grade mineable copper reserve on the DeSoto property is that in the East Zone above the 200 Level (5160 elevation): approximately 1.5 million tons in all categories (27% drill proven), with a 1.2/1 stripping ratio (55^{o} final pit slope), averaging about 1.0% Cu_T. Approximately 70% of the copper in this reserve is "oxide" in form and should be readily soluble in a sulphuric acid solvent. Test work is required to establish the leaching characteristics of the mineralization: acid consumption, optimum acid concentration(s), recovery, residence time, and fragment size(s).

Drill intersections in sulphide mineralization below the 200 Level generally show significantly narrower mineralized widths, and, with the obvious very large increase in waste removal required, it is doubtful that open pitting below the trial pit bottom (5035 elevation) will be feasible.

In the West Zone, a further mineable reserve of 0.9 million tons in all categories (no breakdown), is available to the same 5160 elevation, with a 1.4/1 stripping ratio (55^{o} final pit slope), and an average grade somewhere between 0.6 and 0.9% Cu_T. At least 1.1 million tons of sulphide copper mineralization of comparable grade, mostly in the possible category, occur between the 5160 elevation and the deepest intersection at elevation 4800. Mining of the bulk of this reserve must be by underground techniques due to excessive stripping ratios; and the zone is open to depth below the 4800 elevation.

In general terms, consideration of the magnitude, grade, and probable net smelter value (copper only) of the combined reserve mineable by open pit techniques, 2.5 million tons of 0.9% Cu_T, dictates that only a production system of relatively modest capital cost, say $2.0 million, maximum, be considered. Analytical and metallurgical testing of representative samples from the oxide zones of both deposits would be required both to establish associated values, particularly gold, and to determine whether any of the values were recoverable economically. It is equally apparent that the relatively low average copper grade (1.0% Cu_T and 0.7% Cu_{OX} in the East Zone) would require production at a daily rate of at least 1000 tpd in order to yield an acceptable cash flow. The indicated reserve is compatible with production at this rate, giving an open pit life expectancy of about 7 years (4 for the East Zone, and a 3 for the West Zone). No remotely "conventional" plant employing crushing, grinding, and processing with this daily throughput could be constructed for the allowable capital cost, and an alternative, lower capital cost system is required.

Clearly defined, high grade zones to 30 feet in width are evident at surface, in the underground workings, and in the percussion and diamond drill hole intersections, and it is very likely that high grade and low grade products could be segregated readily in a small open pit. To provide estimates of the relative quantities and grades of such material, all drill hole intersections within the deposit were averaged separately using 1.0% Cu_T as the cut-off, as follows:

	Total Intersection Length (feet)	Distribution %	Average Grade $\%Cu_T$
High grade	767	29	1.78
Low grade	1918	71	0.65
Combined	2685	100	0.97

Because no breakdown was available for some of the averaged intersections, it is very likely that a smaller proportion of higher grade material could be segregated. The overall average is in very close agreement with the reserve grade, and, through selective mining, it appears reasonable that either 30% of the reserve averaging 1.8% Cu_T and 70% averaging 0.65% Cu_T (total 1.0% Cu_T), or 25% averaging 2.0% Cu_T and 75% averaging 0.65% Cu_T (total 1.0% Cu_T), could be segregated and processed independently.

Because the material mined initially will contain oxidized copper minerals almost exclusively, acid leaching/cementation is the obvious processing technique to employ, and offers the greatest potential for meeting the financial constraint. In order of increasing capital and operating costs, and increasing recovery, in situ leaching, heap leaching, combination heap leaching/modified vat leaching, and possibly combination heap leaching/agitation leaching all have valid potential application. Test work is required to substantiate the applicability of acid leaching through establishment of the basic economic leaching parameters, namely, acid consumption and acid soluble copper.

In Situ Leaching

While the physical orientation of both oxidized zones is favourable for in situ leaching, the nature of the finely permeated oxidized copper mineralization in the East Zone, at least, is not. Expectable low recovery at 20 to 30% of the "oxide" copper (15 to 20% of the total copper) over 1 or 2 years, low production rates (about 3T Cu/day, pure basis), probable widely variable production rates, inflexibility of the system, the critical nature of each production blast (and disasterous consequences of error), and the relatively high cost of preparing the site for blasting, do not, in the writer's opinion, favour application of in situ leaching as the primary recovery system.

It is very important to appreciate, however, that once open pit mining for any of the other more costly production systems has commenced, it always is possible to fall back on in situ leaching as a salvage system without incurring the substantial site preparation costs, above, or any other plant costs. Engineering, drilling, blasting, and application of solution sprays would be the only additional costs.

Heap Leaching

While the topography at the property is moderately rugged, a number of potential heap and dump sites were observed which may or may not have been solely on the DeSoto claim group. For site and production planning, it is essential that the property and adjacent areas be mapped topographically at a scale of 1"=200', and the immediate mine area at a scale of 1"=100' (plus enlargements to 1"=50 feet). Reservoir sites are at a premium on the property, and it may or may not be possible to utilize some of the underground workings for reservoirs. The winze from the 600 Level to the 900 Level is full of water, and the underground workings reportedly make a significant volume of water. An accurate measure of the inflow should be obtained, since additional wells would be required to make up the deficit. Depending on evaporation losses (15 to 20% is expectable) as well as on many other factors related to the leaching characteristics of the mineralization, process water requirements for the production of 5T Cu/day (pure basis) should be substantially less than 100 gpm.

Because malachite is the predominant oxidized copper mineral, leaching rates should be relatively rapid; however, the apparent fine permeation, as opposed to more typical fracture fillings which often may be exposed at relatively coarse fragment sizes, suggests that recovery will be directly related to fragmentation, and could be low. Short term test work on representative (physically and chemically) samples of low grade and high grade mineralization is required to determine if maximum recoveries from run-of-mine material under flooded conditions are acceptable. It is conceivable that single-stage crushing of the high grade mineralization, at least, might be necessary, in which case conveyor stacking of this product also might be advantageous. The highly siliceous mineralization should yield uniformly permeable heaps with minimal tendency for solution channeling.

Recovery of 40 to 50% of the "oxide" copper (30 to 35% of the total copper) in a 4-month leaching cycle is possible; and, under these conditions, production of 5T Cu/day would require continuous ore mining at a rate of about 1000 to 1250 tpd initially, with a 150,000-ton starting heap, increasing to 1500 to 1800 tpd as the proportion of sulphide copper increased to the East Zone average (0.3% Cu_S). With an overall waste/ore ratio of 1.2/1, continuous mining at 2500 to 4000 tpd is indicated.

Heap Leaching/Modified Vat Leaching

As previously discussed, there is a good possibility that high grade oxidized copper mineralization can be segregated from low grade in a small East Zone pit yielding either a 30/70% combination grading 1.8/0.65% Cu_T or a 25/75% combination grading 2.0/0.65% Cu_T. Vat leaching of the high grade material, following

2-stage crushing, should yield at least 80% recovery of the "oxide" copper (55% of the total copper); and, with the readily acid soluble malachite, residence time should be relatively short, possibly only 2 days for the leaching cycle. Run-of-mine low grade material would be heap leached, as above, probably yielding 40% recovery in a 4-month leach cycle.

The modified vat leaching system referred to is a simple, continuous, countercurrent system requiring construction of only a single vat.

Segregating a much-reduced tonnage of high grade mineralization for independent processing enhances very substantially the probability of re-processing the tailings for recovery of associated values, particularly gold, as well as for recovery of sulphide copper, as the proportion increases with depth. Significantly reduced grinding costs for the highly siliceous material should result from pre-grind acid leaching.

For production of about 5T Cu/day, pure basis, continuous mining of high grade and low grade mineralization at 1000 tpd (250/750 or 300/700 tpd) is required, plus an additional 1200 tpd of waste (2200 tpd total). Overall recovery of the "oxide" copper should be at least 60%, equivalent to 42% of the total copper.

Heap Leaching/Agitation Leaching

Only through the treatment by agitation leaching of a relatively small tonnage of high grade oxidized copper mineralization, say 250 tpd averaging 2.0% Cu_T, and through judicious purchasing of second-hand equipment, could a maximum mine/plant capital cost of \$2.0 million be achieved. Even with a relatively coarse mesh-of-grind, say minus 10 mesh, comminution cost for the very siliceous ore is likely to be high. While it does not appear that a slime problem exists, this aspect requires close attention since a sophisticated solid/solution separation system could not be tolerated. Recovery of the "oxide" copper by agitation leaching should be at least 90%, equivalent to 46% of the total copper, including copper recovered from heaps (750 tpd averaging 0.65% Cu_T).

As the proportion of sulphide copper increases, flotation treatment of the leach tailings, with or without regrinding could be justified; and, when the oxidized ore is depleted, the possibility exists of expanding throughput substantially and changing over to a full flotation system.

Leach residues should be tested for possible recovery of associated values, particularly gold. If flotation were justified economically at some later stage, the associated values in gold, silver, and zinc could be particularly important, and their recovery probably would be simplified.

SUMMARY AND CONCLUSIONS

1. In the East Zone, the mineable reserve in all categories (27% drill proven) to the bottom of a trial open pit designed to extract all of the mineral reserve to the 200 Level (5160 elevation) is approximately 1.5 million tons averaging about 1.0% Cu_T and 0.7% Cu_{OX}. With steep, 55^o pit slopes, justified, in the writer's opinion, by the shallow depth of the pit and the competent wall rocks, a 1.2/1 waste/ore ratio is indicated, increasing to about 1.4/1 with 50^o pit slopes. The presence of open stopes in the pit area, possibly aggregating 116,000 tons, complicates, but does not preclude open pit mining.

2. Although partially stoped sulphide copper in highgrade shoots extends from the above pit floor to the 900 Level (4550 elevation), there is inadequate data on which to base a reserve calculation. However, all indications point to significantly narrower overall widths than the 200 to 300 feet in the expanded oxide zone, and it is doubtful that open pitting below the trial pit floor (5035 elevation) will be possible. Sixteen intersections in at least 3 parallel zones averaged 1.8% Cu_T over an average core length of 17 feet, and the indicated true horizontal width of the intersections varied from 6 to 35 feet, averaging 19 feet. Grade of the inter-zone material is uncertain, although one hole averaged 0.78% Cu_T over 130 feet near the 900 Level.

3. In the West Zone, the mineable reserve in all categories (not broken down) to the bottom of a similar trial open pit designed to extract all of the mineral reserve to the 200 Level (5160 elevation) is approximately 0.9 million tons averaging somewhere between 0.6 and 0.9% Cu_T. The average grade is uncertain because the vertical percussion holes averaged 0.39% Cu_T, the inclined hammer percussion holes averaged 0.61% Cu_T, and the diamond drill holes averaged 0.86% Cu_T. Losses in the highly fractured host rock could be responsible for the grade discrepancy, and the writer believes that 0.9% Cu_T probably is closer to the actual grade. The proportion of "oxide" copper probably is similar to, or slightly less than, the East Zone at about 70%. With 55^o pit slopes the waste/ore ratio is 1.4/1, increasing to about 1.7/1 with 50^o slopes.

4. A further 1.1 million tons of sulphide copper mineralization of comparable grade, mostly in the possible category, is available to the deepest intersection at the 4800 elevation, and the deposit is open to depth. Due to excessive open pit stripping ratios, it would be necessary to mine most of this material by underground techniques.

5. Associated values in gold, silver, and zinc accompany the copper in the sulphide zones, and it is warranted to establish the

associations in the oxide zones of both deposits.

East Zone	1.0% Cu_T, 0.02 to 0.04 ozAu/T, 0.4 ozAg/T, 0.4% Zn.
West Zone	0.9% Cu_T, 0.05 ozAu/T, 0.3 ozAg/T, 0.4% Zn.

6. The high proportion of oxidized copper mineralization, principally malachite, in the highly siliceous oxidized zones of both deposits from surface to the 200 Level, readily extractable by open pit mining, indicates that processing by acid leaching/cementation could be viable. In the absence of data on the leachability of the mineralization, preliminary tests should be conducted to establish the basic leaching parameters, acid consumption and acid soluble copper, followed by bench-scale tests to define the best recovery system(s) and operating parameters.

7. In consideration of the total reserve mineable by open pit techniques, approximately 2.5 million tons averaging 0.9% Cu_T, maximum capital cost for a production system should be about $2.0 million. In situ leaching, heap leaching, combination heap leaching/modified vat leaching, and possibly combination heap leaching/agitation leaching all have potential application. Production would be initiated on the East Zone.

   <u>In situ leaching</u>. Because of the relatively high site preparation cost, expectable very low recovery (20 to 30% of the oxide copper in 1 or 2 years), low production rate (about 3T/day), etc., in situ leaching should not be employed as the primary recovery system, but could be employed as a backup salvage system, if required.

   <u>Heap leaching</u>. Assuming 40 to 50% recovery of the oxide copper values in a 4-month leaching cycle, production of 5T Cu/day would require continuous ore mining at an initial rate of 1000 to 1250 tpd, with a 150,000-ton starting heap, increasing to 1500 to 1800 tpd as the proportion of sulphide copper increased Total ore and waste mining would be at a rate of about 2500 to 4000 tpd.

   <u>Heap leaching/modified vat leaching.</u> At an ore mining rate of 1000 tpd (2200 tpd ore and waste), it is possible that 250 tpd averaging 2.0% Cu_T and 750 tpd averaging 0.65% Cu_T could be segregated readily in a small East Zone pit, and processed independently by modified vat leaching, yielding 80% recovery of the oxide copper with a 2-day leaching cycle, and by heap leaching, as above, yielding a total recovery of 60% of the oxide copper, and producing 5T Cu/day. Re-processing the vat residue for gold and/or sulphide copper values might be feasible at some stage.

Heap leaching/agitation leaching. In the same manner as above, 90% recovery of the oxide copper would be expectable, yielding, with heap leaching, a total recovery of 65% of the oxide copper, and producing over 5T Cu/day. Integration with a flotation circuit as the proportion of sulphide copper and associated values increased, could be accomplished at less cost and with greater facility than with the other systems. However, even with the judicious purchase of second-hand process equipment, mine/plant capital costs could exceed $2.0 million; and processing costs at the limited production rate could be prohibitive.

CERTIFICATION

I, CHRISTOPHER MACKENDRICK ARMSTRONG of the City of Vancouver, Province of British Columbia, do hereby certify:

THAT I am a practicing Geological Engineer residing at 4085 West 29th Avenue, Vancouver, British Columbia, V6S 1V4, Canada.

THAT I am a registered Professional Engineer in good standing in the Provinces of British Columbia and Ontario.

THAT I received the degree of B.Sc. in Geological Engineering from Queen's University, Kingston, Ontario in 1960, and practiced my profession continuously in the period between leaving university in 1959 and returning to university in 1966.

THAT I enrolled in the Department of Mineral Engineering at the University of British Columbia in 1966, and in the period to 1969 completed course work and research work requirements in an M.A.Sc. program, specializing in bacterial/acid leaching systems; thesis writing was not completed; post graduate courses in economic geology and Nortn American geology also were taken and completed.

THAT since leaving university in 1969, I have practiced my profession both as a Geological Engineer and as a Specialist/Advisor in ambient temperature/pressure leaching systems.

THAT the following is a true record of my employment and experience:

1957	4 mos. Junior Geologist. Noranda Mines Ltd. Noranda, Quebec.
1958	4 mos. Party Chief. Hollinger North Shore Exploration Co. Ltd. New Quebec and Labrador.
1959-1961	27 mos. Assistant Geologist. Pickle Crow Gold Mines Ltd. Pickle Crow, Ontario. Teck Corporation Ltd.
1961-1962	9 mos. Assistant Geologist. Willroy Mines Ltd. Manitouwadge, Ont.
1962-1964	28 mos. Chief Geologist. Metal Mines Ltd. Werner Lake, Ontario. Consolidated Canadian Faraday.
1964-1966	24 mos. Chief Geologist. Tegren Goldfields Ltd. Kirkland Lake, Ontario. Teck Corporation Ltd.
1967	6 mos. Project Geologist. McLeese Lake property, B. C. Geophysical Engineering & Surveys Ltd. Teck Corporation Ltd.
1969-1970	13 mos. Laboratory Manager, Chief Geologist, and Consulting Engineer. S. M. Industries Ltd. Vancouver, B. C.
1970-1974	4 yrs. Independent Consulting Engineer.

THAT I do not have any interest, direct, indirect, or contingent, in the securities or properties of CUTLASS EXPLORATION LIMITED.

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG

Dated at Vancouver this 18th Day of September, 1974

C. M. Armstrong, P.Eng.

59. The Geology of the Iron King Mine

PAUL GILMOUR,* ARTHUR R. STILL†

Contents

* Texas Gulf Sulphur Company, New York, N.Y.
† Still & Still, Prescott, Arizona.

Illustrations

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ABSTRACT

The ore deposit of the Iron King mine occurs in a group of steeply-dipping metamorphosed eugeosynclinal volcanic and sedimentary rocks of Precambrian age. Within this sequence, the ore deposit lies at the contact of a unit made up of rhyolitic tuff and interbedded andesite and a structurally underlying series consisting of andesitic tuffs, minor rhyolitic tuffs, and argillaceous sediments. Field evidence suggests that these rocks have been overturned. The ore deposit is made up of a series of overlapping, conformable or bodies consisting of massive pyrite and associated lenses of massive quartz. These contain recoverable amounts of gold, silver, lead, zinc, and copper. A parallel zone of copper mineralization occurs within the hanging-wall rocks.

The deposit is considered to belong to a large and important group characterized by mineralogical composition and the lithology and structural type of the host rocks. Evidence regarding the origin of the deposit is reviewed, and it is concluded that the ore bodies were

formed through the agency of volcanic hot springs on, or near, a submarine surface of deposition.

INTRODUCTION

General

Apart from their economic importance, the ore bodies at the Iron King mine are of special interest because they represent a type of deposit which, although commonplace in some other countries, is relatively rare in the United States, especially if only mines which are operating today are considered.

Location

The Iron King mine lies about one mile west of the village of Humboldt which, in turn is situated on Highway 69, 15 miles east-southeast of Prescott and 75 miles north-northwest of Phoenix or, more specifically, at 34°30′N and 112°15′30″W.

History and Production

As is true of most mines, the Iron King has had an interesting history. According to Anderson and Creasey (26, p. 90), some of the older inhabitants of the area believe that the original location was made around 1880

TABLE I. Production of Recoverable Metals, Iron King Mine, Humbolt, Arizona

Year	Tons	Gold (ounces)	Silver (ounces)	Lead (pounds)	Zinc (pounds)	Copper (pounds)
1606–38[1]	78,452	15,690	313,808	3,138,080	6,276,160	470,700
1638[2]	13,477	2,317	45,938	404,300	1,078,160	67,400
1939	70,227	9,911	272,604	1,872,680	5,854,020	351,120
1940	65,812	9,239	266,497	1,891,060	7,220,440	329,060
1941	69,159	9,720	331,746	2,320,040	7,617,100	345,800
1942	88,200	11,659	392,458	3,540,100	10,585,560	441,000
1943	73,721	9,167	307,465	3,164,380	10,095,300	220,720
1944	99,164	9,460	308,567	3,611,660	13,623,860	423,820
1945	117,287	13,068	436,506	5,259,640	16,156,180	455,280
1946	115,615	13,065	467,387	5,734,280	16,875,320	485,780
1947	122,368	15,298	533,642	6,194,880	16,925,320	411,820
1948	145,823	17,036	540,548	6,854,120	19,048,100	453,020
1949	175,111	21,432	737,925	8,414,680	23,547,440	546,660
1950	203,063	27,289	904,284	10,645,040	28,220,800	686,460
1951	202,581	27,135	764,731	9,528,680	26,075,380	657,100
1952	197,747	23,430	730,280	10,203,740	29,306,000	672,000
1953	190,735	26,703	730,515	10,528,000	27,008,000	610,000
1954	180,512	28,106	745,514	11,372,000	30,074,000	722,000
1955	222,909	31,296	884,949	12,170,000	32,902,000	758,000
1956	253,956	35,452	992,968	14,476,000	37,992,000	914,000
1957	300,729	38,644	1,118,712	16,540,000	47,696,000	1,082,000
1958	314,266	39,629	1,147,071	18,038,000	53,236,000	1,194,000
1959	299,981	38,728	1,124,929	17,338,000	51,476,000	1,140,000
1960	304,485	34,285	1,020,025	16,442,000	52,980,000	1,158,000
1961	235,885	22,857	702,937	11,700,000	39,912,000	1,038,000
1962	271,171	28,066	854,189	13,776,000	44,635,000	1,138,000
1963	280,807	27,463	901,390	12,468,000	39,200,000	1,010,000
1964	314,163	30,348	917,356	13,132,000	39,522,000	1,320,000
1906–64	5,007,406	616,493	18,494,491	250,757,360	735,138,140	19,101,740

([1] Production before milling.)
([2] Last 3 months of milling.)
Source of data: Shattuck Denn Mining Corporation.

ation

ne lies about one mile
Humboldt which, in turn
ay 69, 15 miles east-
nd 75 miles north-north-
more specifically, at
0"W.

Production

mines, the Iron King
history. According to
(26, p. 90), some of
f the area believe that
as made around 1880

rizona

Zinc (pounds)	Copper (pounds)
6,276,160	470,700
1,078,160	67,400
5,854,020	351,120
7,220,440	329,060
7,617,100	345,800
0,585,560	441,000
0,095,300	220,720
3,623,860	423,820
6,156,180	455,280
6,875,320	485,780
6,925,320	411,820
,048,100	453,020
,547,440	546,660
,220,800	686,460
,075,380	657,100
,306,000	672,000
,008,000	610,000
,074,000	722,000
,902,000	758,000
,992,000	914,000
,696,000	1,082,000
,236,000	1,194,000
476,000	1,140,000
980,000	1,158,000
912,000	1,038,000
635,000	1,138,000
200,000	1,010,000
522,000	1,320,000
138,140	19,101,740

for gold and silver. The initial mining apparently proved disappointing. In 1906 and 1907, the Iron King mine was operated by the Rev. Ben Blanchard, and in the latter year production amounted to 1253 ounces of gold, 35,491 ounces of silver, and 3933 pounds of copper. The mine was then inactive until World War I, during which it was operated for a short time. It was again idle until the late 1930's, when it was reopened by Fred Gibbs of Prescott. Shattuck Denn Mining Corporation purchased the Iron King in 1942. The mine has been in continuous production since then. A differential flotation mill was erected in 1939, and the capacity of the mill has gradually been increased from the original 225 tons per day to just over 1000 tons at the present time. Both zinc and lead concentrates are produced; the former containing recoverable amounts of cadmium and the latter gold, silver, and copper. The total production to date is shown on Table I. These data show that the overall grade of ore mined is 0.123 ounces of gold and 3.69 ounces of silver per ton, 2.50 per cent lead, 7.34 per cent zinc, and 0.19 per cent copper.

Previous Work

The fullest published accounts of the geology of the area around Humboldt and of the Iron King mine are those by Creasey (10,12) and Anderson and Creasey (26). A number of earlier papers, by Mills and Kumke and Mills, described the occurrence (5) and mining procedures (11,7,9) at the Iron King. The most recent account of the mining method is that by Mitchell (44) based on information supplied by C. R. Sundeen, the present mine manager.

No doubt a large number of reports on the Iron King have been written and are now lost or inaccessible. However, of the reports which survive, that written in 1941 by Louis J. Reber was outstandingly prescient (6). Useful contributions to the geology of the Iron King were also made by Bulmer and his colleagues (17).

LITHOLOGY OF HOST ROCKS AND DEPOSITS

General

Anderson and Creasey (26, p. 97) considered that the sulfides at the Iron King and United Verde belong to a class of deposits recognizable by mineralogical composition which includes the ore bodies of Rio Tinto in Spain, Rammelsberg in West Germany, Sullivan in British Columbia, and numerous deposits in the Shasta district in northern California.

The writers would emphasize that this class of deposit is also characterized by the lithology and orogenic setting of the host rocks and by the form and structure of the deposits themselves. Concordant, massive, pyritic sulfide deposits are typically associated with rocks of a eugeosynclinal environment—keratophyric and spilitic flows, tuffs and agglomerates, chert, iron formation, basic and ultrabasic intrusions, shale and graywacke. (It must be acknowledged that the term "eugeosynclinal" should be used with some caution when referring to Precambrian rocks, yet its usage seems justified if employed as a description for such an assemblage of rocks.) It would seem to follow that the host rocks of massive pyritic sulfide deposits, and possibly the deposits as well, have been subjected to orogenesis. It is not surprising, therefore, that the host rocks for most examples of this class of deposit are deformed and metamorphosed with the result that the original nature and structure are obscured. The rocks in and around the Iron King mine are no exception, and their mineralogical composition, texture and structure have been more or less influenced by metamorphism. Consequently, rock names are a mixture of sedimentary and volcanic terms on the one hand and metamorphic terms on the other (see surface geological map, Figure 1).

The vast majority of the consolidated rocks in the neighborhood of the Iron King mine are Precambrian in age and belong to the Yavapai Series as defined by Anderson and Creasey (26, p. 8). These authors divided the Yavapai Series into two groups, the Ash Creek Group which occurs in the massif of the Black Hills and the Alder Group which forms most of the western foothills of the Black Hills and the eastern foothills of the Bradshaw Mountains farther to the southwest. The Ash Creek Group consists of thick units representing two cycles of basaltic to rhyolitic volcanism overlain by pyroclastic deposits containing interbeds of chert and magnetic jasperoidal chert. The Alder Group is made up of thinly-bedded sediments of both detrital and volcanic derivation with some recognizable flows and tuffs. The relationships of the Ash Creek and Alder Groups are being studied by the U.S. Geological Survey.

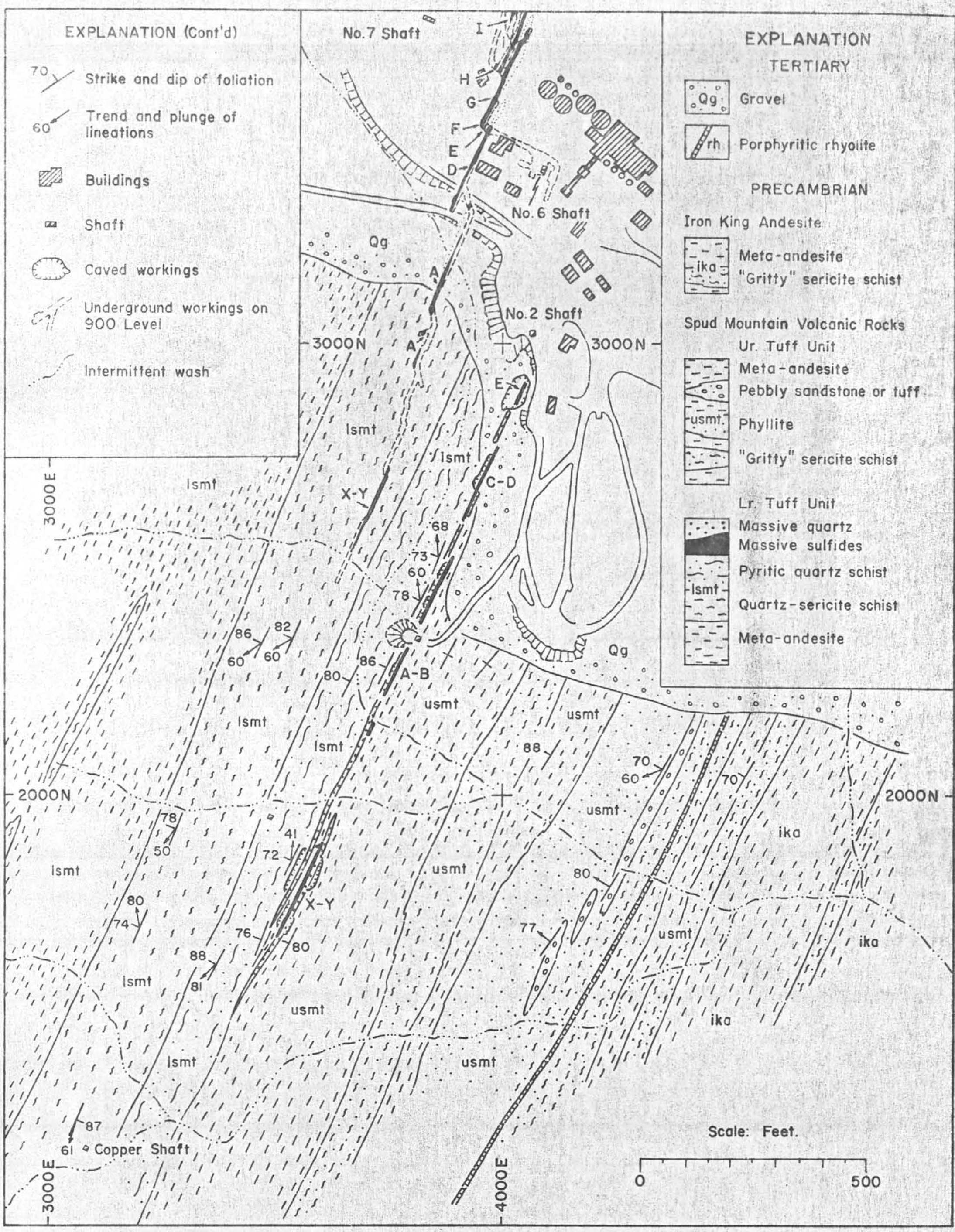


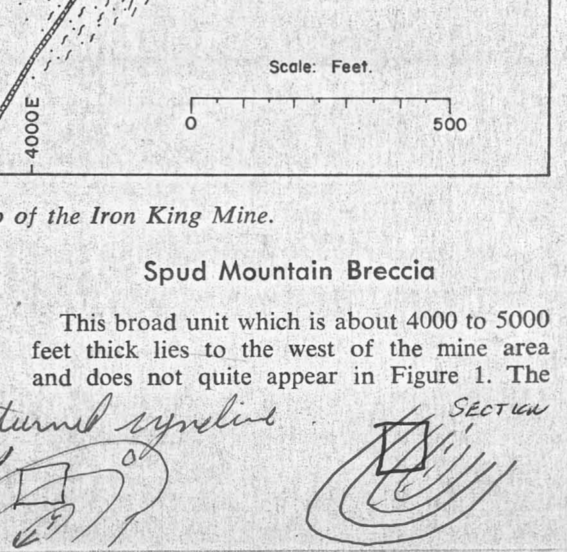
FIG. 1. *Geological Map of the Iron King Mine.*

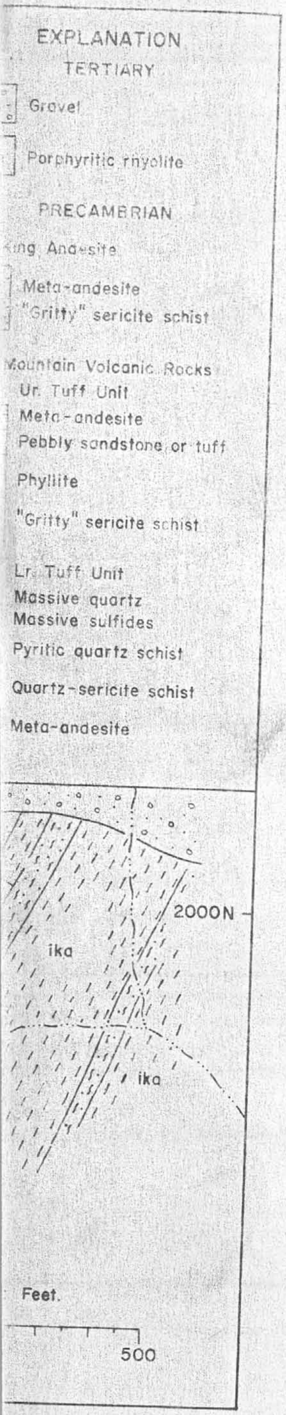
The schists in the vicinity of the Iron King mine all belong to the Alder Group, and, apart from one minor modification, these will be described using the subdivisions proposed by Anderson and Creasey (26).

Spud Mountain Breccia

This broad unit which is about 4000 to 5000 feet thick lies to the west of the mine area and does not quite appear in Figure 1. The

Note: Part of western limb of overturned syncline.





...n Breccia

...about 4000 to 5000 ...t of the mine area ...r in Figure 1. The

Spud Mountain Breccia is of volcanic derivation and is predominantly andesitic in composition. It may represent either flow breccias or welded ash flows.

Lower Spud Mountain Tuff

The writers propose to modify Anderson and Creasey's subdivisions by treating the Spud Mountain Tuff as two members, the Lower and the Upper, the former including the ore horizon at the Iron King mine (Figure 2). The Lower Spud Mountain Tuff succeeds the Spud Mountain Breccia on the east and has a thickness of about 2000 feet. The formation consists of principally chlorite-sericite schist and quartz-sericite schist, the former predominating to the west and the latter to the east. Locally the schists contain feldspar grains which attain a few millimeters in their longest dimensions.

There does not seem much doubt that the chloritic schists represent metamorphosed andesitic tuffs and crystal tuffs.

On the surface, the quartz-sericite schists weather white to light brown and in places they are heavily stained with limonite from the oxidation of the contained iron sulfides. Beneath the zone of weathering, these schists are found to be green and grey only slightly lighter in color than the metaandesites. There is some room for debate as to the origin of the quartz-sericite schists. Creasey considered the question at some length and concluded that they are hydrothermally altered meta-andesites (26, p. 158), although he did not rule out the possibility that some or all of them are metarhyolites (26, p. 162). Bulmer and his colleagues described the felsic schists containing visible feldspar crystals as intrusive porphyritic rhyolites (17). Creasey showed that significant differences in chemical composition separate the chloritic and siliceous schists, but he did not marshall any evidence to support the conclusion that these differences are epigenetic rather than original. The extremely sharp contacts between the quartz-sericite schist and the adjacent rocks, especially the putative original metaandesite, are difficult to reconcile with an epigenetic origin. On the other hand, the gradational contacts between the porphyritic and non-porphyritic quartz-sericite schist seem to militate against an intrusive origin for the former. It is here concluded that the quartz-sericite schists represent a series of metamorphosed rhyolitic tuffs ad crystal tuffs texturally similar to, but compositionally different from, the metamorphosed andesitic tuffs with which they are interbedded.

Ore Horizon

The massive sulfide lenses which are exploited at the Iron King mine lie at the contact

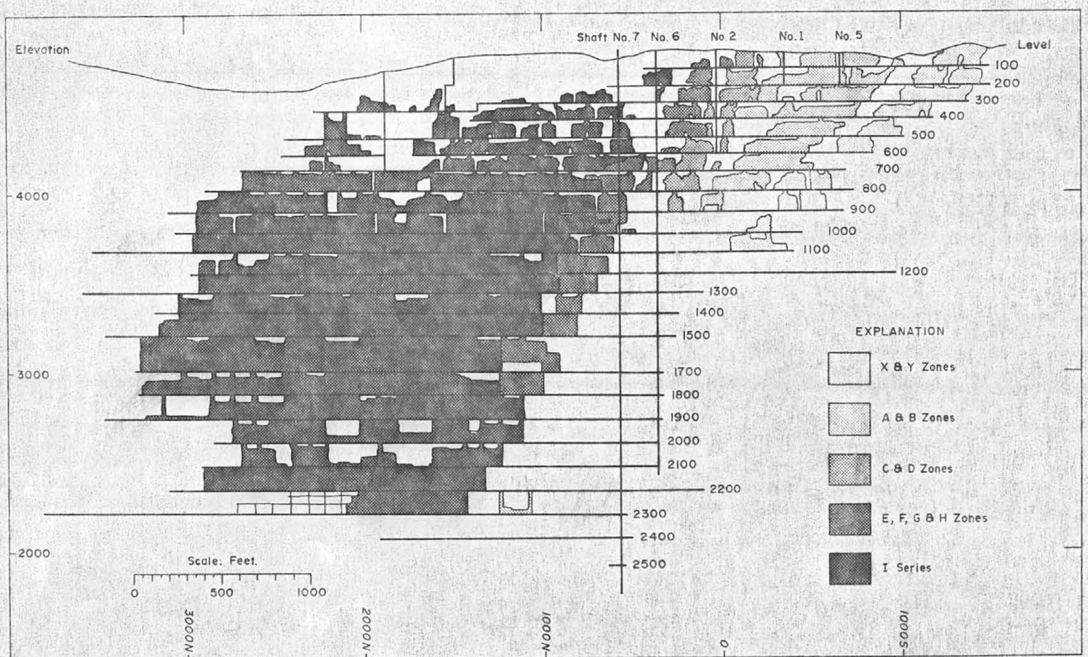


FIG. 2. *Vertical Longitudinal Section of the Iron King Mine.*

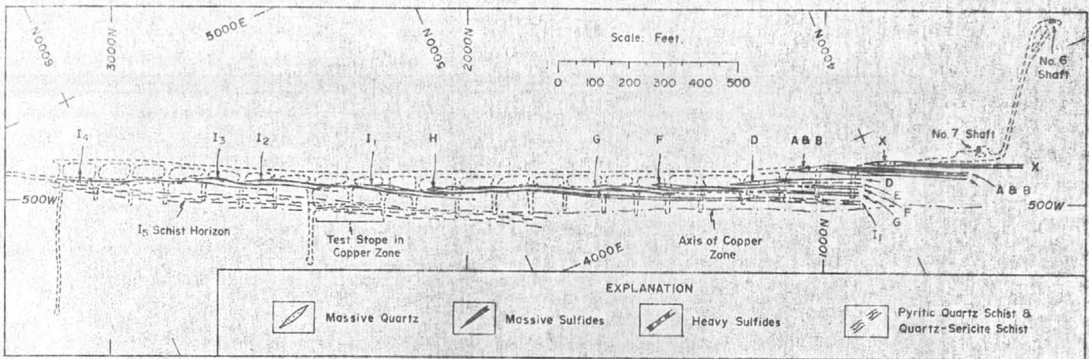


FIG. 3. *Generalized Geological Plan of 1900 Level.*

of the meta-rhyolite of the Lower Spud Mountain Tuff and the metaandesite which forms the base of the Upper Spud Mountain Tuff.

Although simple in essence, the ore horizon is fairly complex in detail. The rock types which make up what is here termed the ore horizon include massive quartz, massive sulfides, and pyritic quartz schist. The last-named appears to grade toward the south into bands of quartz-sericite schist. These rock types occur in compound lenses which have an *en echelon* arrangement. The "style" of their occurrence may be seen in Figures 2, 3, 4, and 5.

As mentioned above, the horizons which contain the massive sulfides are recognizable to the south of the mine as bands of grey-green quartz-sericite schist which are interbedded with metaandesite. As the ore horizons are traced to the north, in drill holes and crosscuts, progressive facies changes occur—some gradually and others abruptly. Initially, the proportion of quartz and pyrite increases as the sericite decreases. The term "massive sulfide" is used where the pyrite and other sulfides make up between an estimated 50 to 80 per cent of the rock, the balance consisting of quartz, carbonates, sericite, and minor chlorite. The massive sulfides are fine-grained and commonly

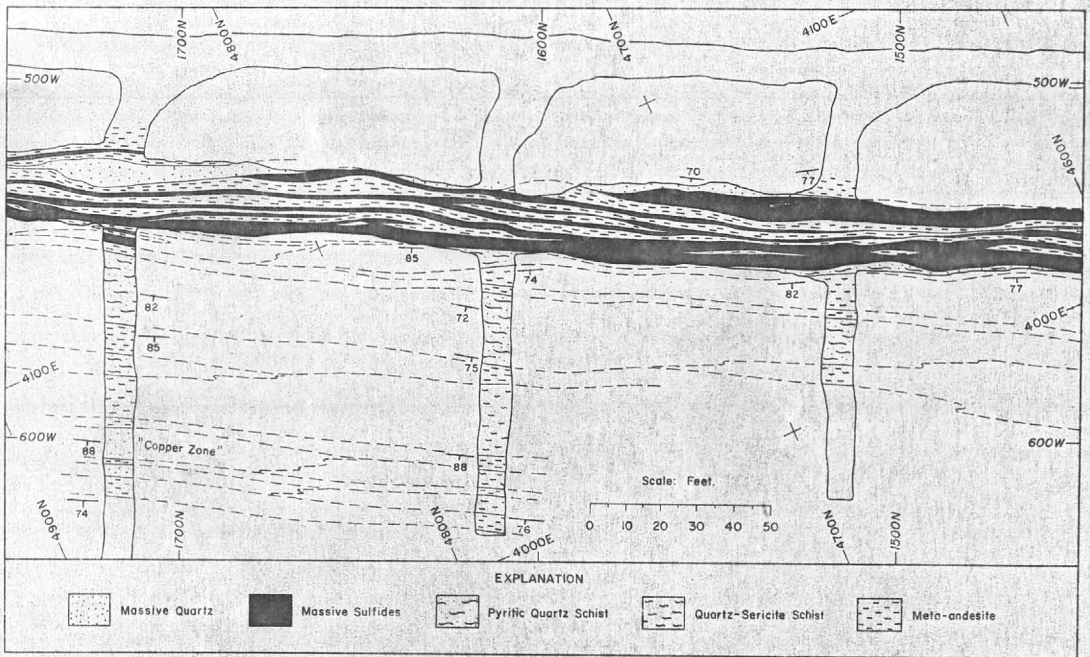


FIG. 4. *Plan of a Portion of the I Series on the 2200 Level.*

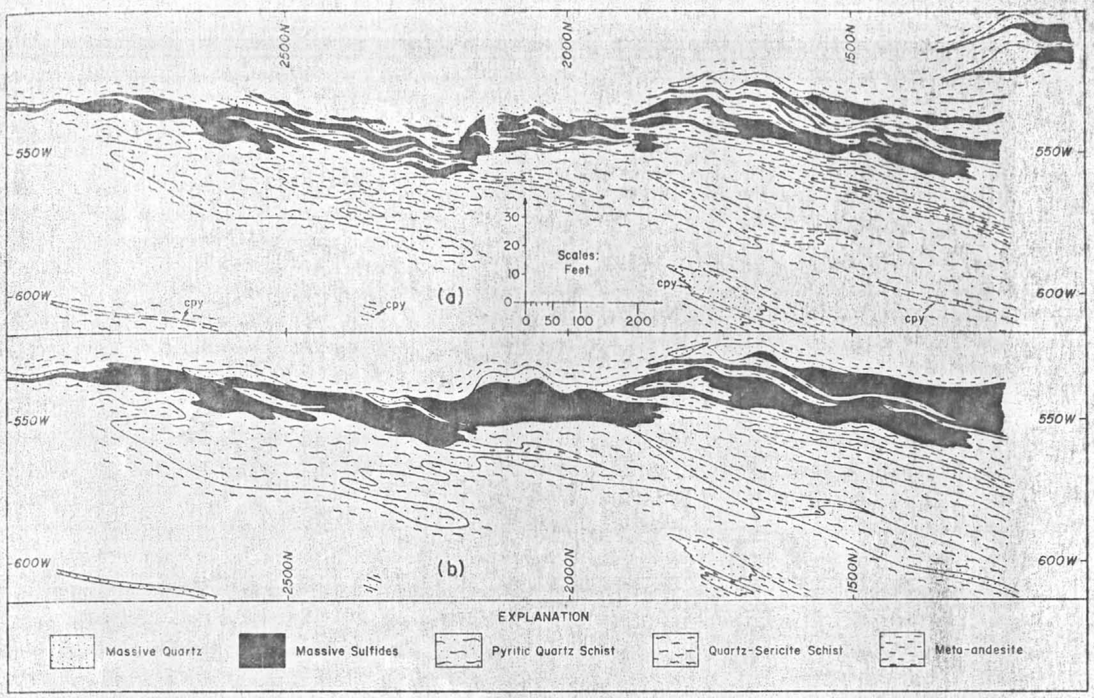


FIG. 5a. *Plan of the 2200 Level Foreshortened along Strike by a Factor of Five.*
FIG. 5b. *Simplified and Idealized Version of (a).*

banded. Pyrite is the dominant metallic mineral, accompanied by lesser amounts of sphalerite, galena, chalcopyrite, and arsenopyrite. Compared to other massive sulfide deposits, pyrrhotite is conspicuous by its absence at the Iron King. If followed northward, the massive sulfides either gradually narrow down to a thin selvage or give way abruptly, and with fairly sharp contacts, to bodies of greenish grey, massive quartz which have a waxy or chalcedonic luster—the so-called "north-end quartz noses." Locally, such quartz is banded and contains irregularly-distributed, coarsely-crystalline sphalerite and galena as well as erratic values in gold and silver. In the development of a "quartz nose," as the massive sulfides diminish in width, the quartz is found to increase in width on the eastern or footwall side. In some localities, the massive quartz gradually narrows and becomes difficult to trace farther north with reliability. In other places, the quartz swells rapidly to form a bulbous lens, wider than the original zone, and then pinches abruptly (Figure 3).

i.e. stratigraphically above

Since a number of individual ore horizons exhibit these facies changes and since the identical facies in each horizon occur with an en echelon pattern, the progressive changes described above are visible across, as well as along, strike. These relationships are illustrated in Figures 3, 4, and 5a and 5b.

Before leaving the subject of the ore horizon and its facies, it should be noted that quartz and/or iron oxides are common in the horizons containing massive sulfide deposits in other districts. Kinkel and Gilmour record siliceous chert or tuff at the top of the ore-bearing horizons in the Shasta and Noranda district respectively (20,46). Sulfides are ubiquitous in the laminated chert in the Noranda district and, locally, this rock strongly resembles the northern extremity of the main sulfide lens at the Iron King. In certain other districts containing massive sulfides, the ore horizon is characterized by the presence of iron, rather than quartz, in the form of hematite in shales and in the upper portions of the rhyolitic tuffs (38,26). Typically, however, both iron and quartz are present in the ore horizon in the form of jasper (26) or banded siliceous iron formation (14,28,24,32,41).

The origin of this quartz which is closely associated with the massive sulfides raises an interesting question. Anderson and Creasey appear to have considered it to be epigenetic. There is not much doubt that the quartz in the other areas mentioned where massive sulfides occur is sedimentary in origin—syngenetic or diagenetic. Similarly at the Iron King, the association with volcanic rocks, the presence of thin interbeds of chert, the lenses of sedimentary jasper (see Upper Spud Mountain Tuff below), the local banding, and the concordant form of the massive quartz all combine to suggest that the quartz may have been syngenetic—possibly a sinter deposited by hot springs. In this connection, it is interesting to note that Kato (3) records the presence of abundant silica associated with a sulfur-pyrite deposit formed by Tertiary volcanism in Japan.

Before terminating the discussion on the Lower Spud Mountain Tuff and the ore horizon, it would be appropriate to mention a second zone of mineralization which, to date, has been of negligible economic significance but is of considerable geologic interest. This second zone of mineralization consists of a persistant band of copper mineralization which occurs ? in the structural hanging wall of the main zinc ore horizon. This "Copper Zone" will be discussed in more detail later in the text.

Upper Spud Mountain Tuff

The horizon containing the massive sulfide lenses is succeeded to the east by a series of laminated to massive metaandesite and interbedded sediments which reach a thickness of about 800 feet. The metaandesite appears to represent pyroclastic deposits including normal tuff, feldspar crystal tuff, and agglomerate. Thin beds, up to a few feet in thickness, of rhyolitic quartz-feldspar crystal tuff, which resemble those in the Lower Spud Mountain Tuff, occur within the sequence.

i.e. underlie ore

A thin bed of conglomerate containing chert and jasper pebbles occurs near the stratigraphic top of this succession. The presence of these pebbles suggests that some of the chert and jasper lower in the succession are sedimentary in origin and essentially syngenetic. Reference will be made to this point again, since it has a bearing on the postulated mode of origin of the sulfides and massive quartz. On the basis of regional mapping, Anderson and Blacet are now of the opinion that these silicic tuffs and argillaceous sediments represent a faulted segment of the Texas Gulch formation of the Alder Group (34). Evidence of the presence of major faults is lacking in the washes which drain the eastern portion of the Iron King property and consequently cross the rocks in question. We, therefore, consider Anderson and Blacet's case "not proven."

Similar to cgl. on top of ore in west shorter →

Iron King Andesite

The Spud Mountain Tuff is succeeded to the east by the Iron King andesite. The rocks which make up the western margin of this formation are scarcely distinguishable from the metaandesite and "gritty" sericite schist of the Upper Spud Mountain Tuff, but farther east the Iron King andesite is composed of quite undeformed, more massive flows in which pillow structures are locally preserved.* The thickness of this formation is unknown since the upper portion has been removed by erosion.

Intrusions

Rhyolitic sills and dikes of Tertiary (?) age and basaltic dikes of Quaternary age are the only definite intrusions found in the mine area. It has already been observed that Bulmer and his colleagues (17) interpreted some of the schistose porphyritic rhyolite that crops out on surface as intrusions, whereas the evidence from underground drilling seems to indicate that these rocks are sheared crystal tuffs contemporaneous and interbedded with the adjacent rocks.

Tertiary Gravel

The northern portion of the mineralized zone at the Iron King mine, as well as the enclosing host rocks, are covered by the gravel deposits of Tertiary age which form the present surface of Lonesome Valley.

STRATIGRAPHY AND STRUCTURE

Stratigraphic Sequence

The rocks in the vicinity of the Iron King mine have an average strike of N26°E and dip at 78°WNW.

Employing independent lines of reasoning, Anderson and Creasey (26) on the one hand and Bulmer and his colleagues (17) on the other, concluded that the rocks in the mine area make up part of the western limb of a large syncline. The trend of this syncline was thought to be approximately parallel to the strike of the rocks; the plunge is low, and the axial plane and both limbs dip steeply toward the west.

* Editor's Note: Anderson considers these rocks to be highly deformed, and it is his observation that pillows can be recognized in them only locally.

If this interpretation is correct, the sequence given above in the discussion of lithology refers to the oldest rocks first.

Other consequences follow from this stratigraphic interpretation. Firstly, the siliceous rocks with which the sulfides at the Iron King are associated must represent the warning stage of a volcanic cycle which commenced with the eruption of andesites and terminated with explosive rhyolitic volcanism. It follows that the rhyolitic tuff is overlain by a formation consisting of sedimentary deposits and volcanic rocks heralding a new phase of andesitic volcanism. Secondly, the massive sulfides occur at the top of the rhyolitic formation in question. Thirdly, the zinc-rich massive sulfide lenses stratigraphically overlie the copper zone in the present hanging wall.

This last consequence, the stratigraphic position of the main sulfide horizon and copper zone, is worthy of elaboration. Examples of this type of zoning are difficult to obtain, partly because the stratigraphy in the vicinity of many massive sulfide deposits is obscure and partly because most authors neglect to describe the distribution of sulfides in stratigraphic terms. However, in the Jerome and Noranda districts, where the stratigraphy is well known, the bulk of the zinc stratigraphically and structurally overlies the copper mineralization (26,46). On the other hand, where the rocks are overturned—such as at Rammelsberg, which occurs in the overturned lower limb of a large recumbent fold—copper predominates in the present hanging wall and lead and zinc in the footwall in a sequence similar to that at the Iron King (19). Taken as a whole, these relationships suggest that the deposits in question were probably formed before the enclosing rocks were folded.

Structural Features of the Ore Zone

There undoubtedly is a wide zone of shearing associated with rock types within which the ore deposits of the Iron King mine occur, as most previous workers have stressed. However, whether the shearing brought about important changes in the original rocks or was localized by certain horizons which readily responded to deformation is debatable. The shearing which occurs in proximity to the ore bodies might be a reflection of the presence of the thin-bedded tuffaceous and argillaceous sediments with which the deposits are associated, and this possibility was noted by Creasey (26, p. 157).

Within this zone of shearing the rocks have a deformational fabric. A fine, penetrative lineation is found in all of the rocks of the ore zone including the more massive sulfides. Small chevron folds and minor buckles are common in the schistose rocks. The sheared conglomerate, which occurs in the Upper Spud Mountain Tuff, contains pebbles of jasper and other resistant rocks which are flattened and elongated (the proportions of the length to breadth to thickness are in the order of 4:2:1). The longest dimension of the deformed pebbles also defines a lineation. Figure 6 shows the attitude of the principal elements of the rock fabric. It may be seen that the fine, penetrative lineation plunges to the north, closely parallel to the plunge of the sulfide lenses (Figures 1, 2, 6). The chevron folds plunge to the southwest in the plane of the foliation. The lineation defined by the pebbles tends to lie between the other two. It seems possible that the chevron folds which are confined to the ductile schists might represent the overprint of a deformation more recent than that which transformed the original rocks and generated the fine penetrative lineation.

Reverse or thrust faults were encountered in the workings in the upper levels of the mine. These faults have approximately the same

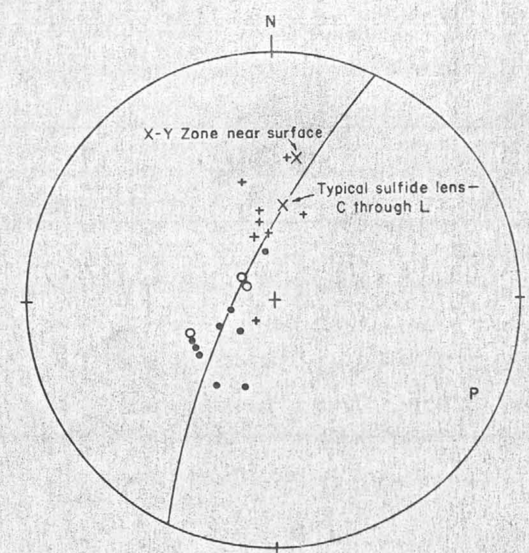


FIG. 6. *Lower Hemisphere Equal-Area Projection, showing the principal elements in the rock fabric: full arc, mean foliation; P, pole of mean foliation; x, longest dimension of ore-shoot; cross, fine penetrative lineation; circle, longest dimension of deformed pebble; dot, axis of chevron fold.*

Note: Jim Knox states that mining below 3,000 ft level was not feasible because of bad ground i.e. crushed zone

strike as the ore zone, although the dips are in some places steeper and in others flatter.

In the lower levels of the mine, a series of crush zones having little or no displacement has locally weakened the rocks. These crush zones have an easterly strike and dip to the north and south at angles between 40 and 80 degrees.

These transverse crushed zones, or cross faults, are of two types and quite possibly of two ages, the distinction between the two being the presence of orange-colored ankerite not only within the faults but also an ankeritization in the wall rocks adjacent to some of these cross structures, and the complete lack of any such alteration either within or adjacent to others. As will be discussed in more detail later in the text, there is some evidence that a spatial relationship exists between the copper zone oreshoots and the ankeritized cross faults.

The Form of the Sulfide Lenses

The separate lenses of massive sulfides have been identified by letters. The lens lying farthest east and south was designated X; the next zone to the north and west was known as Y. Proceeding farther north and west, these were followed by A, B, C, etc., through L (Figures 1 and 2). Most previous workers recognized that below the first few levels in the mine the distinction between I, J, K and L (or I_1, I_2, etc.) diminished and these four lenses came to be considered as a single unit known as the I series. By contrast, X, Y, A, B, and C of the massive sulfide zones were collectively identified as the Footwall series.

The X-Y and A-B zones have been distinct, and recognizable, throughout the entire developed depth of the mine. They were, however, sub-ore in grade between about the 1000 level and the 2000 level. Below the 2000 level these two zones have improved in tenor to the extent that they became ore, and they are currently being mined below that level.

Reference has already been made to the lineations which may be found in the ore zone. The longest axes of the majority of ore shoots or lenses plunge at approximately 60 degrees to the north parallel to the fine, penetrative lineation present in the massive rocks. In the upper levels of the mine, the X-Y zone has a plunge of 45 degrees parallel to the observable lineations at the surface near the outcrop of these lenses (Figure 1).

It is clear that if either the massive quartz or the massive sulfides are joined from one lens to another, the closely-spaced *en echelon* arrangement of the ore bodies would give rise to an apparent strike which differs from the true strike (Figures 3, 5). This reason for the discordance between the strike of the ore

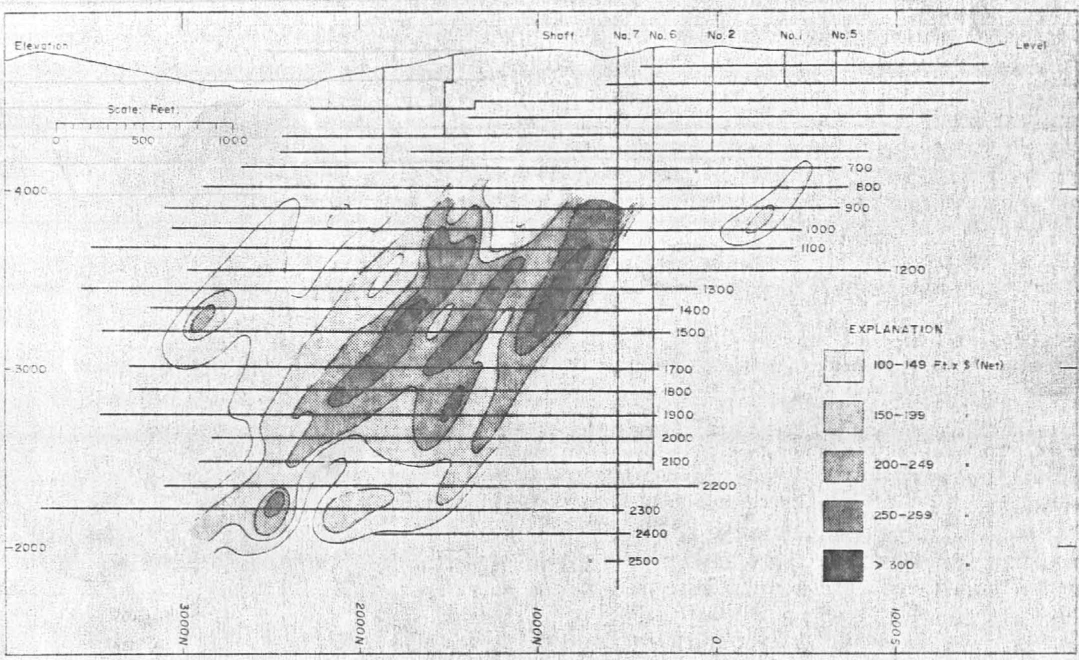


FIG. 7. *Vertical Longitudinal Section of the Iron King mine, showing contours of the product of width and dollar value.*

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. By contrast, X, Y, A,
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the majority of ore shoots
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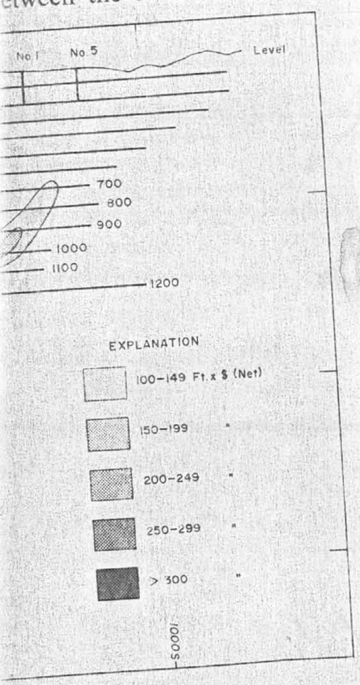

owing contours of the product of

Batten surface would make more ore?

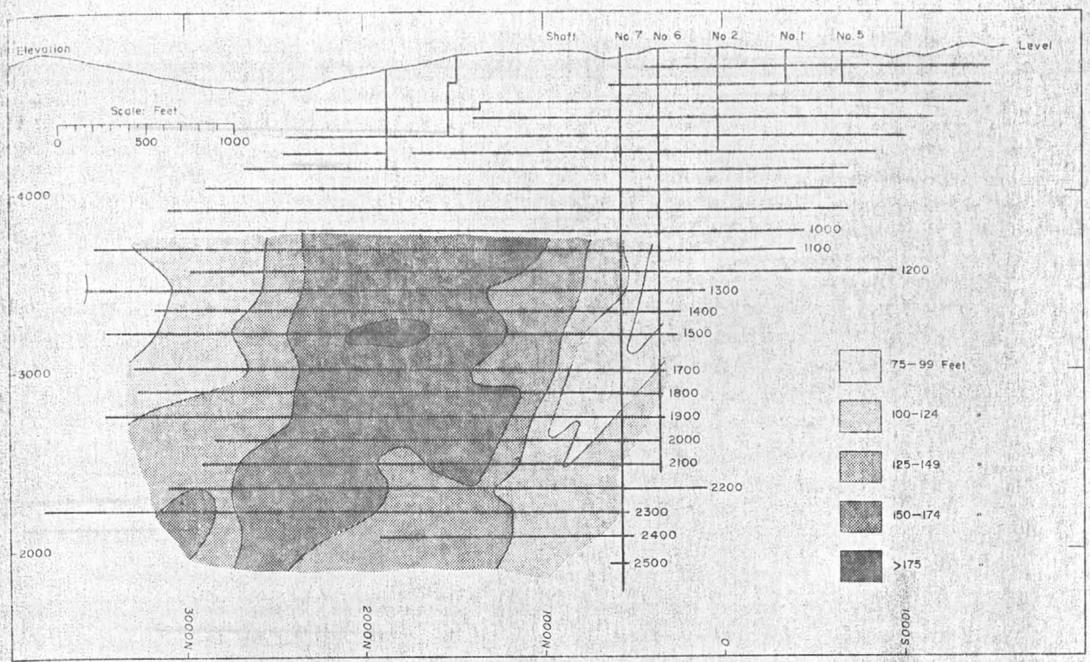

FIG. 8. *Vertical Longitudinal Section, showing contours of the distance between the footwall of the I Series and an arbitrary inclined plane.*

lenses and the ore zone was recognized by Reber (6), although it appears to have been attributed to deformation by Creasey (26).

It follows that the sulfides—or any other facies in the individual ore horizon—do not lie in a single, narrow zone but, rather, occur along a discordant or cross-cutting contact or facies change between two major lithological units.

A longitudinal section of the main I series was prepared showing contours of the product of width in feet and dollar value (using uniform dollar factors throughout the mine), and it was found that four ore shoots show up within the I series (Figure 7). Presumably, these four shoots correspond to the I, J, K and L (I_1, I_2, etc.) ore lenses recognized in the upper levels. The contours attain their highest values in the middle two shoots (J and K), and, in all four, the maxima occur between the 1100 and 1700 levels. A "Conolly Diagram," or longitudinal section showing by means of contours the distance between the footwall of the ore body and a reference plane drawn parallel to the body (4), was prepared for the I series (Figure 8). It is obvious that, if the rocks in the mine area were rotated into their original attitude, the upper surface of the I series (the present footwall) would form an elongated basin. In other words, the maximum concentration of sulfides roughly coincides with the "trough" of the "unrolled" I series. Whether this elongated basin is an original structure or a result of deformation is uncertain.

COPPER ZONE

Historical Background

The existence of copper mineralization in the structural hanging wall of the main Iron King ore zone has been known for many years. The strongest known surface expression of this mineralization outcrops (200 feet west of "X" vein) as two disconnected lenses of copper-stained quartz and was described by Anderson and Creasey (26) as "the widest and most continuous of the nonproductive veins." A shallow shaft (the "Copper Shaft," 130 feet deep) was sunk on the southernmost segment of the Copper Vein many years ago (Figure 1), and in 1947 Shattuck Denn leased these workings to a local miner who produced 132.4 tons of 6.13 per cent (oxide) copper ore before abandoning the lease in 1948. This is the only known production from the near-surface workings.

In the main underground mine, the existence of a "copper stringer" in the hanging wall was recognized by the operating staff as early as the 1940's, but the "stringer" appeared to be discontinuous and received little attention. This lack of interest probably stemmed from the fact that the mining system above the 1000 level simply employed an ore drift—without a footwall haulage and regular hanging wall crosscuts—and very few workings probed the hanging wall sufficiently far to cross the locus of the copper mineralization. Below the 1000 level, where the currently-used system for haulage was started, the "copper stringer" was cut more often, but even so, many hanging-wall crosscuts did not reach it, and, where they did, the mineralization did not appear to be sufficiently wide to warrant further interest.

The first serious attention to be given to the copper zone came in the fall of 1960 when the 1100 N crosscut on the 2100 level penetrated a band of chalcopyrite-chlorite mineralization which assayed 3.44 per cent copper, with minor gold, silver, lead, and zinc over a width of 5.1 feet. This intersection prompted a decision: (1) to drive all new crosscuts sufficiently far to reach this zone; (2) to drive two exploratory crosscuts further south on the 2100 level; and (3) to re-evaluate all available data (old drill logs and accessible crosscuts) higher in the mine. Also, a program of short diamond drill holes to probe the hanging wall was started and is still, intermittently, in progress.

As a result of this work, by late 1961, six potential copper zone "oreshoots" were indicated, as are shown on Figure 9. These shoots had relatively short strike lengths (maximum of 250 feet), but, in the plunge direction, one of the six could be traced in crosscuts for 1100 feet and another for 800 feet. The northernmost oreshoot, which is the largest, probably relates to a number of favorable earlier diamond drill hole intersections near the surface and to a more recent diamond drill hole below the 2300 level. This suggests that it has a plunge length of at least 2500 feet.

Using a minimum stoping width of 3.0 feet, ore widths ranged up to 10.1 feet and averaged 5.3 feet. Between the 1500 and 2100 levels reserves were estimated at 150,000 tons containing copper, gold, silver, lead, and zinc ore that appeared to be mineable.

In late 1961, a stope 200 feet long was started on the 2000 level between the 2200 N and 2400 N crosscuts. Based upon the development work sampling (50 cut samples), this ore block was estimated to contain 12,578 tons that would average 0.029 ounces of gold, 0.67 ounces of silver per ton, 0.05 per cent lead, 4.32 per cent zinc and 2.36 per cent copper over an average width of 4.65 feet. Unfortunately, excess dilution from the walls and the

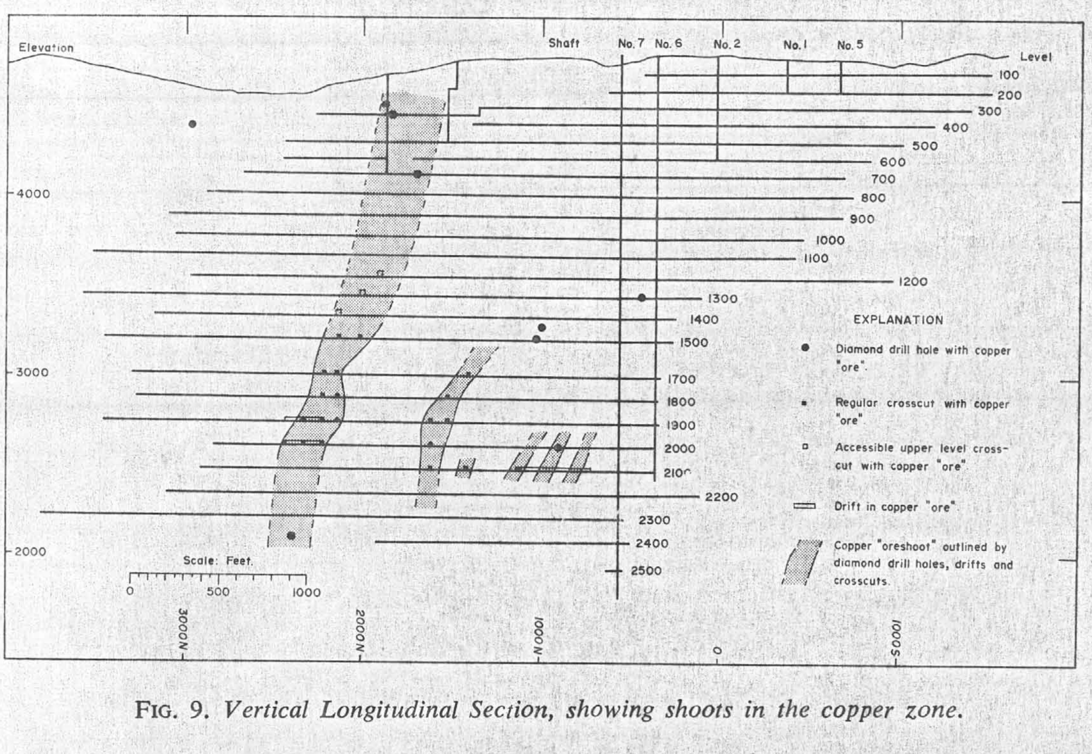

FIG. 9. *Vertical Longitudinal Section, showing shoots in the copper zone.*

absence of a copper circuit in the mill caused the stope to be uneconomic, and it was shut down after approximately 50 per cent of the estimated tonnage had been extracted. At the time of writing no further mining has been attempted within the copper zone.*

Mineralogy and Structure

The mineralization within the copper zone consists of two types; massive sulfides similar to the main zinc-lead ore zone but having an appreciably higher copper content, and chalcopyrite-tennantite mineralization associated with quartz stringers and/or coarse, black chlorite.

Material of the massive sulfide type has been found in only one area (between the 700 and 900 crosscuts on the 2100 level) where a massive sulfide lens 85 feet long and up to 6 feet wide was exposed by drifting. In all other crosscuts and diamond drill hole intersections, the copper-zone mineralization consists of coarse-grained mixed chalcopyrite-tennantite either in bands of quartz or disseminated throughout coarse dark chlorite. Accompanying the copper minerals are irregular amounts of sphalerite but typically only very minor galena. In most places, precious metal amounts are below those found in the main lead-zinc ore zone.

The copper zone might be described as a 400-foot wide belt of mineralization which is very nearly parallel to the main ore zone, though they seem to converge slightly to the north and in depth. Within this broad zone of scattered sulfide mineralization, the distinct shoot are confined to bands of quartz-sericite schist. There is also a suggestion that these oreshoots may have been localized by the ankeritized cross faults since four out of the six known copper shoots occur near the intersection of such a fault and the copper zone.

The evidence regarding the overall form of the copper zone is conflicting and slight differences of opinion separate the writers on this subject. One author (A.R.S.) believes that the copper zone cuts across the lithological horizons at a slight angle and persists in the metaandesite between the shoots in quartz-sericite schist. This view is supported both by the presence of copper mineralization in the metaandesite and chlorite schist and by the fact that, within the quartz-sericite schist bands, the individual higher-grade copper shoots cross from east to west as they are traced from north to south. The other author believes that the structure of the copper zone is comparable to that of the principal ore zone and consists of a series of tabular lenses having an *en echelon* arrangement. According to this view, the apparent planar form is simply a reflection of the close-set spacing of the *en echelon* lenses, and the mineralization in the intervening chlorite schist is part of the overall mineralization of the broad copper zone.

CHANGES IN THE DEPOSIT WITH DEPTH

There have been some very marked changes in the Iron King deposit with depth. Four such changes that, in the opinion of the writers, are the most significant will be discussed in this paper. Two of these changes are structural and two are mineralogical.

Structural Changes

From an economic point of view, the most important of the changes with depth has been a breaking up of the massive sulfides, by the introduction of multiple metaandesite bands so that, at depth, the ore zone consists of alternating thin bands of sulfides and metaandesite in roughly equal proportions, or with the metaandesite predominating (Figure 4). This change started at about the 1500 level. Above that horizon the massive sulfide lenses were essentially devoid of either waste horses or of metaandesite bands, save for the continuous schistose partings that separated the individual veins. This is illustrated quite well by the geologic level plans (for the 700 through 1100 levels) which accompany the Professional Paper by Anderson and Creasey (26). Below the 1500 level, metaandesite bands and lenticular horses within the massive sulfide veins gradually started to appear. At the depth of the 1900 level, these partings constituted roughly 50 per cent of the volume of the previously continuous sulfide lenses. The levels below the 1900, as well as deep drilling below the 2400 level, have proved that this condition of increasing metaandesite within the ore zone, at the expense of the sulfides, continues progressively with depth.

The second structural change, which was first noted in stopes between the 1800 and

* Since writing the above, a stope was begun on the copper zone between 800 and 1000N on the 2100 level. Between March and June, 1967, approximately 2000 tons had been produced, of which the first 1000 tons contained 0.04 ounces of gold and 1.5 ounces of silver per ton, 2.1 per cent lead, 4.4 per cent zinc, and 2.46 per cent copper.

1900 levels, is the existence of "rolls" (broad, gentle folds) in the immediate contact at the structural footwall of the ore zone. These rolls generally have a flat (5° to 20°) plunge to the north, and they are in the order of 10 to 20 feet in width and 2 to 4 feet in height. These gentle folds persist only a few feet into the footwall metaandesite. The rolls are not persistent in a plunge direction for more than a matter of tens of feet, but as one dies out another begins to form. Deformation of this type within the footwall adjacent to the ore zone was unknown above the area of the 1800 level but persists from the 1800 level downward.

Mineralogical Changes

Probably the most significant mineralogical change with depth is the occurrence of a milky white, brecciated quartz-carbonate-feldspar which contains coarse-grained sphalerite and galena. This material occurs as irregular pods within the massive sulfide veins, with the borders of the pods being corroded and embayed by the finer-grained massive sulfide. Because of the relationships, this characteristic milky white material has been termed "early brecciated quartz" by the geological department at the mine. Early brecciated quartz was unknown in the deposit above the 1900 level, and, at that level, it started to appear over small areas as roughly equidimensional masses from 1 to 4 inches in diameter. On the 1900 level, these areas of early brecciated quartz were usually localized within the fine-grained sulfides adjacent to some of the east-west striking and steeply dipping cross faults. With increased depth below the 1900 level, both the number and the size of these masses of early brecciated quartz rapidly increases, so that at the depth of the 2200 level, masses 3 to 5 feet in diameter were not uncommon, and such masses existed at random throughout the entire length of the I series veins. In addition to the coarse-grained sulfides that occur within the early brecciated quartz, some of these masses were separately sampled and were found to have very high precious metal contents (i.e., more than 1 ounce of gold and 30 to 50 ounces of silver per ton).

MINING AND MILLING

The engineering problems at the Iron King are, of course, mainly a consequence of the form of the ore deposit and involve mining tabular bodies that have strike lengths ranging from 100 to more than 2000 feet, widths ranging up to 20 feet, and dips approximating 80 degrees. These tabular ore bodies consist of alternating bands of massive sulfides and metaandesite, with local large bodies of massive quartz. Although the contacts between the massive sulfides, massive quartz, and/or the metaandesite are very sharp, the margins of the ore bodies, particularly on the hanging wall side, approximate an "assay wall." The advantages and disadvantages of taking additional sulfide bands with concomitant interbeds of metaandesite and simultaneously increasing recovery and mining width while lowering grade have to be considered when determining the ore cut-off. The footwall contact is fairly sharp in most places. One of the common problems on the footwall side of the I series involves the selection of the mining footwall when approaching the ends of the *en echelon* structures.

As in most mines, the strength of the wall rocks varies from place to place. In general, however, the metaandesite on the footwall makes a moderately good wall, whereas the schist on the hanging wall does not stand as well. Locally, the cross faults or crush zones, mentioned above, have weakened both the ore and the wall rocks. In many places, the massive sulfides are broken by three mutually perpendicular joint systems, one of which is nearly horizontal, one of which is parallel to the strike of the ore, and one of which is at right angles to the strike of the ore.

Above the 2200 level, the ore was mined by square-set, cut-and-fill and shrinkage stoping (Figures 2 and 3). Commencing on the 2200 level, a modified form of sub-level stoping was adopted in which blast holes were drilled down the dip of the ore body rather than fanned out from the sub-levels. The spacing of the sub-levels and interconnecting raises, as well as the manner in which the sub-levels are retreated, is indicated on the 2300 level in the longitudinal section (Figure 2). Shrinkage and cut-and-fill stoping are employed in the shoots with short strike length, such as the A-B zone on the 2400 through 2000 levels. These methods are also utilized on the ends of the I series on the upper levels where a few blocks of ore grade were outlined early in 1965.

Massive pyritic sulfide deposits commonly are difficult to concentrate on account of the fine-grain size and the degree of intergrowth of the constituent sulfide minerals. However, this problem is not quite so acute at the Iron King as it is elsewhere and grinding to minus

275 mesh provides 80 to 85 per cent recoveries of lead and zinc with acceptable concentrate grades.

Perhaps the variable grain size exhibited by the sulfides in this class of deposits reflects the degree of metamorphism to which the deposits have been subjected.

CLASSIFICATION AND ORIGIN

Although the ore bodies at the Iron King belong to a class of deposits which is rather uncommon in the United States, and may therefore be unfamiliar to some readers, similar deposits are extremely important in other areas, notably, Canada. Australia, and western Europe. It has already been noted that Anderson and Creasey drew attention to the similarity that exists between the mineralogy of the sulfides in the Jerome-Humboldt area and those in certain other parts of the world and that the writers have suggested that these similarities extend to the lithology and orogenic setting of the host rocks as well. Using these types of criteria, the deposit at the Iron King can be recognized as a member of a large group that includes the massive sulfide ore bodies in the Noranda (46) and Matagami (49) districts in Quebec, and the Manitouwadge district in Ontario (24), the Sullivan deposit (8), the massive sulfide deposits in the Skellefte district in northern Sweden (27) and the ore bodies of Broken Hill (14), Mt. Isa (13,15), and Mt. Morgan (16) in Australia. All of these are found in Precambrian rocks. Examples in rocks of Paleozoic age include the deposits in the Shasta district (32,20), the Ducktown and similar deposits in the southern Appalachians (39), the massive sulfides in the Bathurst-Newcastle district in northern New Brunswick (28), and the Rio Tinto (38), Rammelsberg (19) and Meggen (18) deposits in western Europe. The massive sulfides on Balabac Island in the Philippines associated with Mesozoic rocks (40) and certain deposits associated with Tertiary rocks on Cyprus (47) and in Turkey (50) may represent more recent examples.

All of these concordant, massive, pyritic sulfide deposits can be recognized as belonging to a well-defined class in an orogenic classification (35,37). Furthermore, they may be considered as a linear series in which one end-member consists of deposits containing gold-copper mineralization associated with eugeosynclinal volcanic rocks and the other end-member consists of silver-lead-zinc deposits found in eugeosynclinal sediments. It follows that any given deposit in this series will have clear-cut affinities with another close to it and less obvious affiliations with examples farther removed.

In the same way that the deposits themselves may be rather unfamiliar to some readers, the genetic theory presented here may seem new, although this is by no means true. It is some years since Ehrenburg and Kraume proposed this theory in reference to the origin of the famous deposits of Rammelsberg and Meggen (18,19). King (14), Amstutz (21,25), Miller (30), Stanton (29), Martin (23), Thomson (31), Goodwin (33,43), Williams (38), Kinkel (36), Pereira (41,42), and Gilmour (46), among others, have postulated a volcanic-sedimentary mode of origin for concordant, massive, pyritic sulfide deposits.*

One of the principal difficulties confronting attempts to determine the origin of ore deposits entails the problem of distinguishing between those features which are genetically significant from those which are merely incidental. In the writers' opinion, the best way of doing this is to compare and contrast as many deposits of a particular type as possible and to single out those features which are common to all. It may then be assumed that these are the characteristics which must be accommodated into a general theory of origin for the group as a whole. This is one of the reasons why a good deal of space has been devoted to the classification and affinities of the Iron King deposit.

Those characteristics of the Iron King deposit that seem to the writers to throw light on the origin of the deposit will be reviewed and the possible genetic significance brought out. A reading of any or all of the references cited will reveal how common these features are among massive sulfide deposits in general.

(1) The Yavapai schists within which the ore bodies at the Iron King mine occur represent a series of metamorphosed volcanic and sedimentary eugeosynclinal rocks. This association may imply a genetic relationship.

(2) Within this series of metamorphosed rocks, the Iron King ore bodies lie at the (stratigraphic) top of a unit consisting pre-

* The interested reader is also referred to a Symposium on Strata-Bound Sulphide Deposits and Their Formative Environments held by the Canadian Institute of Mining and Metallurgy (Canadian Inst. Min. and Met. 1965, v. 143, p. 253–300) and to contributed remarks by G. G. Suffel (Canadian Inst. Min. Met. 1965, v. 143, p. 301–307).

dominantly of metamorphosed rhyolitic tuffs that were evidently deposited during a period of explosive volcanism.

(3) Quartz is abundant in the ore horizon either in the form of large, lenticular bodies or in thin, laminated, chert-like beds. Both contain pyrite and other sulfides. Jasperoidal lenses are also fairly common in the tuffs that stratigraphically overlie the ore zone, and the presence of jasper pebbles in a thin conglomerate indicates that some, at least, of the jasper is syngenetic. This, in turn, suggests that the quartz represents a siliceous sinter deposited by submarine fumaroles.

(4) The sulfides exhibit a stratigraphic zoning in which copper tends to underlie lead and zinc. The occurrence of this sequence both in deposits which are right-way-up and over-turned, as is the Iron King, would seem to indicate that the sulfides were deposited before orogenesis deformed the deposit and the enclosing host rocks.

(5) It has already been argued that the sericite schists, which stratigraphically underlie the ore bodies, most probably represent metamorphosed rhyolitic tuffs, rather than altered metaandesite. It is noteworthy, however, that if the sericite schists can be proven to be a product of hydrothermal alteration, they occur only (stratigraphically) below the massive sulfides and massive quartz. This would indicate that they were altered by hydrothermal solutions before the hanging wall metaandesites were laid down, and the most probable explanation of this would be that they were altered by fumarolic action.

(6) The contacts between the massive quartz and massive sulfides on the one hand and the metaandesite that occurs on the foot-wall and as septa within the sulfides on the other are extremely sharp, and it is known that the recoverable metal content of the metaandesite is negligible. These observations are difficult to reconcile with a theory appealing to epigenetic replacement.

(7) The massive sulfides, together with the host rocks, have a metamorphic texture and fabric. This indicates that the sulfide minerals must have been emplaced before the last plastic deformation which affected the rocks took place.

These may not be the only observations which might be quoted as having a bearing on the origin of the massive sulfides. Only field, as distinct from laboratory, evidence has been recorded, although the value of some of the evidence traditionally derived from the latter source is debatable. However that may be, the observations cited above and the interpretation placed upon them are compatible with a theory of origin that depicts the ore deposits at the Iron King primarily as products of volcanism, possibly modified by deformation and metamorphism. According to this theory, the events leading up to and accompanying the formation of the ore deposit might be reconstructed as follows:

(1) During the development of a Precambrian eugeosynclinal trough, a cycle of volcanism took place which began with the eruption of ash flows and crystal tuffs of predominately andesitic composition (Spud Mountain Breccia).

(2) With the passage of time, the volcanic deposits became finer-grained and more rhyolitic, although thin beds of andesitic tuffs continued to form (Lower Spud Mountain Tuff). If the area was not under water during the formation of the breccias, it was inundated at this stage and a shallow depression may have been formed at the site of the mine. The latter part of the phase of explosive rhyolitic volcanism was accompanied by a period of intermittent hot-spring activity when thin beds of chert and large masses of siliceous sinter were deposited. Commercially-appreciable amounts of gold and silver as well as small amounts of lead, zinc, and copper were incorporated in the sinter deposits. Some or all of the constituents of the massive sulfide lenses were also deposited at or about this time.

(3) This phase of explosive rhyolitic and fumarolic activity was succeeded by a period when andesitic and subordinate rhyolitic tuffs were erupted, possibly with slight angular unconformity on the underlying rhyolitic tuffs and sinter deposits (Upper Spud Mountain Tuff). Towards the close of this stage, argillaceous sediments were laid down. Throughout this period, sporadic hot springs gave rise to local accumulations of jasper and chert some of which were broken up after consolidation and incorporated in a thin but fairly extensive bed of conglomerate. One of the effects of the continuing fumarolic activity may have been the deposition of the remaining constituents in the massive sulfide lenses if, indeed, any were still lacking.

(4) The intermittent andesitic volcanism was followed by the submarine eruption of a thick sequence of andesitic lavas in which pillow structures are locally preserved (Iron King Andesites).

(5) Eventually, all of these rocks, including the massive sulfides, were deformed by earth movements which took place during Precam-

brian time. The host rocks were metamorphosed and acquired a deformational fabric, more conspicuous in the thinly-bedded rhyolitic and andesitic tuffs than in the more massive breccias and lava flows. The sulfides may also have been re-crystallized. It is possible that they were mobilized and redistributed so that they assumed a new form, partly controlled by the deformation.

(6) The strike faults and slips which border the footwall of the ore bodies were formed at a much later date, following the period of plastic deformation.

(7) Subsequent erosion and partial burial by gravel deposits were evidently geologically straightforward.

Recent metallic deposits associated with volcanic hot springs prove that precipitation and at least a first level of concentration from such a source can occur. Numerous examples of siliceous sinters containing precious metals have been recorded by Lindgren (2) and Goodwin (45), and the latter has cited localities in Nevada, California, and New Zealand where as much as 0.5 ounces of gold and 4.0 ounces of silver per ton have been found. Deposits of base metals associated with volcanism are also known. Examples include some of the sulfur-pyrite deposits in Japan (3), the Leviathan sulfur-copper deposit in Alpine County, California (22, H. Wright, verbal communication), the concentrations of iron, lead, zinc, and copper in the Valley of Ten Thousand Smokes (1), and a recently-reported deposit of iron, lead, zinc, and copper that is forming on the floor of the Red Sea (48). Although not directly comparable in mineralogical composition and tectonic setting to the Iron King, these occurrences of metallic deposits related to recent volcanism, lend some support to the theory presented here.

The actual role of the fumaroles in the formation of a massive sulfide deposit like the Iron King is obscure, but a number of possibilities exist. Firstly, the fumaroles may have transported the constituents of the deposit directly from depth (1). Secondly, they may have leached the constituents from underlying tuffs (46,1). Thirdly, submarine hot springs may have precipitated constituents already present in sea water (48). The mode of deposition of the metals from the fumaroles is likewise a matter of speculation and suggestions include: direct precipitation of metals from vapors, perhaps through the agency of bacterial action (31); precipitation of sulfides following the decomposition of magnetite containing lead, zinc, and copper (46,1); or modification by continuing fumarolic action of existing buried deposits of sulfur (1).

It should, perhaps, be emphasized that the hypothesis presented here is not opposed to the classical hydrothermal theory. It is rather, a special facet of the general theory, and this seems to have been implicitly acknowledged by Lindgren when he wrote (2, p. 69):

"The remarkable poverty in metals of the deposits of the springs in the Yellowstone National Park, for instance, will to many seem an argument against the hydrothermal theory of genesis of ore deposits."

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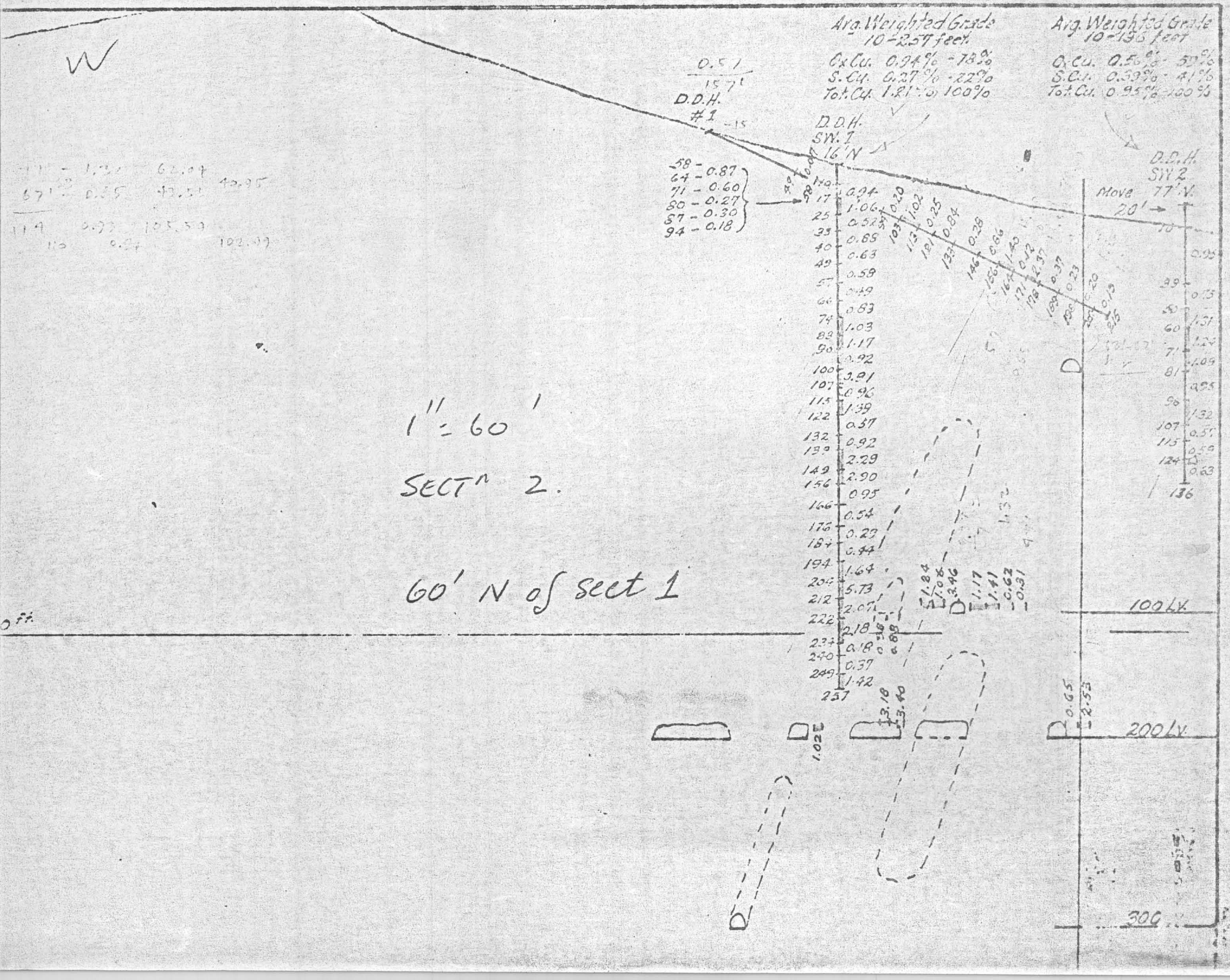
Appendix i

DDH #1, SW 1, SW 2, SW 3



W
E
1" = 60'
SECTn 2.
60' N of sect 1
Avg. Weighted Grade
10-257 feet
OxCu. 0.94% - 78%
S.Cu. 0.27% - 22%
Tot.Cu. 1.21% 100%
Avg. Weighted Grade
10-136 feet
O.Cu. 0.56% - 59%
S.Cu. 0.39% - 41%
Tot.Cu. 0.95% - 100%
0.51
157'
D.D.H.
#1
D.D.H.
SW. 1
16'N
D.D.H.
SW 2
77'N
Move
20'
58 - 0.87
64 - 0.60
71 - 0.27
80 - 0.30
87 - 0.30
94 - 0.18
10 0.94
17 1.06
25 0.52
33 0.65
40 0.63
49 0.58
57 0.49
66 0.83
74 1.03
83 1.17
90 0.92
100 3.91
107 0.96
115 1.39
122 0.57
132 0.92
139 2.29
149 2.90
156 0.95
166 0.54
176 0.29
184 0.44
194 1.64
204 5.73
212 2.07
222 2.18
234 0.18
240 0.37
249 1.42
257
103 0.20
113 1.02
121 0.25
133 0.84
146 0.28
156 0.66
164 1.40
171 0.42
178 2.37
189 0.37
198 0.23
207 0.20
0.13
215
10
0.95
39 0.73
50 1.31
60 1.24
71 1.09
81 0.95
96 1.32
107 0.57
115 0.59
124 0.63
136
1.84
1.06
3.46
1.17
1.41
0.62
0.31
0.96
0.89
3.18
3.40
1.02
0.65
2.53
100 Lv.
200 Lv.
300 Lv.

SW 3

9	0.80	7.20
10	1.83	18.30
9	4.95	44.55
9	2.02	18.18
8	2.45	19.64
7	2.53	17.71
9	0.99	8.91
9	0.32	2.88
9	2.22	19.98
9	0.80	7.20
10	0.77	7.70
8	0.65	5.20
8	2.15	17.20
10	2.78	27.80
9	0.19	1.71
9	0.79	7.11
9	0.96	8.64
9	0.31	2.79
8	0.87	6.96
4	0.75	3.00
172	(1.50)	258.26
61	(2.68)	163.58
111	(0.85)	94.68

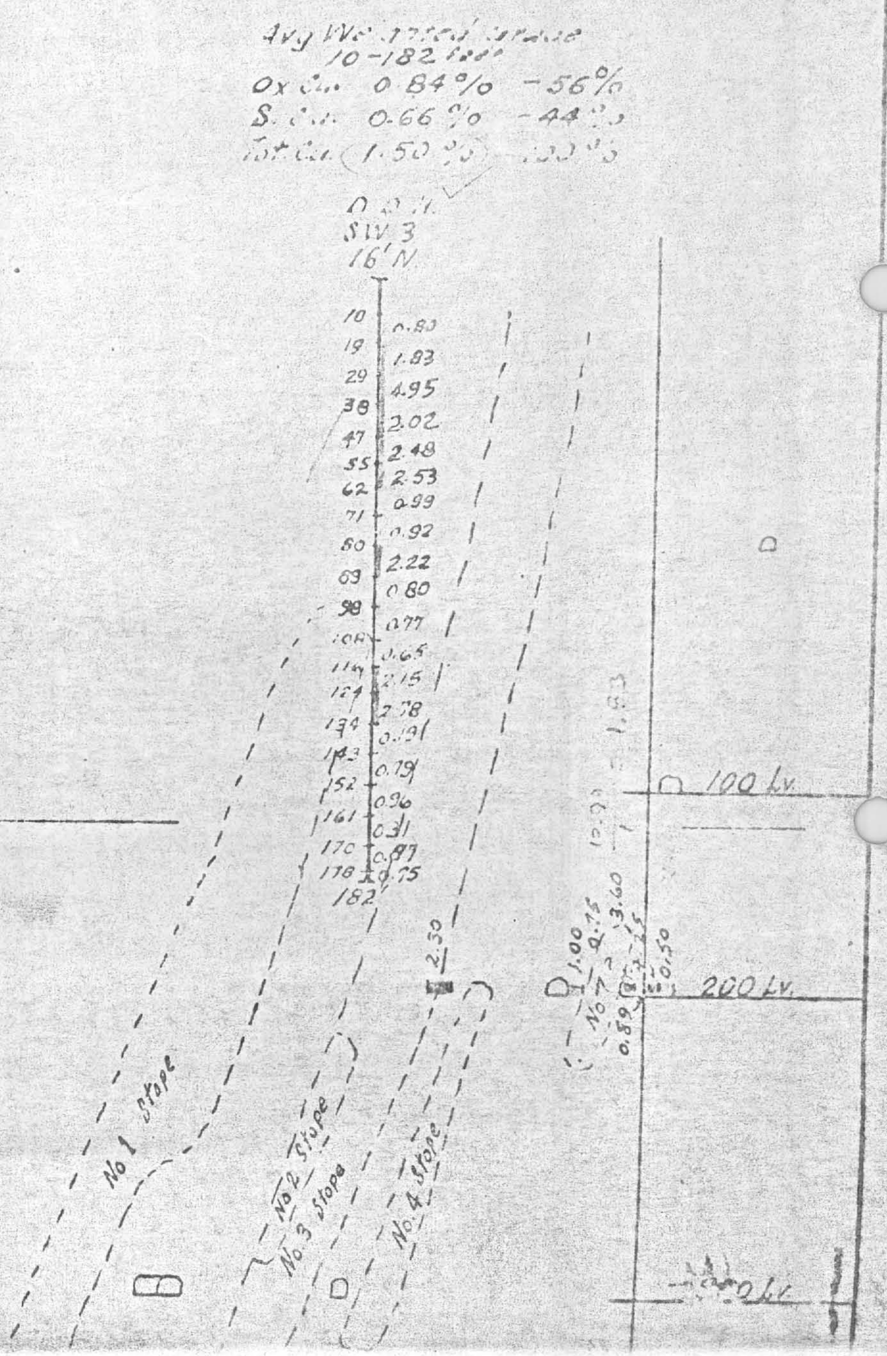
W

E

1" = 60

SECTION 1

View approx N 30°E



7	0.94	6.58
8	1.06	8.48
8	0.52	4.16
7	0.88	6.16
9	0.63	5.67
8	0.58	4.64
9	0.49	4.41
8	0.83	6.64
9	1.03	9.27
7	1.17	8.19
10	0.92	9.20
7	0.91	6.37
8	0.96	7.68
[illegible]	1.39	9.73
[illegible]	0.57	5.70
7	0.92	6.44
10	2.29	22.90
7	2.90	20.30
10	0.95	9.50
10	0.59	5.90
8	0.29	2.32
10	0.44	4.40
10	1.64	16.40
8	5.73	45.84
10	2.07	20.70
12	2.18	26.16
6	0.18	1.08
9	0.37	3.33
[illegible]	1.42	11.36
247	(1.21)	299.01
57	(2.67)	152.30
190	(0.77)	146.71

29	0.95	27.55
11	0.75	8.25
10	1.31	13.10
11	1.24	13.64
10	1.09	10.90
15	0.95	14.25
11	1.32	14.52
8	0.57	4.56
9	0.59	5.31
12	0.63	7.56
126	(0.95)	119.64
57	(1.17)	66.41
69	(0.77)	53.23

6	0.87	5.22
7	0.60	4.20
9	0.27	2.43
7	0.30	2.10
7	0.18	1.26
9	0.20	1.80
10	1.02	10.20
8	0.25	2.00
12	0.84	10.08
13	0.28	3.64
10	0.86	8.60
8	1.40	11.20
7	0.12	0.84
7	2.37	16.59
11	0.37	4.07
9	0.23	2.07
9	0.20	1.80
8	0.19	1.52
157	(0.57)	89.62
22	1.30	28.63
135	0.45	60.99

Appendix ii

Oxide-sulphide ratios & associated metal values

MEMO

Date: June 19, 1974

To: Steve Radvak

From: J. W. Simpson

Re: Desoto Property - Recent Assays

(1) Oxide-Sulphide Ratios

The attached summaries of assays for the East Zone (Table 1) show that the copper values obtained from surface holes are primarily derived from oxides for the first 50 feet. Percussion hole results show a substantial drop in the oxide vs. sulphide abundance below 50 feet from surface and another drop below 100 feet. The consistency of this ratio in diamond drill holes is difficult to explain. It may be that, at depth, some copper oxides were not being recovered by percussion drilling.

Underground drilling values are virtually all from sulphide.

For the West Zone it can be seen from the attached assay summary at mineralized sections (Table 2) that below 200 feet from surface over 75% of the copper values are in sulphide. Above this level the oxide-sulphide ratio is almost 50% and unfortunately it is not possible to calculate this exactly with given data.

(2) Precious Metal Values

On the East Zone Composites, gold averaged 0.05 oz/ton and silver ran 0.20 oz/ton. If only one half of these values are recovered and paid for by the smelter then about $4.50 (todays prices) can be added to copper returns.

Values on the East Zone drilling cannot be calculated with available data but I imagine values from surface drilling would be similar and those from underground drilling would be significantly higher.

Wait June

(3) DDH-14 results were obtained after I left the property at the beginning of June and are herewith attached.

The zinc values are unusually high, averaging 0.53% over 121', from 283 to 404 feet.

From 323 to 393 feet (70') a composite sample averaged 1.08% Cu, 0.36% Zn, 0.02 oz Au per ton and 0.60 oz Ag per ton. An arithmetic average of the individual assays from this interval is 1.09% Cu which is excellent correlation. This is best grade intersection on the West Zone.

J.W.S. (RWR)

J. W. Simpson

JWS;rr

cc: Murray Pezim

TABLE 1

East Zone Composites for Oxide-Sulphide Ratio Study

Diamond Drill Hole Composite
(DDH 1-4 incl.)

Depth	Total Cu (%)	Oxide Cu (%)	Oxide/Sulphide Ratio
0-50'	0.34	0.30	88%
50-100'	0.91	0.79	87%
100'-plus	0.87	0.75	86%
	2.12/3 = 0.71	1.84/[illegible] 0.61	[illegible]

Percussion Hole Composite
(P 1-18 incl. and P 29,33,34)

Depth	Total Cu (%)	Oxide Cu (%)	Oxide/Sulphide Ratio
0-50'	0.42	0.37	82% [illegible]
50-100'	0.43	0.24	~~45%~~ 56
100- plus	0.78	0.25	32% [illegible]
	1.59/3 = 0.53	0.86/[illegible] = 0.29	55
	3.71/6 = 0.62	2.70 6 0.45	73

DESOTO MINE CLEATOR, ARIZONA

WEST ZONE COMPOSITES

HOLE #	INTERVAL	Cu%	Zn%	oz/ton Au	oz/ton Ag	Oxide-Sulphide Ratio for Cu
P-39	170-220'	0.51	0.20	Tr	0.24	80%
P-39A	70-160'	0.32	0.10	Tr	Tr	59%
H-1	30-60'	0.71	0.42	.020	0.58	82%
H-2	70-240'	0.56	0.30	0.10	0.39	27%
H-4	100-180'	0.49	0.20	.005	0.20	41%
H-7	140-210'	0.71	0.26	0.10	0.27	45%
H-8	180-320'	0.76	0.24	0.10	0.19	25%
H-11	230-250'	0.59	0.18	.005	0.06	5%
DDH-7	234-386'	0.71	0.26	Tr	0.10	14%
DDH-8	314-317'	1.78		.005	0.22	1%
DDH-10	180-190'	1.29		.005	0.26	2%
DDH-12	407-431'	1.04	0.30	.010	0.25	1%
DDH-13	278-308'	1.04	0.38	.025	0.24	7%
Total Footage	869'			Ave. Precious Metals .05	.20	

Less H-2

Less H-7

45%

57%

61%

Appendix iii

Associated metal values

CUTLASS EXPLORATION, INC.

Assay Results - Desoto Mine - Cleator, Arizona

Hole #	Footage	Gold	Silver	%Pb	%Zn	%Cu	SUMMARY %Cu	SUMMARY Au	SUMMARY Ag	SUMMARY Interval
U-1	1-1½ & 15-19					0.30				
	19-28					0.18				
	28-38					0.50				
	38-48					0.08				
	48-58					1.51	1.24			
	58-68					1.03				
	68-77					0.95				
U-2	0-12½					0.30	.52			
	12½-23					0.73				
	23-33					0.05				
	33-43					0.04				
	43-53					0.04				
	53-63					0.09				
	63-72					0.05				
	72-82					0.02				
	83-87					0.03				
U-3	7-16½	.54	2.38		2.65	6.12				
	16½-20½				.10	.12				
	20½-26				.06	2.17	2.0	.09	1.09	38'
	26-35				Tr	.34				
	35-45				.45	.33				
	45½-55½					0.15				
	55½-66½					0.04				
	66½-77½					0.03				
	77½-88½					0.04				
	88½-98½					0.04				
	98½-109½					0.03				
	109½-120½					0.03				
	120½-131½					0.03				
	131½-141½					0.03				
	141½-151½					0.03				
	151½-153½					0.42				
U-4	2-11					0.04				
	11-21					0.06				
	21-31					0.08				
	31-41	.010	0.39	Tr	0.30	0.75				
	41-51			0.06	0.28	0.31				
	51-67					0.15	.54			59'
	67-77	.025	0.64	Tr	0.03	0.34				
	77-90	.010	0.29	0.04	1.76	1.20				
	90-100					0.15				
	100-110					0.04				
	110-120					0.18				
	120-130					0.03				
	130-140					0.03				
	140-152					0.04				
	152-162					0.21				
	162-172	.040	0.44	0.06	0.60	2.06				
	172-182					0.03				
	182-187			0.04	0.40	0.40				
	187-197					0.03				
	197-207					0.06				
	207-212					0.05				

							SUMMARY			
Hole #	Footage	Gold	Silver	%Pb	%Zn	%Cu	%Cu	Au	Ag	Interval
U-5	7-17					0.24				
	17-27					0.21				
	27-41				0.34	0.37				
	41-51					0.06				
	51-61					0.04				
	61-78½				0.14	0.21				
	78½-87½	.165	1.80		1.24	4.94				
	87½-97½	.030	0.95		1.20	2.44				
	97½-103	.060	0.36		0.18	0.99	2.21			44½'
	103-113					0.29				
	113-123					2.12				
	123-133				0.20	0.11				
	133-143				0.14	0.19				
	143-158				0.16	0.25	0.30			65'
	158-168					0.64				
	168-178					0.24				
	178-188				0.22	0.39				
	188-198				0.10	0.03				
	198-208				0.10	0.04				
	208-218				0.20	0.06				
	218-228				0.30	0.08				
	228-238				0.32	0.13				
	238-248				0.26	0.03				
	248-264				0.26	0.03				
	264-274					0.17				
	274-284					0.98				
	284-294					0.76				
	294-304				0.20	0.79	1.1			56'
	304-314				0.22	1.68				
	314-330	.005	0.16		0.22	1.17				
	330-340				0.18	0.17				
	340-350				0.14	0.06				
	350-360				0.18	0.03				
	360-373				0.22	0.08				

development. The mill was installed in 1920 and is reported to yield a concentrate containing 44 ounces of silver and $10 in gold to the ton, besides 6 per cent of copper and 11 per cent of lead.[49]

BLUE BELL MINE

History.—The Blue Bell mine has for many years been the property of the Consolidated Arizona Smelting Co. and has yielded a large quantity of low-grade pyritic copper ore. It is now held by the successor of this company, the Southwest Metals Co., which also operates the Humboldt smelter. Jaggar and Palache in 1901 mentioned the property briefly, but it was not until 1906, when the mine was transferred to the Consolidated Arizona Smelting Co., that it began to acquire importance. Since then it has developed into the largest producer in the Bradshaw Mountains. The total production of ore to 1921, inclusive, was 800,000 tons of copper ore, with an average gross value of $10 a ton. In common with many other mines the property was idle in 1921, but it reopened in 1922 at the same time as the Humboldt smelter.

Development.—The Blue Bell mine is 4 miles south of Mayer, at an altitude of 4,500 feet. (See pl. 18, *A*.) It has modern, electrically driven equipment and is connected with the railroad siding by an aerial tram 1 mile long. There is a concentrating plant of 350 tons capacity at the Humboldt smelter. The developments consist of a vertical shaft 1,400 feet deep in 1922 with almost 30,000 feet of workings north and south. There are five smaller shafts.

Production.—The total output since 1906 of about 900,000 tons has in recent years been distributed as follows, according to the published figures of the company:

Copper, gold, and silver produced at Blue Bell mine, 1903–1925

[Compiled by V. C. Heikes, U. S. Geological Survey]

Year	Crude ore	Copper	Gold	Silver
	Tons	*Pounds*	*Fine ounces*	*Fine ounces*
1903	392	43, 120	43. 12	921
1904				
1905				
1906	4, 452	204, 230	204. 00	4, 855
1907	18, 342	789. 731	640. 59	16, 007
1908				
1909	1, 175	82, 250	59. 00	3, 055
1910	26, 726	1, 997, 166	1, 342. 00	39, 937
1911	35, 783	2, 667, 332	1, 925. 20	50, 344
1912	22, 741	1, 157, 008	822. 25	24, 347
1913	37, 812	1, 976, 363	1, 021. 00	30, 657
1914	56, 058	3, 673, 696	1, 872. 58	55, 348
1915	82, 171	4, 218, 260	1, 882. 00	64, 805
1916	75, 070	3, 568, 913	2, 899. 55	76, 065
1917	102, 773	6, 528, 635	6, 062. 00	150, 116
1918	131, 090	7, 263, 779	8, 251. 05	202, 995
1919	122, 069	5, 546, 996	5, 594. 36	144, 501
1920	111, 749	4, 989, 565	5, 544. 00	129, 560
1921				
1922	24, 716	1, 519, 584	1, 081. 42	29, 058
1923	81, 028	4, 855, 575	2, 660. 78	77, 043
1924	10, 111	850, 338	691, 50	17, 069
1925	13, 454	68[illegible], 771	846. 23	20, 767

[49] Weed, W. H., Mines Handbook, 1922.

Geology.—The deposit is contained in the Yavapai schist, which here is of complex character, including ledges of quartzite, biotite schist, small lenses of crystalline limestone, chloritic schist, clay slate, and schistose quartz porphyry. The strike is N. 27° 30′ W. and the dip 70° W.

A reef of brownish quartzite crops out prominently at the village below the mine, and close by is some light-gray fissile clay slate. Quartzite also occurs on the 1,400-foot level. Another specimen collected from the same level is a dark-green dense schist which looks like hornfels and contains dirty-green biotite, magnetite, zoisite, chlorite, and quartz mosaic. Still another specimen from the same level is a greatly crushed quartz porphyry with microcrystalline groundmass of quartz and orthoclase, with some microcline, albite, and sericite. The quartz phenocrysts are also greatly crushed.

Veinlets of calcite and quartz are abundant in the mine. A persistent dike is exposed on all levels and is not affected by schistosity. A thin section of this dike shows partly idiomorphic augite, brown hornblende, some of it with kernels of augite, lathlike labradorite, magnetite, and apatite. Secondary chlorite and sericite are present. The grains average 1 to 2 millimeters in size. This is a granular dike rock related to camptonite. This dike is generally vertical and intersects ore about 200 feet south of the shaft. It shows no mineralization.

The ore body.—The ore body is essentially a silicified and mineralized zone that conforms to the schist. The width is about 100 feet. The ores form a series of about six flat lenses within the zone; in part they overlap and they pitch about 75° S. in the zone. These lenses are as much as 40 feet wide and occur on both foot and hanging walls.

The stopes stand well; some of them are 80 feet high and 20 feet wide. Relatively to the walls of the zone, each shoot keeps its position well. Most of them continue from points near the surface down. The surface is generally barren, probably as a result of leaching. The fourteenth level was just opened in 1922. The developed length of the deposit is 1,600 feet. (See pl. 16.)

Structure.—The ore bodies are intersected by several faults that dip 30°–40° NW., thus intersecting the lenses at an oblique angle. The faults are of the reverse type and show a slip of 50 to 100 feet. One fault observed on the 500-foot level strikes east and dips 60° S., and the fault plane shows striations parallel both to strike and dip. The faults cut the ore cleanly and show no mineralization.

The walls of the ore shoots are mostly well defined, but in some places they show transitions to silicified country rock. Locally on such walls a groove structure is noted which conforms with the southward pitch of the shoot.

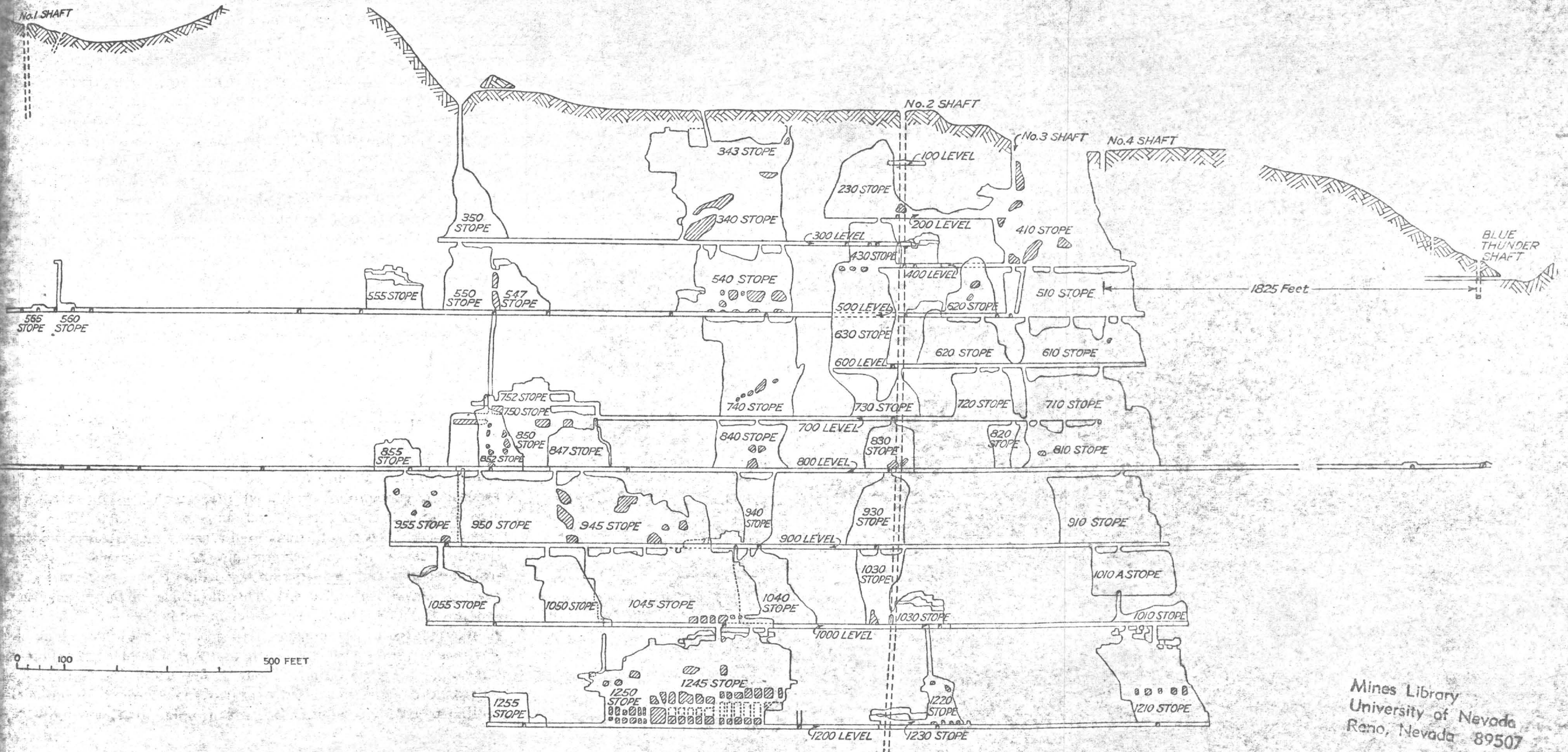
The ore.—The most common country rock is a chlorite-biotite schist. Within the silicified zone it may, however, be difficult to recognize the original character of the rock. The silicified material may contain sparse sulphides, but as a rule the ore is well defined. The ore is classified as heavy smelting ore and siliceous concentrating ore and averages 3 per cent in copper. The smelting ore contains also 1.5 ounces of silver and 0.05 ounce of gold to the ton; the siliceous ore averages 1 ounce of silver and 0.03 ounce of gold to the ton. As a rule the siliceous ore contains a little less gold and silver than the smelting ore.

The smelting ore is usually massive and rather fine-grained; it consists of pyrite intergrown with more or less chalcopyrite and containing spots of quartz and imperfectly replaced schist. A little calcite is universally present, but a fine-grained quartz mosaic replacing the schist is the principal gangue. The siliceous ore usually shows the schistose structure of the original rock and contains streaks of replacing sulphides. The minerals are pyrite and chalcopyrite, with very small amounts of arsenopyrite, sphalerite, and galena. No tetrahedrite was observed.

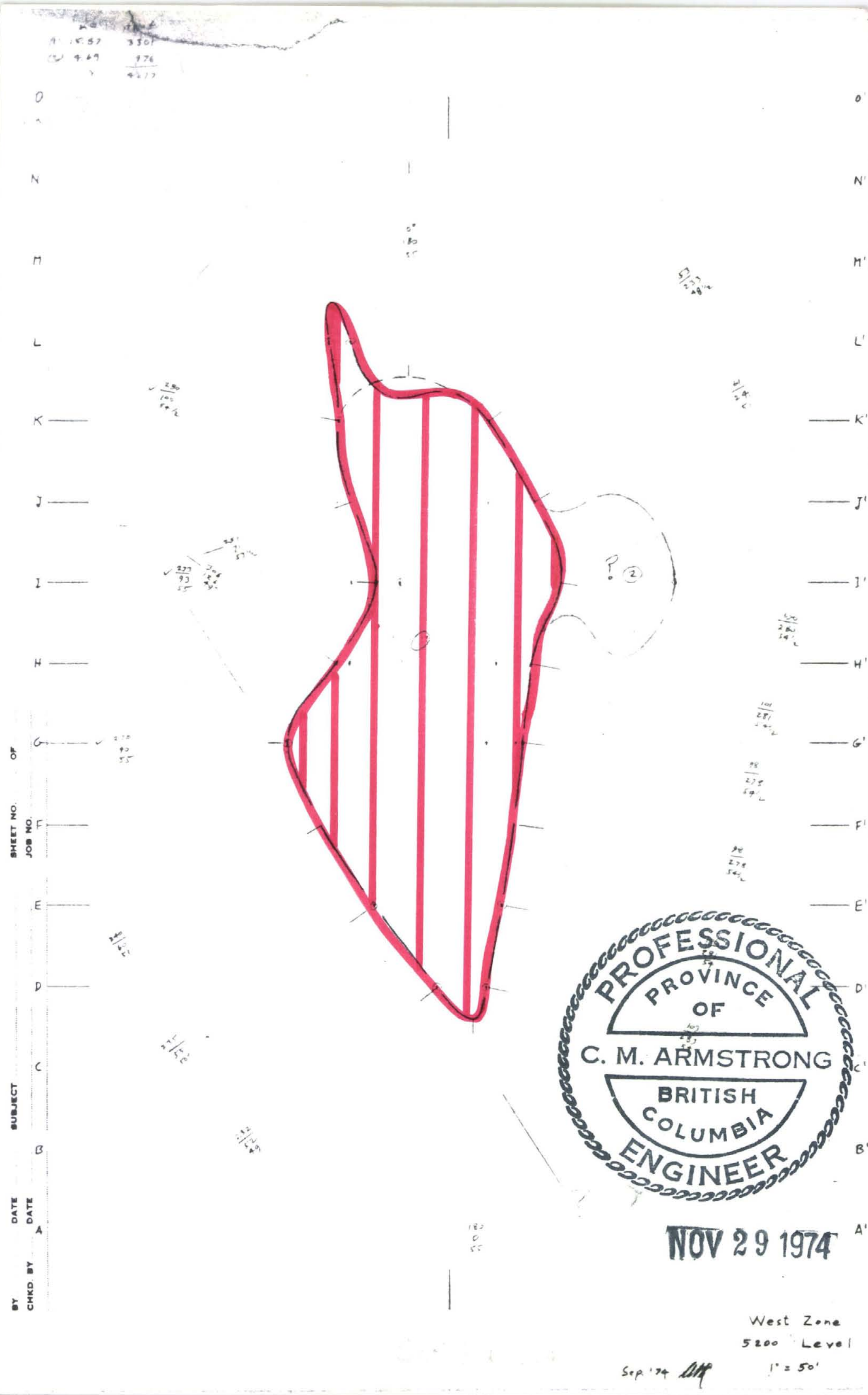
Polished sections show rude crystals and rounded grains of pyrite, fractured, cemented, and replaced by chalcopyrite and the other scant sulphides, among which dark sphalerite is the most abundant. (See pls. 9, *B;* 17, *B.*)

The ore shows less ankerite and more quartz than usual in the schist replacement deposits, but the succession of minerals is the same: quartz and carbonates are the oldest, followed by pyrite and later by chalcopyrite. The same dark-green iron-rich chlorite that was observed at Jerome appears here again in places.

Oxidation and water.—At the surface the ore is leached and rusty brown; there is no chalcocite zone proper, but a little chalcocite may be found in places down to the 1,200-foot level. On the 400-foot level a little oxidation is observed, and 1,800 feet farther north on the 500-foot level the drift encountered a 2-foot vein of chrysocolla and cuprite fully oxidized and containing no silver. The workings are rather warm, and the mine water is acidic and contains much copper. The original water level was probably at the 400-foot level; the mine makes little water, say 150,000 gallons in 24 hours.



LONGITUDINAL SECTION OF BLUE BELL MINE



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JOB NO.
SUBJECT
DATE
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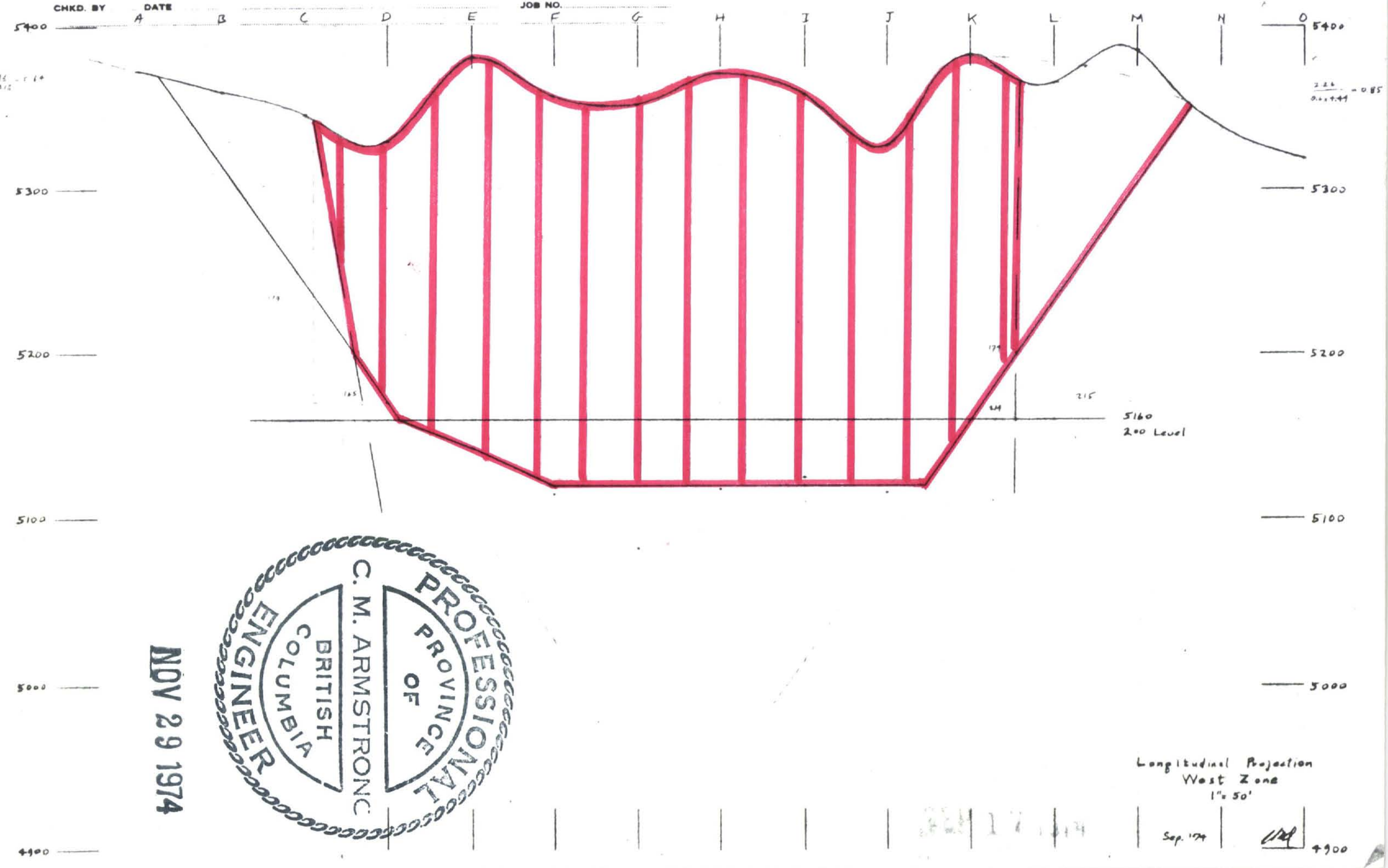
BY
CHKD. BY
PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG
NOV 29 1974
West Zone
5200 Level
1" = 50'
Sep. '74

BY DATE SUBJECT SHEET NO. OF
CHKD. BY DATE JOB NO.



NOV 29 1974

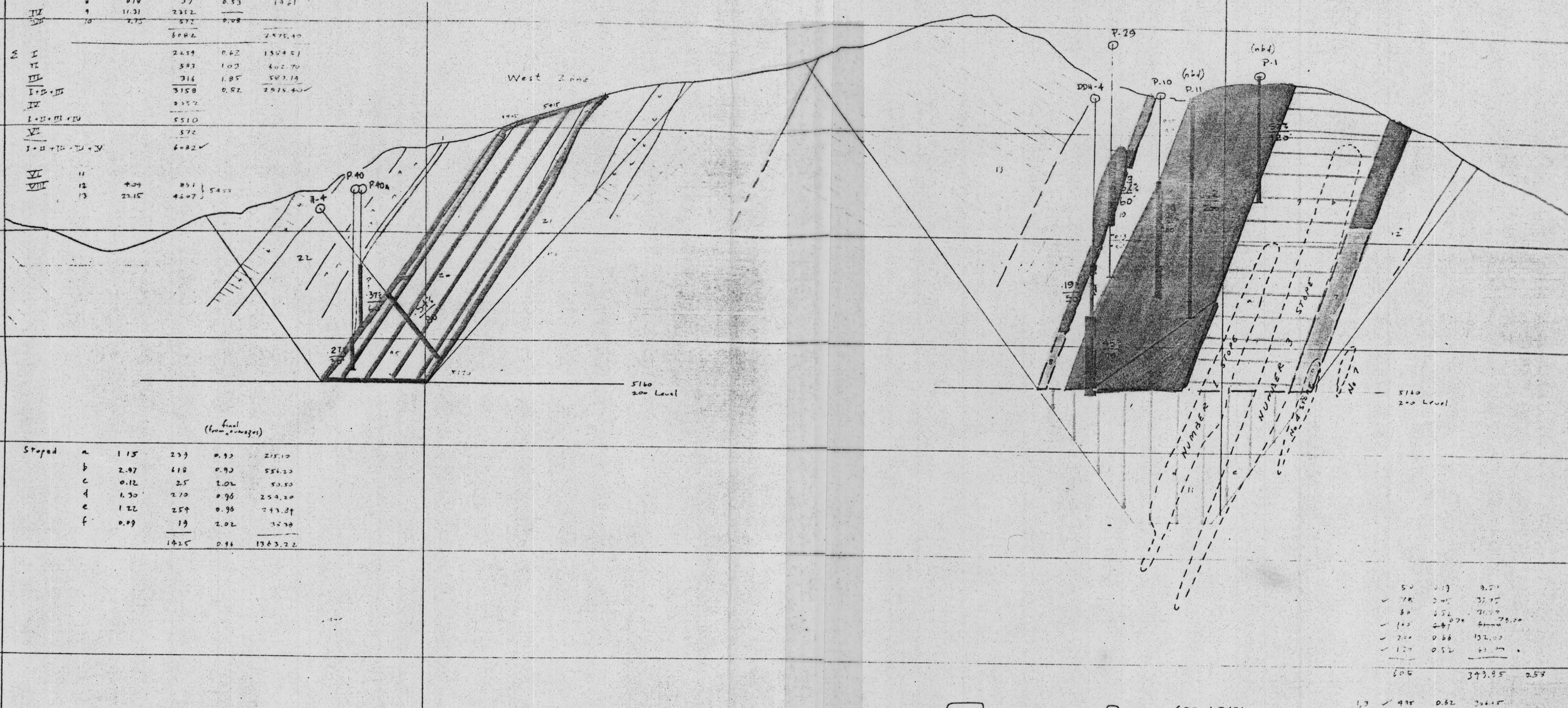
East Zone
5300 Level

SEP 17 1974

BY DATE SUBJECT SHEET NO. OF
CHKD BY DATE JOB NO.

East Zone

Category	Block	in□	tph£	Cu	Product
I	1	10.27	2136	0.62	1324.32
	2	0.59	123	0.53	65.19
II	3	1.44	300	0.62	186.00
	4	0.27	56	0.53	29.68
	5	0.23	49	0.53	25.44
	6	0.86	179	2.02	361.58
III	7	1.34	279	2.02	563.58
	8	0.18	37	0.53	19.61
IV	9	11.31	2352		
VII	10	2.75	572	0.08	
			6082		2575.40
Σ I			2259	0.62	1389.51
II			583	1.03	602.70
III			316	1.85	583.19
I+II+III			3158	0.82	2575.40 ✓
IV			2352		
I+II+III+IV			5510		
VI			572		
I+II+III+IV+V			6082 ✓		
VI	11				
VIII	12	4.09	851 } 5458		
	13	22.15	4607		



(from final averages)

Stoped	a	1.15	239	0.90	215.10
	b	2.97	618	0.90	556.20
	c	0.12	25	2.02	50.50
	d	1.30	270	0.96	259.20
	e	1.22	254	0.96	243.84
	f	0.09	19	2.02	38.38
			1425	0.96	1363.22

	50	0.19	9.50	
✓	75	0.45	33.75	
	60	0.52	31.20	
✓	100	~~2.81~~ 0.78	~~[illegible]~~ 78.00	
✓	200	0.66	132.00	
✓	120	0.52	62.40	
	605		349.85	0.58
1,3 ✓	495	0.62	306.45	
	90	0.05	4.50	
	50	0.13	6.50	
10	140	0.08	11.00	

West Zone

Category	Block	in□	tph£
O	20	9.03	1878
W	21	4.11	855
	22	10.51	2136
			4919 ✓
Σ W			3041
O+W			4919

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG

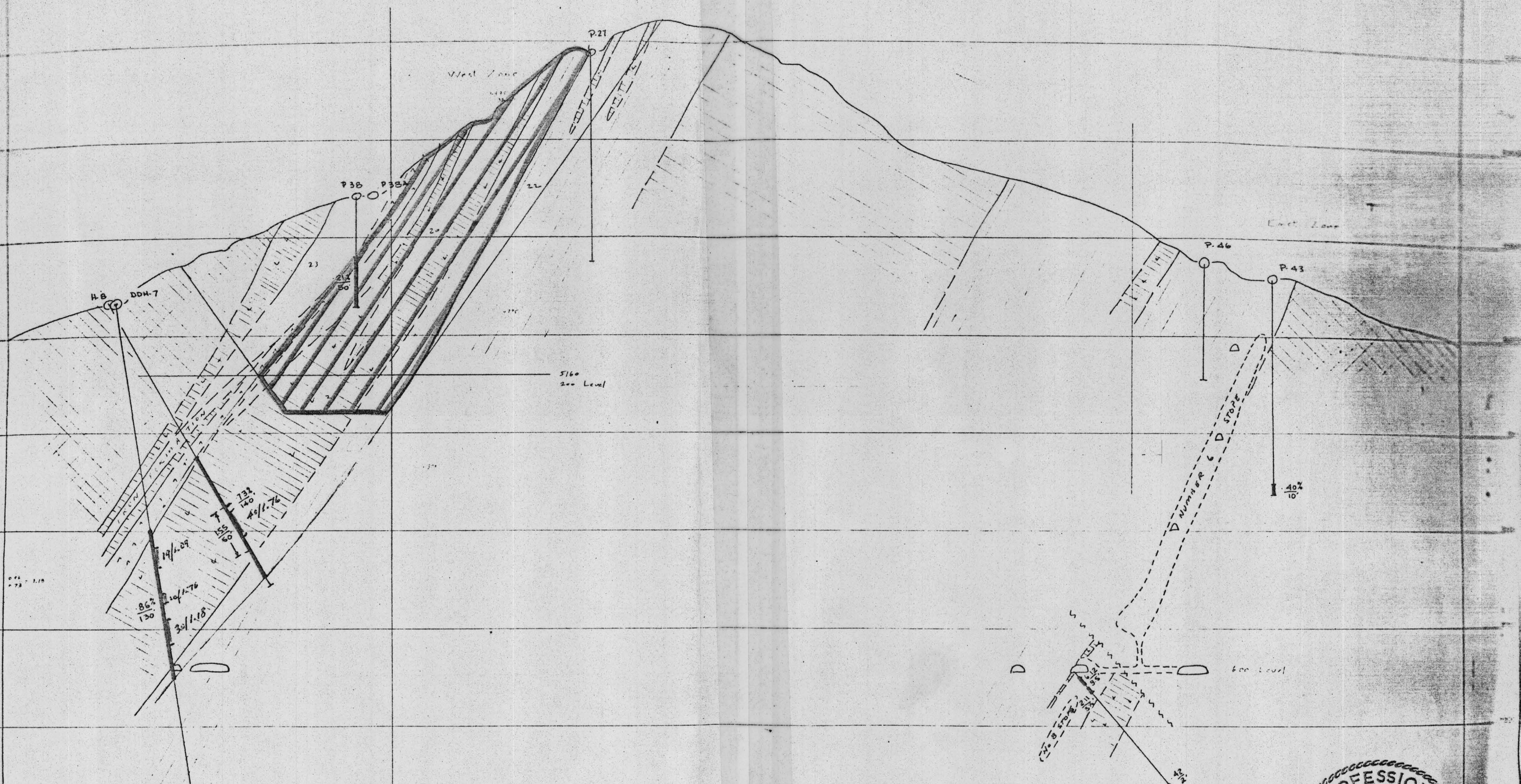
NOV 29 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

SECTION K-K'

YAVAPAI COUNTY, ARIZONA



West Zone

Category	Block	in²	tph L
O	20	14.42	2999
	21	2.12	441
W	22	3.76	782
	23	7.28	1514
			5736
Σ O			3440
W			2296
O+W			5736

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C. M. ARMSTRONG

NOV 29 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

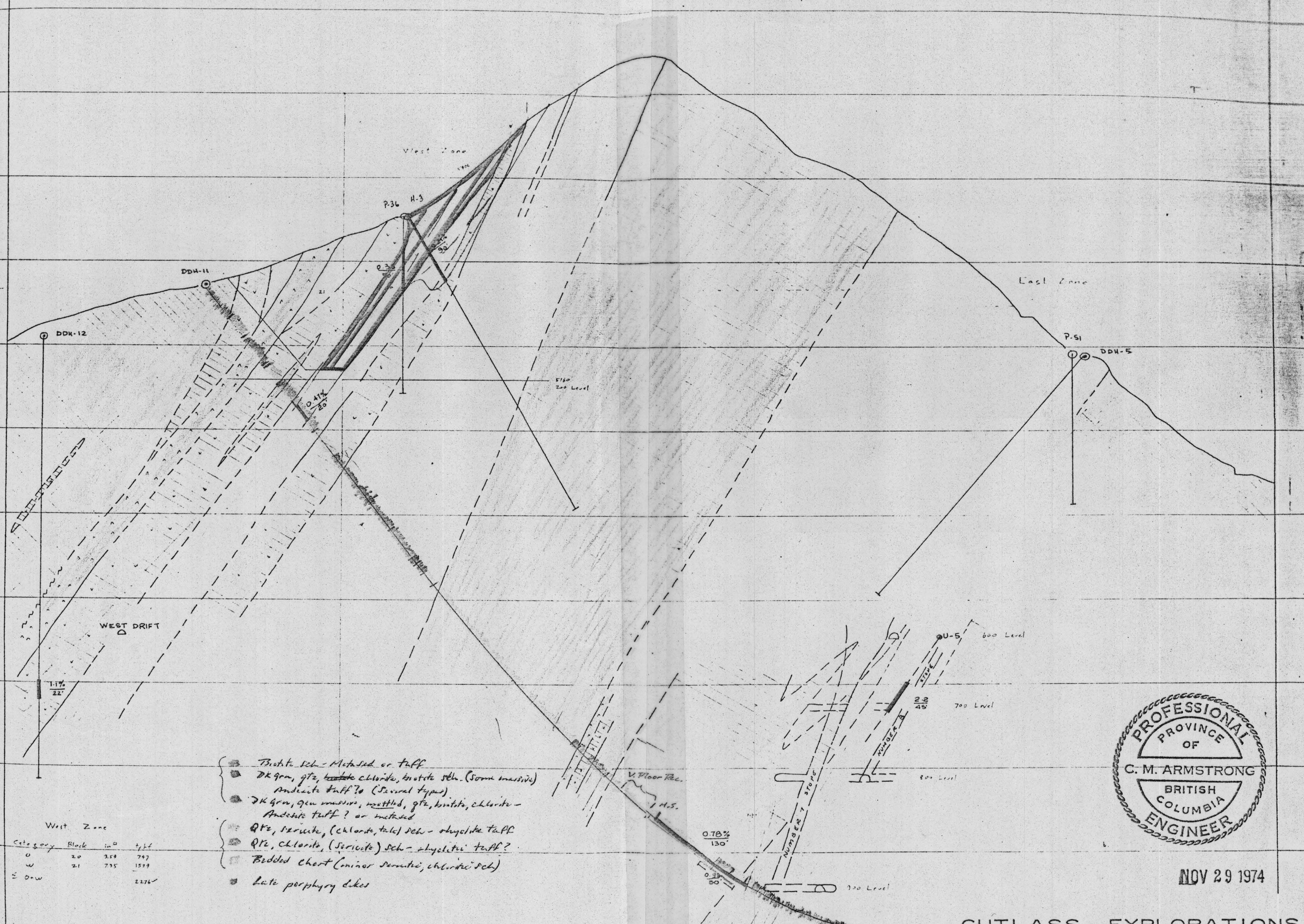
DE SOTO MINE

SECTION G·G'

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET



NOV 29 1974

CUTLASS EXPLORATIONS

LTD. (N.P.L.)

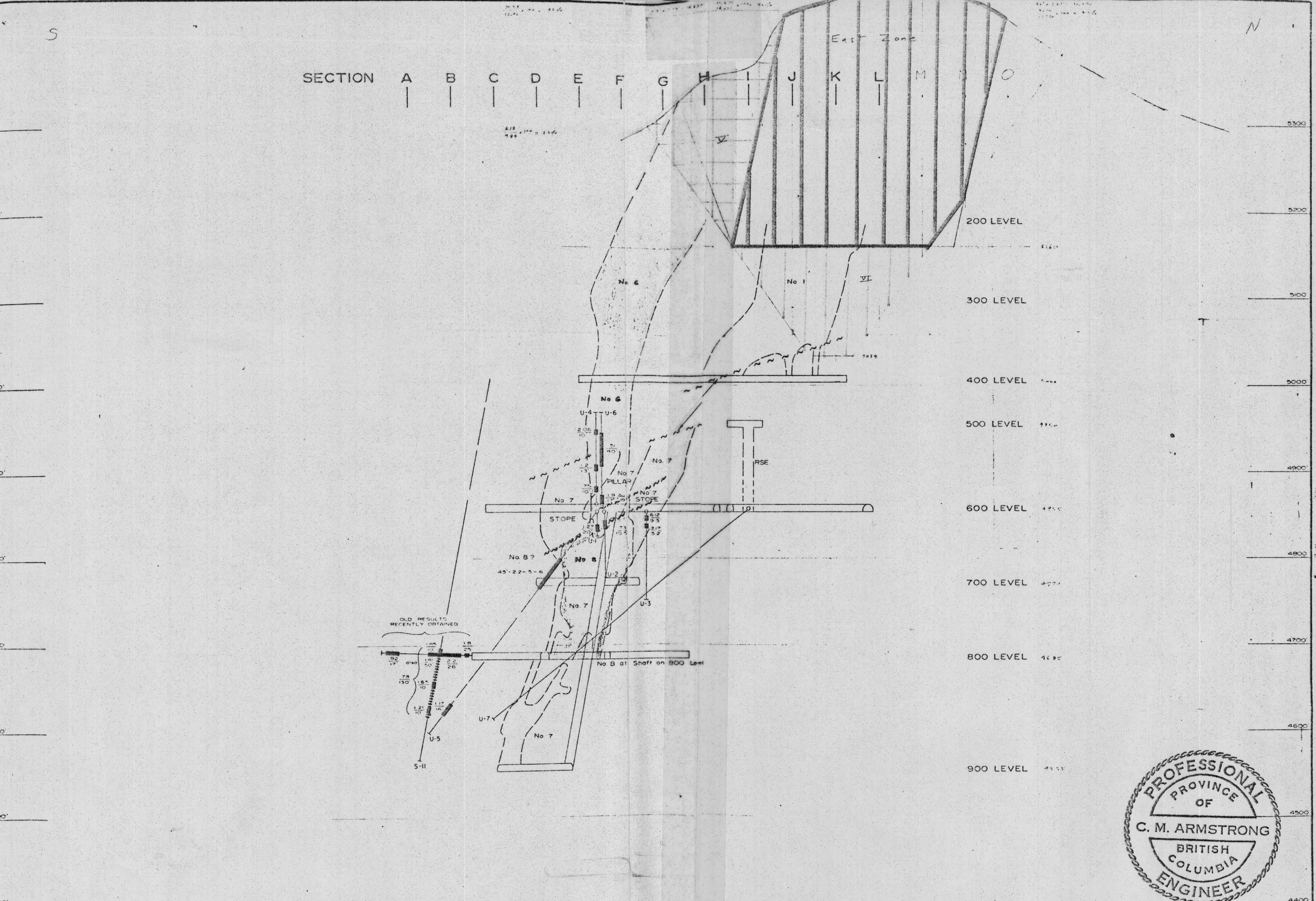
DE SOTO MINE

SECTION D-D'

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET



NOV 29 1974

LOOKING WEST

CUTLASS EXPLORATIONS

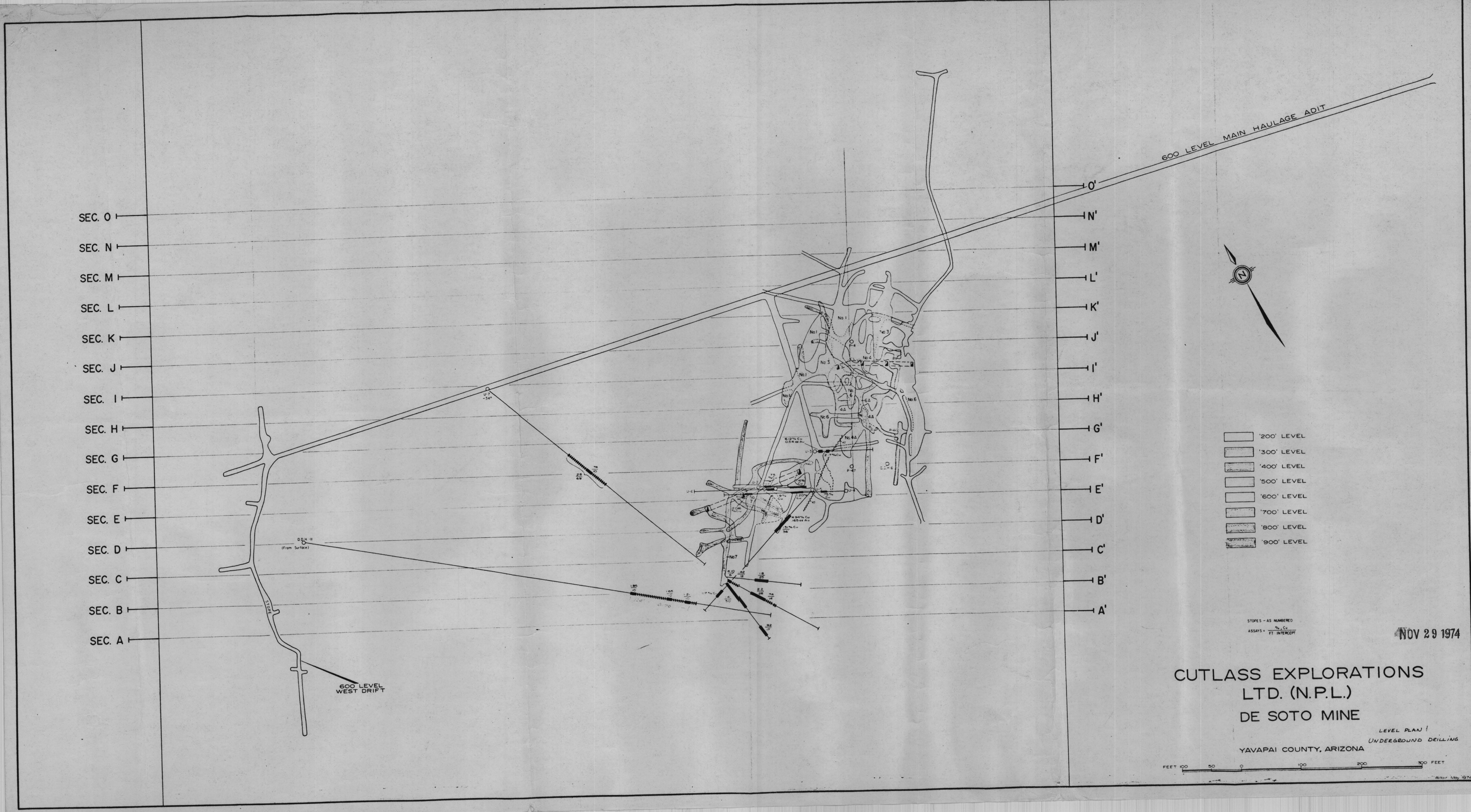
LTD. (N.P.L.)

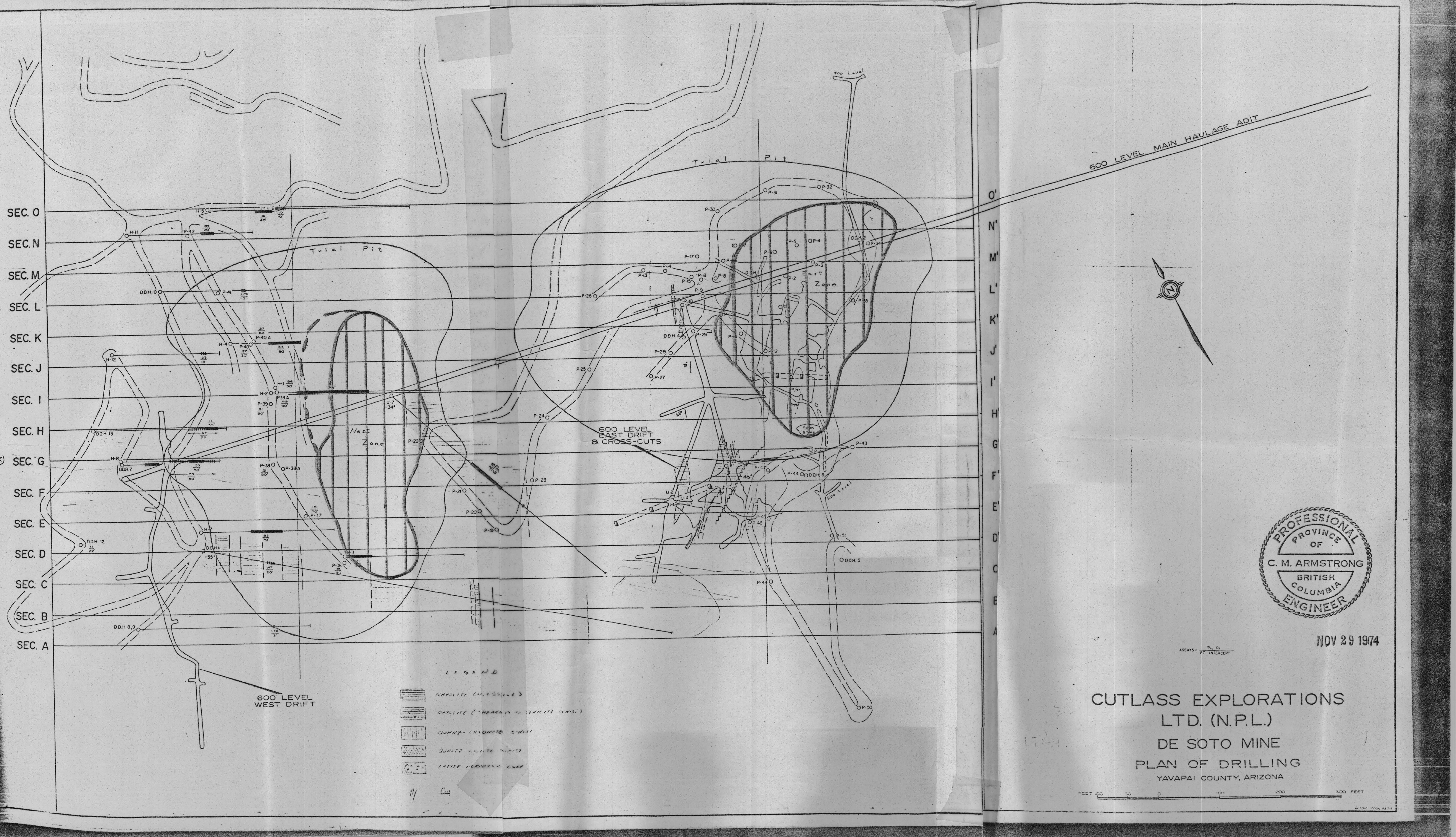
DE SOTO MINE

LONGITUDINAL SECTION East Zone

YAVAPAI COUNTY, ARIZONA

SCALE

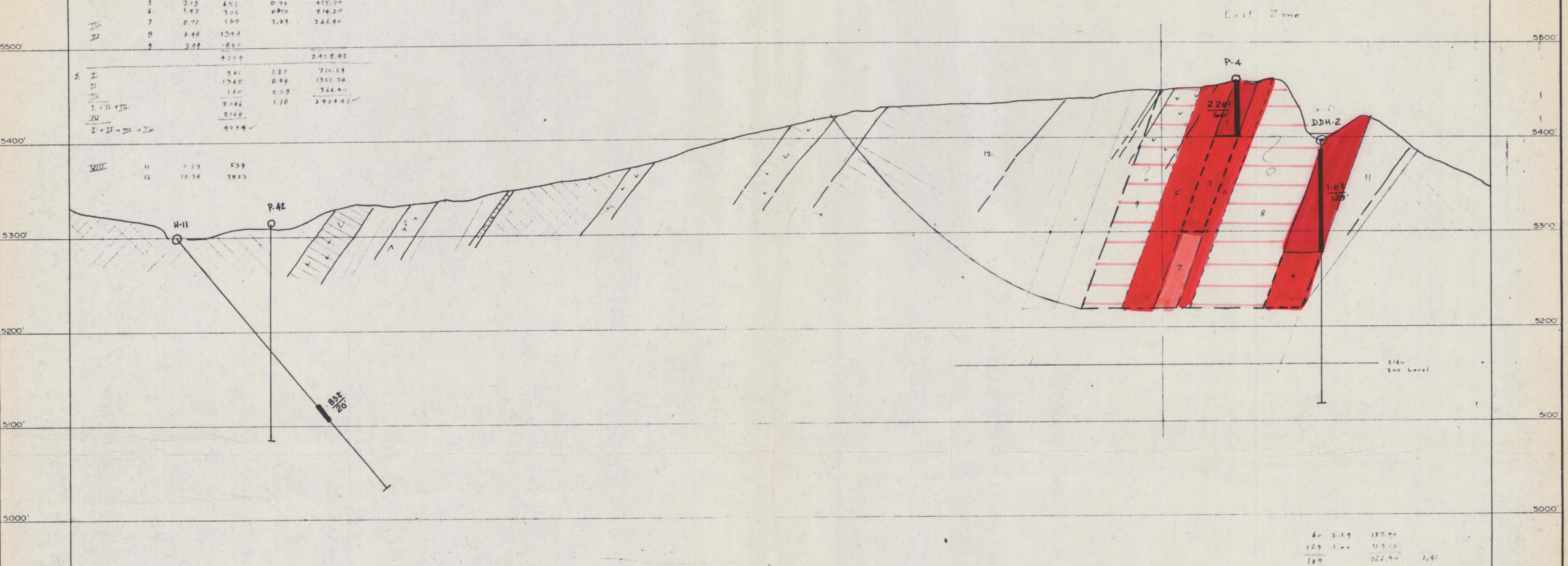

600 LEVEL MAIN HAULAGE ADIT
SEC. O
SEC. N
SEC. M
SEC. L
SEC. K
SEC. J
SEC. I
SEC. H
SEC. G
SEC. F
SEC. E
SEC. D
SEC. C
SEC. B
SEC. A
O'
N'
M'
L'
K'
J'
I'
H'
G'
F'
E'
D'
C'
B'
A'
600 LEVEL
WEST DRIFT
STOPE
No 7
'200' LEVEL
'300' LEVEL
'400' LEVEL
'500' LEVEL
'600' LEVEL
'700' LEVEL
'800' LEVEL
'900' LEVEL
STOPES - AS NUMBERED
ASSAYS = %, Cu / FT INTERCEPT
NOV 29 1974
CUTLASS EXPLORATIONS
LTD. (N.P.L.)
DE SOTO MINE
LEVEL PLAN !
UNDERGROUND DRILLING
YAVAPAI COUNTY, ARIZONA
FEET 100 50 0 100 200 300 FEET
Aitoir May 1974



SEC. O
SEC. N
SEC. M
SEC. L
SEC. K
SEC. J
SEC. I
SEC. H
SEC. G
SEC. F
SEC. E
SEC. D
SEC. C
SEC. B
SEC. A
O'
N'
M'
L'
K'
J'
I'
H'
G'
F'
E'
D'
C
B
A
Trial Pit
Trial Pit
West Zone
East Zone
600 LEVEL MAIN HAULAGE ADIT
600 LEVEL EAST DRIFT & CROSS-CUTS
600 LEVEL WEST DRIFT
300 Level
200 Level
LEGEND
N
PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG
NOV 29 1974
CUTLASS EXPLORATIONS LTD. (N.P.L.)
DE SOTO MINE
PLAN OF DRILLING
YAVAPAI COUNTY, ARIZONA
FEET 100 50 0 100 200 300 FEET

East Zone

Category	Block	t_n	tphf	Cu	Product
I	1	0.56	116	2.29	265.64
	2	2.14	445	1.00	445.00
II	3	1.02	212	2.29	485.48
	4	0.94	196	1.00	196.00
	5	3.13	651	0.70	455.70
	6	1.47	306	0.70	214.20
III	7	0.77	160	2.29	366.40
IV	8	6.48	1348		
	9	3.94	820		
			4259		2428.42
Σ I			561	1.27	710.64
II			1365	0.99	1351.38
III			160	2.29	366.40
I + II + III			2086	1.16	2428.42
IV			2168		
I + II + III + IV			4254		
VIII	11	2.59	539		
	12	18.38	3823		



60	2.29	137.40	
129	1.00	129.00	
189		266.40	1.41

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PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG

NOV 29 1974

SEP 17 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

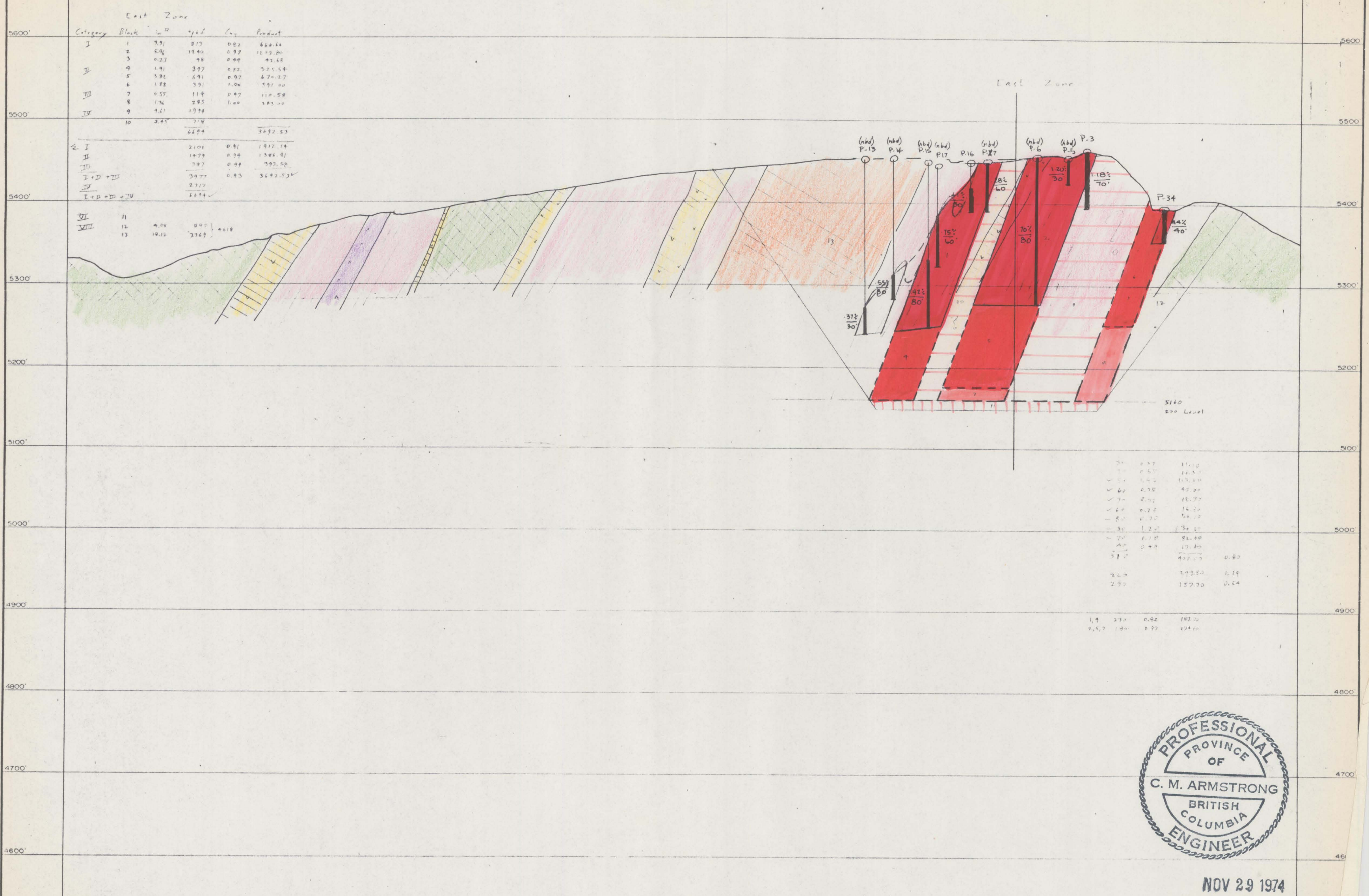
DE SOTO MINE

SECTION N-N'

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET



CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

SECTION M-M'

YAVAPAI COUNTY, ARIZONA

SCALE

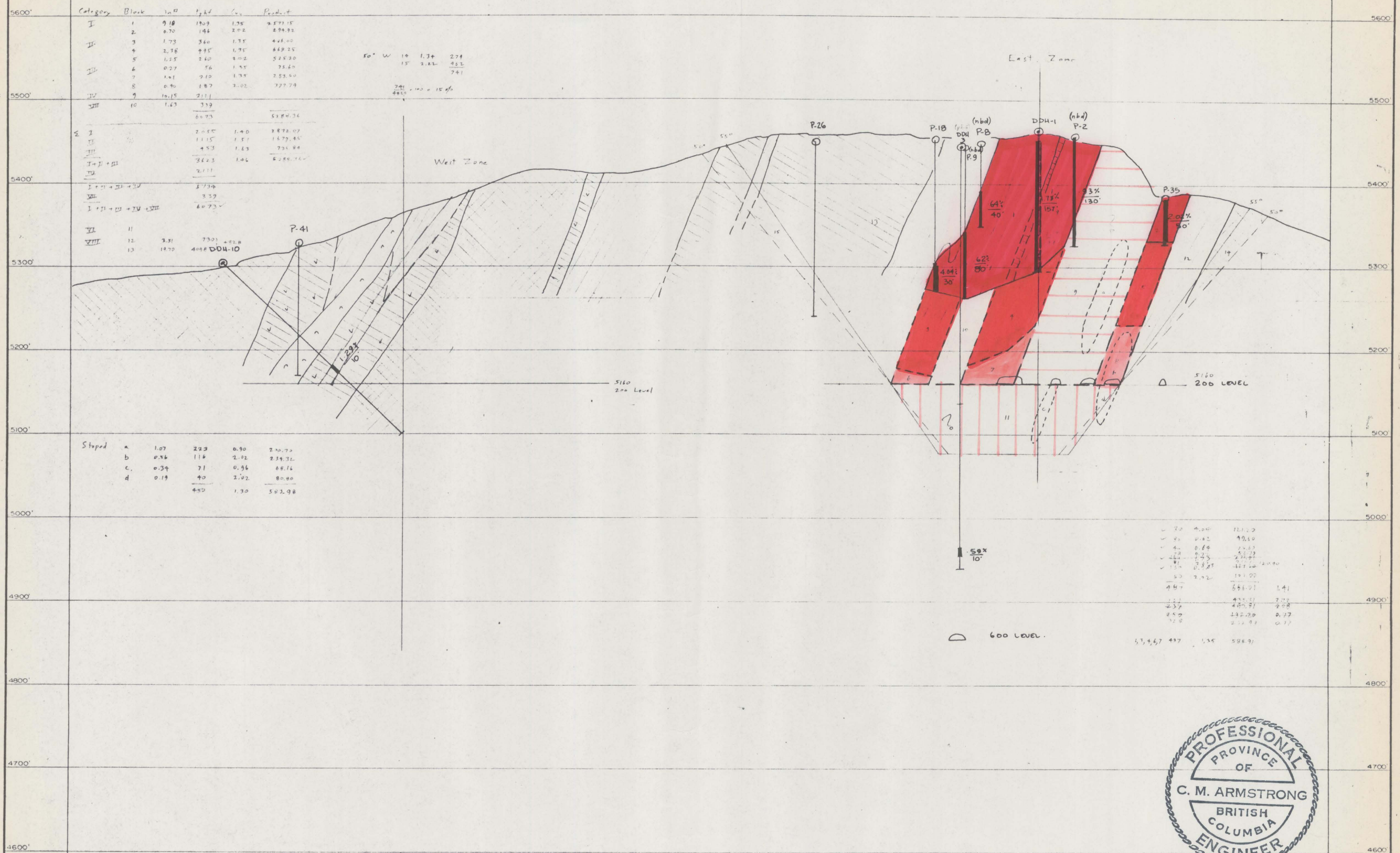
FEET 50 25 0 50 100 150 200 FEET

SEP 17 1974

Altair MAY-1974

East Zone

Category	Block	in □	t/ht	Cu	Product
I	1	9.18	1909	1.35	2577.15
	2	0.70	146	2.02	294.92
II	3	1.73	360	1.35	486.00
	4	2.38	495	1.35	668.25
	5	1.25	260	2.02	525.20
III	6	0.27	56	1.35	75.60
	7	1.01	210	1.35	283.50
	8	0.90	187	2.02	377.74
IV	9	10.15	2111		
VII	10	1.63	339		
			6073		5288.36
Σ I			2055	1.40	2872.07
II			1115	1.51	1679.45
III			453	1.63	736.84
I + II + III			3623	1.46	5288.36
IV			2111		
I + II + III + IV			5734		
VII			339		
I + II + III + IV + VII			6073		
VI	11				
VIII	12	3.51	7301		
	13	19.70	4098		



Stoped	a	1.07	223	0.90	200.70
	b	0.56	116	2.02	234.32
	c.	0.34	71	0.96	68.16
	d	0.19	40	2.02	80.80
			450	1.30	583.98

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PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG

NOV 29 1974

SEP 17 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

SECTION L-L'

YAVAPAI COUNTY, ARIZONA

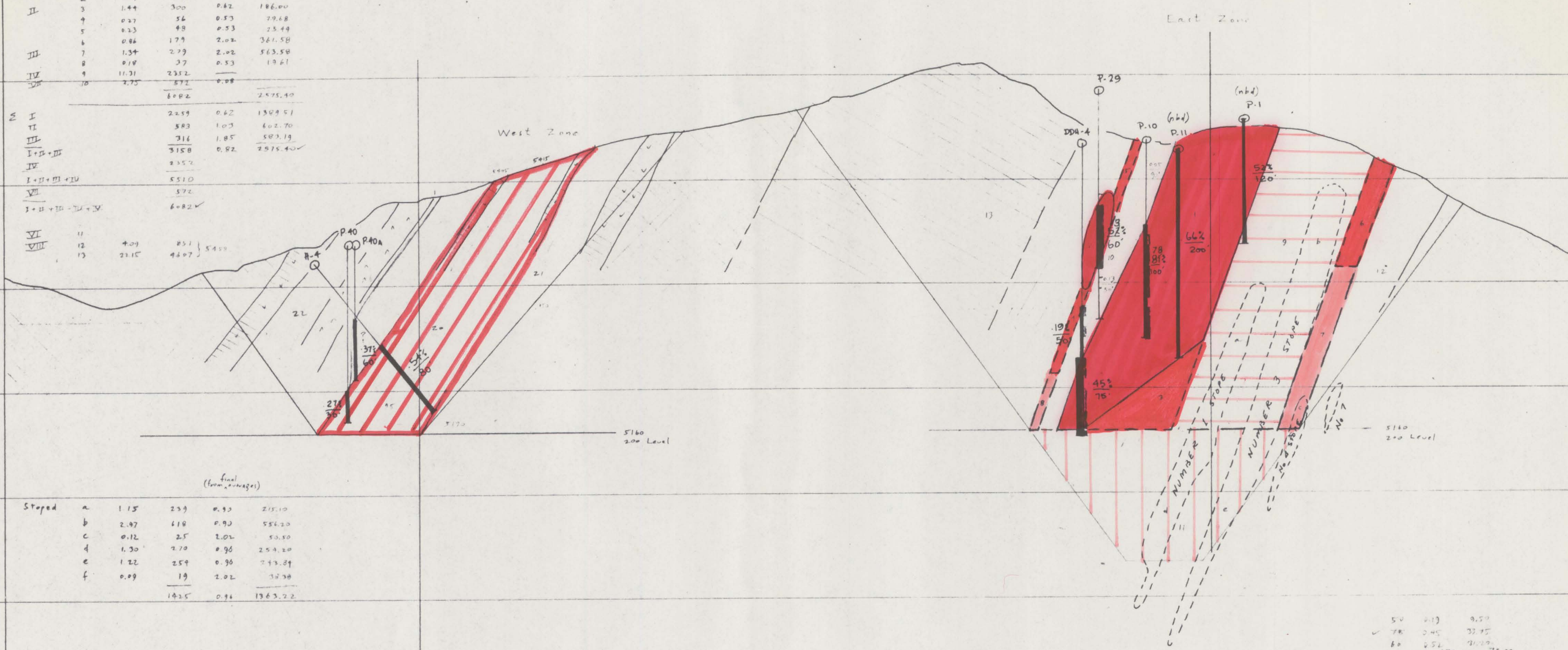
SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair MAY-1974

East Zone

Category	Block	in□	tphf	Cu	Product
I	1	10.27	2136	0.62	1324.32
	2	0.59	123	0.53	65.19
II	3	1.44	300	0.62	186.00
	4	0.27	56	0.53	29.68
	5	0.23	48	0.53	25.44
	6	0.86	179	2.02	361.58
III	7	1.34	279	2.02	563.58
	8	0.18	37	0.53	19.61
IV	9	11.31	2352		
VII	10	2.75	572	0.08	
			6082		2575.40
Σ I			2259	0.62	1389.51
II			583	1.03	602.70
III			316	1.85	583.19
I + II + III			3158	0.82	2575.40
IV			2352		
I + II + III + IV			5510		
VII			572		
I + II + III + IV + V			6082		
VI	11				
VIII	12	4.09	851 } 5458		
	13	22.15	4607		



(final from averages)

Stoped	a	1.15	239	0.90	215.10
	b	2.97	618	0.90	556.20
	c	0.12	25	2.02	50.50
	d	1.30	270	0.96	254.20
	e	1.22	259	0.96	243.39
	f	0.09	19	2.02	38.38
			1425	0.96	1363.22

West Zone

Category	Block	in□	tphf
O	20	9.03	1878
W	21	4.11	855
	22	10.51	2186
			4919
Σ W			3041
O + W			4919

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PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG

NOV 29 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

SECTION K-K'

YAVAPAI COUNTY, ARIZONA

SCALE

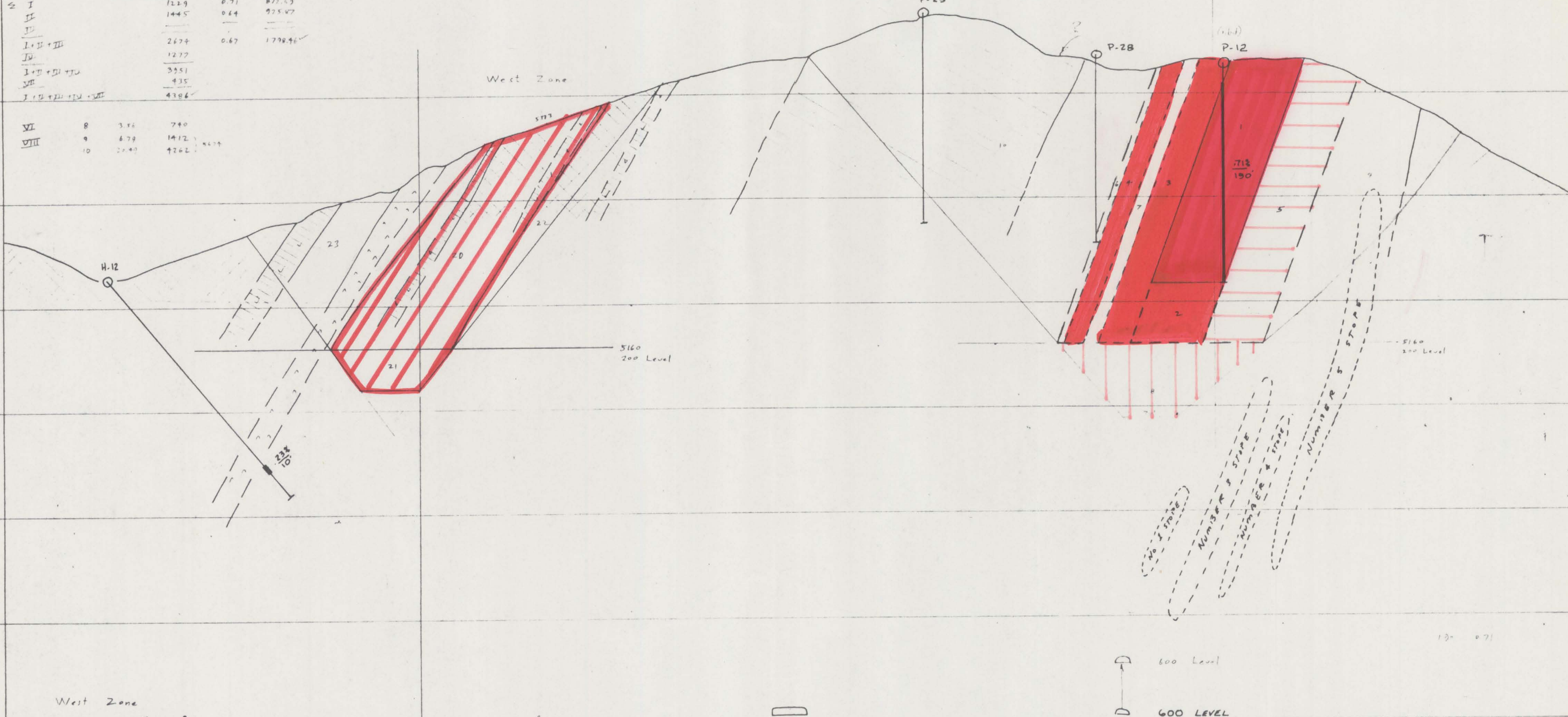
FEET 50 25 0 50 100 150 200 FEET

SEP 17 1974

Altair MAY-1974

East Zone

Category	Block	in°	tphf	Cu_I	Product
I	1	5.91	1229	0.71	872.59
II	2	1.60	333	0.71	236.43
	3	3.30	686	0.62	425.32
	4	2.05	426	0.62	264.12
IV	5	6.74	1277		
VII	6	0.62	129		
	7	1.47	306		
			4386		1798.46
Σ I			1229	0.71	872.59
II			1445	0.64	925.87
III					
I+II+III			2674	0.67	1798.46
IV			1277		
I+II+III+IV			3951		
VII			435		
I+II+III+IV+VII			4386		
VI	8	3.56	740		
VIII	9	6.79	1412	5674	
	10	20.49	4262		



West Zone

Category	Block	in°	tphf
O	20	8.85	1841
	21	1.38	297
W	22	2.08	433
	23	5.80	1206
			3767
Σ O			2128
W			1639
O+W			3767

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C. M. ARMSTRONG

NOV 29 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

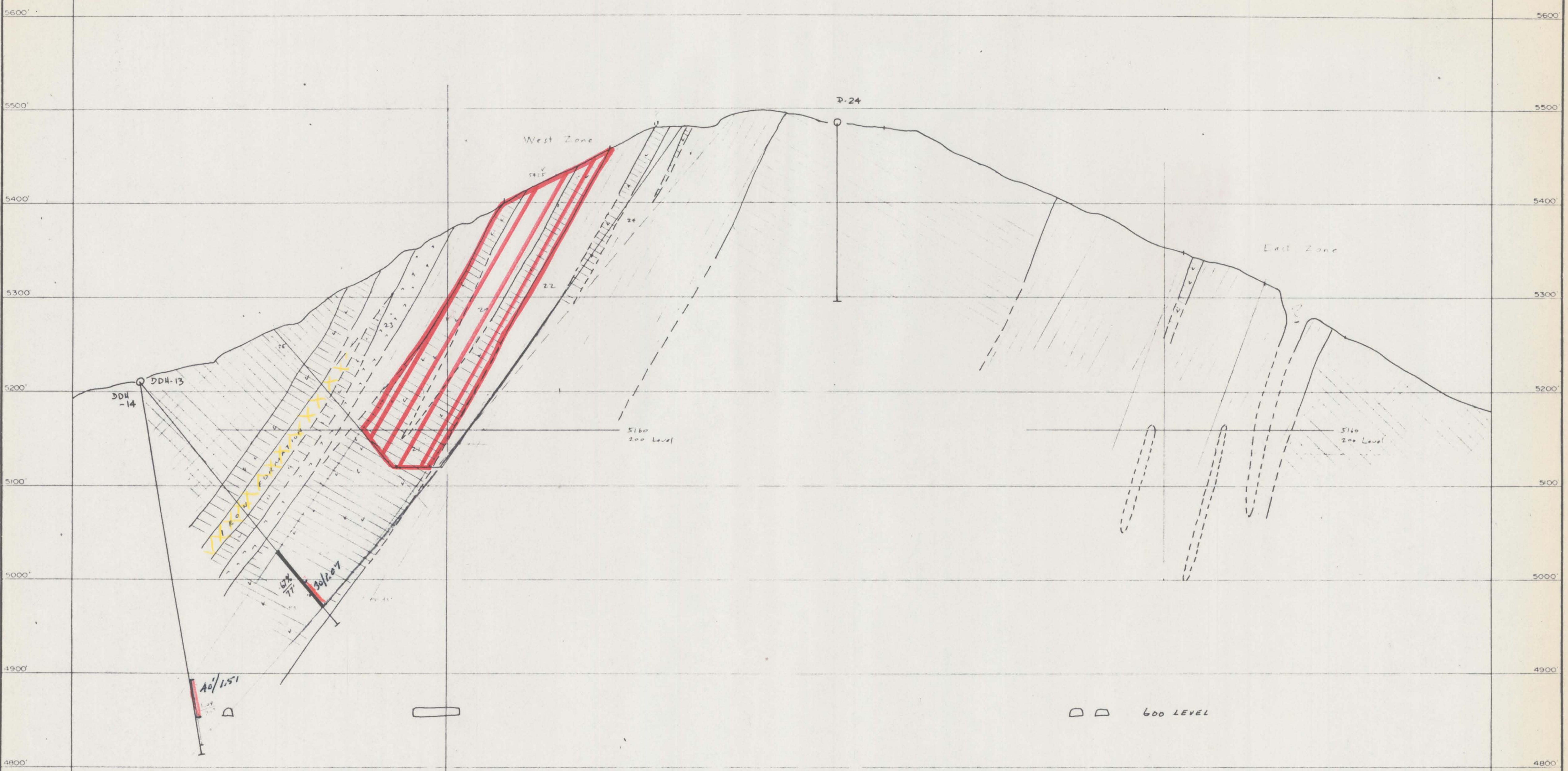
SECTION J-J'

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair MAY-1974



West Zone

Category	Block	in²	tphf
O	20	9.08	1889
	21	1.07	223
W	22	5.17	1075
	23	8.08	1681
			4868 ✓
Σ O			2112
W			2756
O + W			4868

500	24	2.91	605
	25	0.39	81
			686

$\frac{686}{2756} \times 100 = 25\%$

NOV 29 1974

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C. M. ARMSTRONG

SEP 17 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

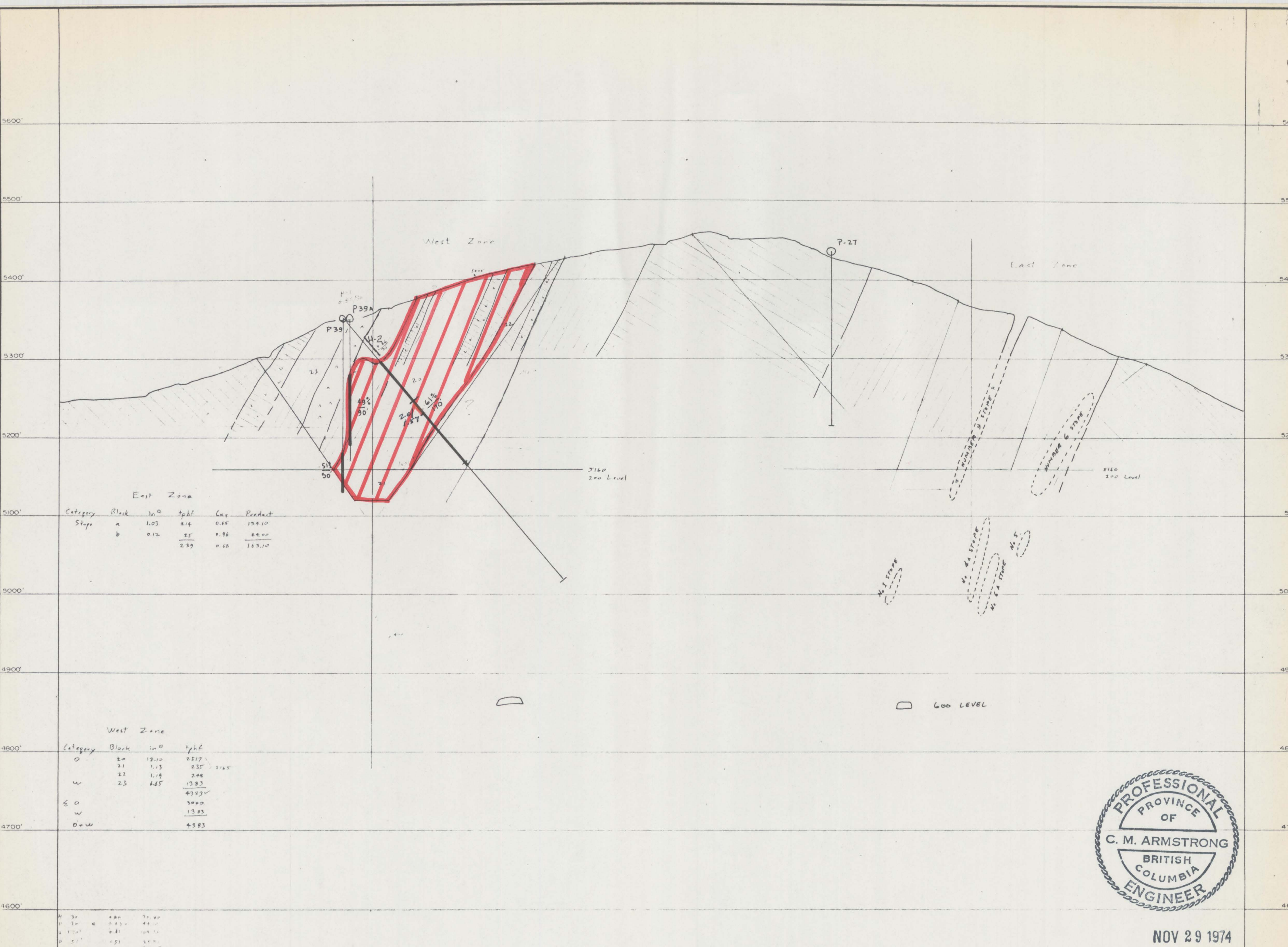
SECTION H-H'

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair MAY 1974



East Zone

Category	Block	in²	tphf	Cu%	Product
Stope	a	1.03	214	0.65	139.10
	b	0.12	25	0.96	24.00
			239	0.68	163.10

West Zone

Category	Block	in²	tphf	
O	20	12.10	2517	
	21	1.13	235	2765
	22	1.19	248	
W	23	6.65	1383	
			4383	
Σ O			3000	
W			1383	
O + W			4383	

H 30		[illegible]	[illegible]
[illegible]		[illegible]	[illegible]
[illegible]		[illegible]	[illegible]
P [illegible]		.51	[illegible]
340		(0.59)	193.10

NOV 29 1974

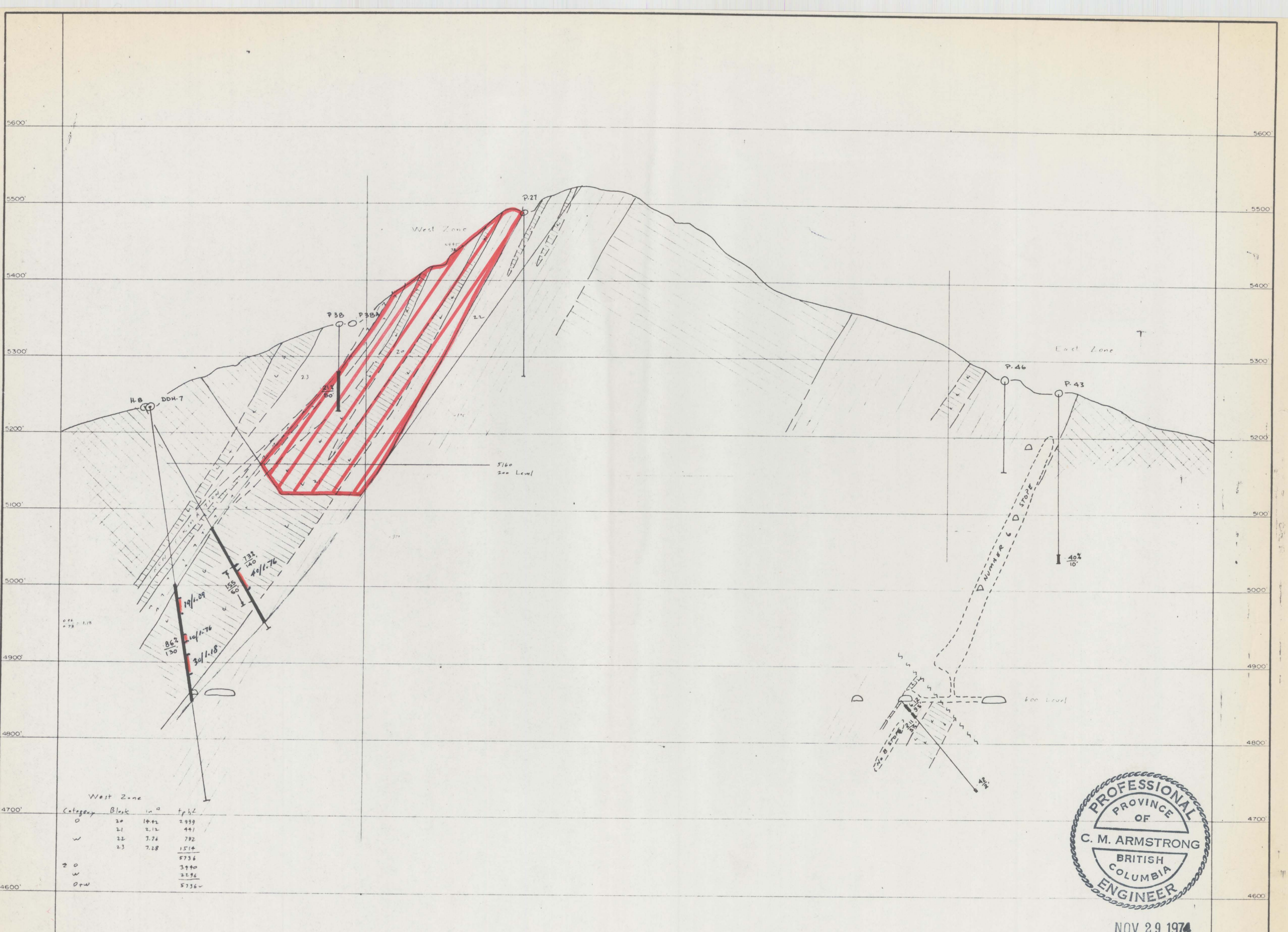
CUTLASS EXPLORATIONS
LTD. (N.P.L.)

SEP 17 1974

DE SOTO MINE
SECTION I-I'
YAVAPAI COUNTY, ARIZONA

SCALE
FEET 50 25 0 50 100 150 200 FEET

Altair MAY-1974



West Zone

Category	Block	in²	tph²
O	20	14.42	2999
	21	2.12	441
W	22	3.76	782
	23	7.28	1514
			5736
Σ O			3440
W			2296
O+W			5736

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

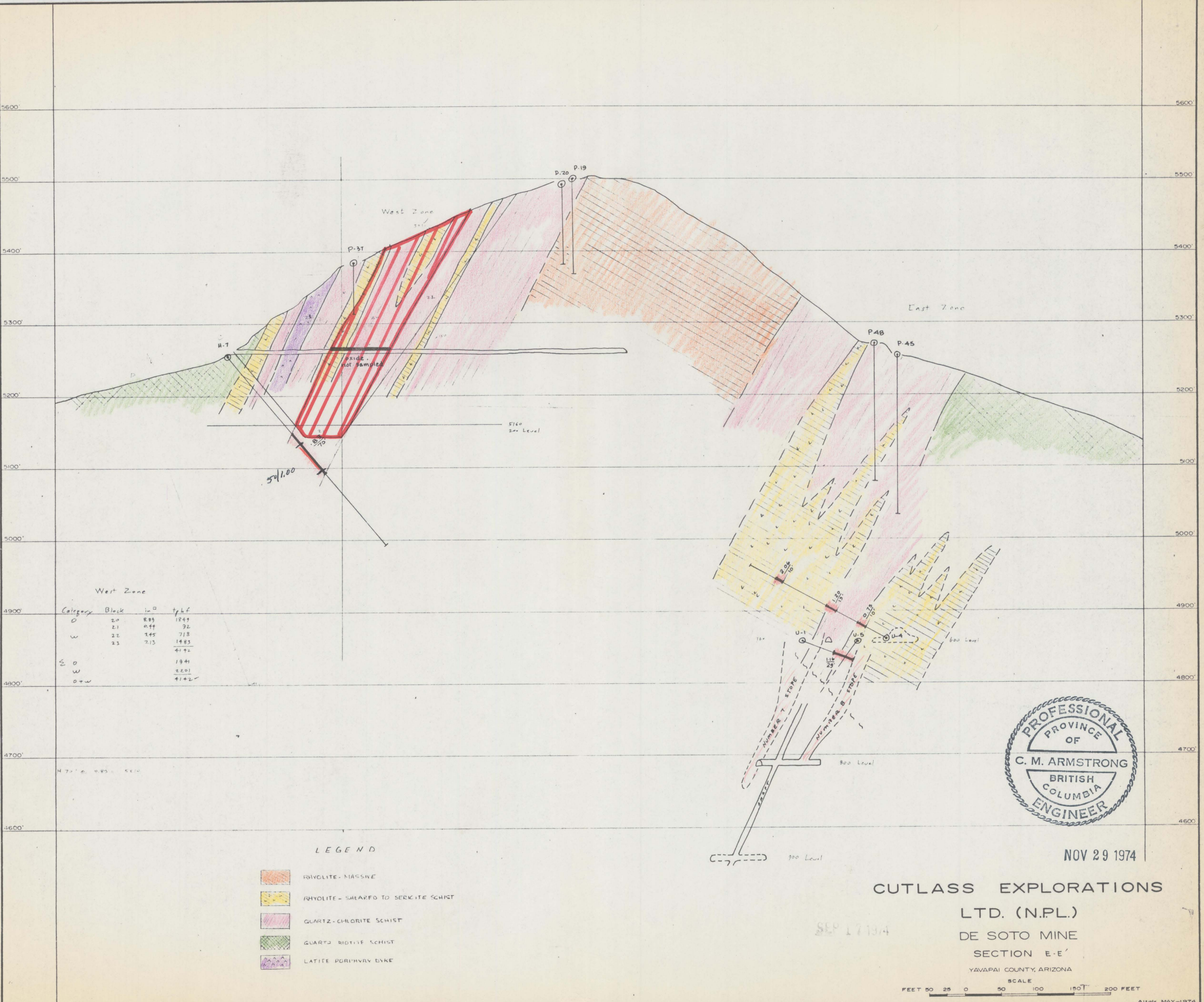
SECTION G-G'

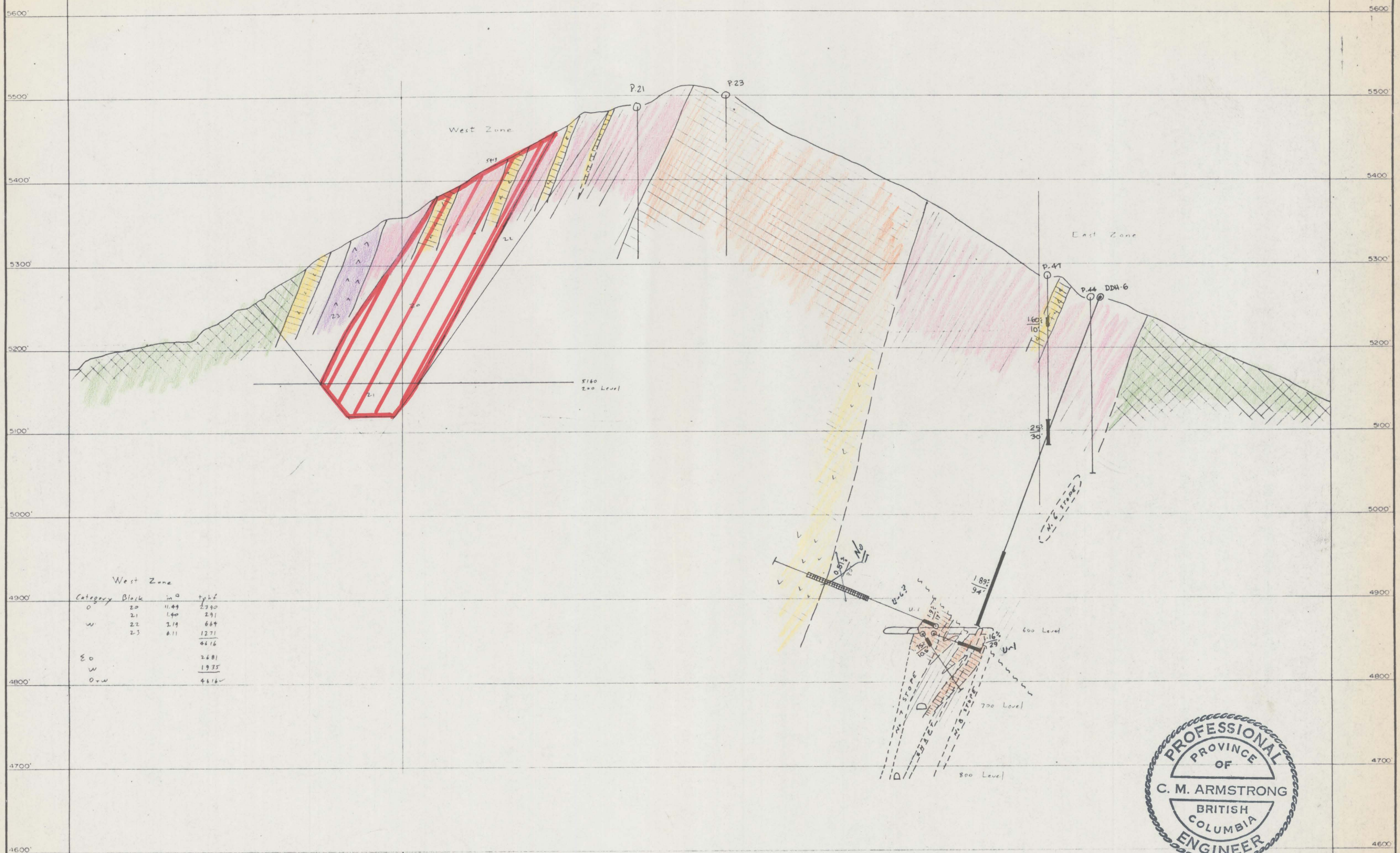
YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair MAY-1974


West Zone
P-37
P-20
P-19
oxide - not sampled
H-7
5160
200 Level
.50/1.00
East Zone
P48
P-45
U-1
U-5
U-4
600 Level
NUMBER 7 STOPE
NUMBER 8 STOPE
800 Level
900 Level
5600'
5500'
5400'
5300'
5200'
5100'
5000'
4900'
4800'
4700'
4600'
West Zone
Category Block in□ tphf
O 20 8.89 1849
21 0.44 92
W 22 3.45 718
23 7.13 1483
4142
Σ O 1941
W 2201
O+W 4142
LEGEND
RHYOLITE - MASSIVE
RHYOLITE - SHEARED TO SERICITE SCHIST
QUARTZ - CHLORITE SCHIST
QUARTZ BIOTITE SCHIST
LATITE PORPHYRY DYKE
PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG
NOV 29 1974
SEP 17 1974
CUTLASS EXPLORATIONS
LTD. (N.P.L.)
DE SOTO MINE
SECTION E-E'
YAVAPAI COUNTY, ARIZONA
SCALE
FEET 50 25 0 50 100 150 200 FEET
Altair MAY-1974



West Zone

Category	Block	in °	tph f
O	20	11.49	2390
	21	1.40	291
W	22	3.19	664
	23	6.11	1271
			4616
Σ O			2681
W			1935
O + W			4616

NOV 29 1974

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

SECTION F-F'

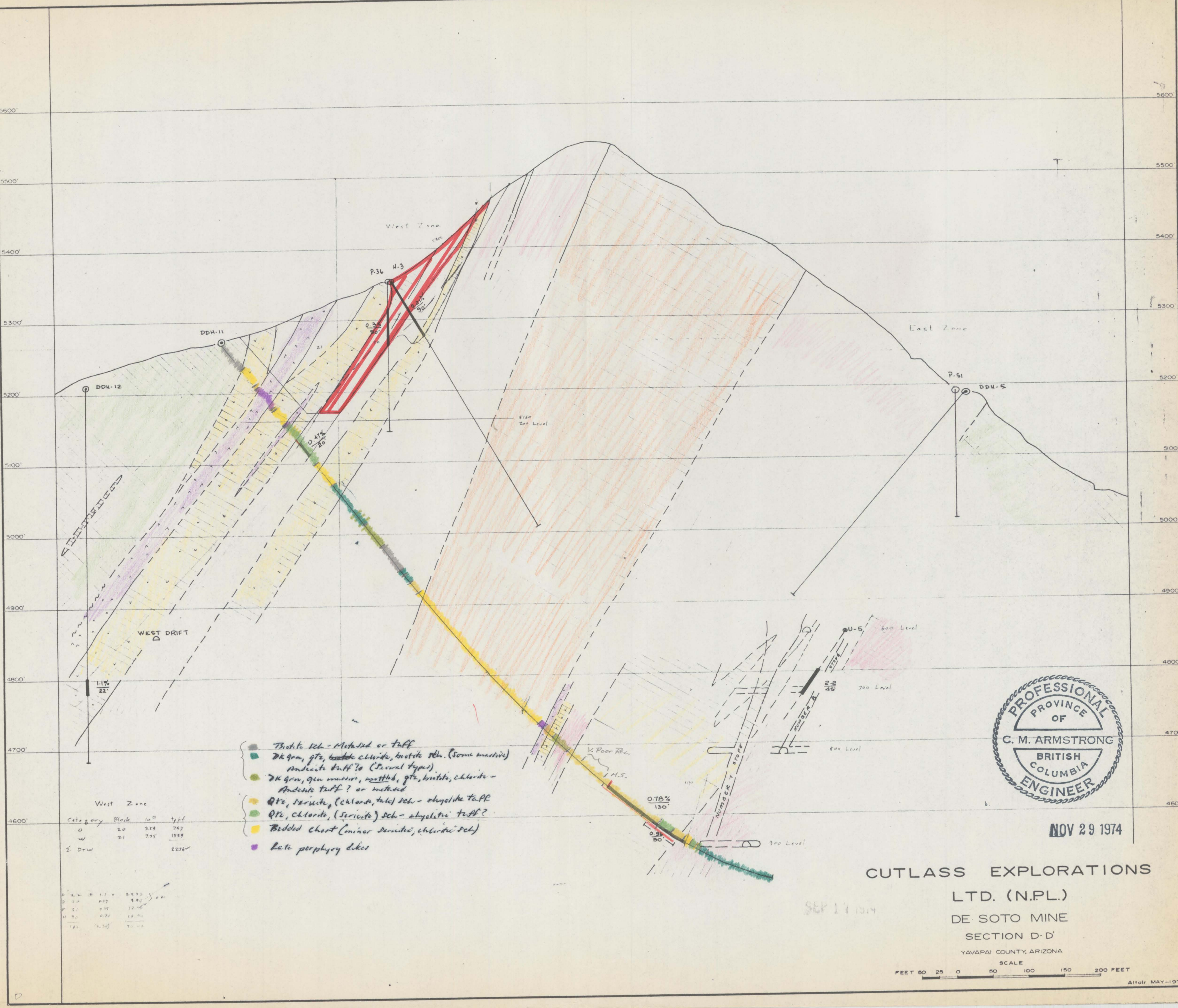
YAVAPAI COUNTY, ARIZONA

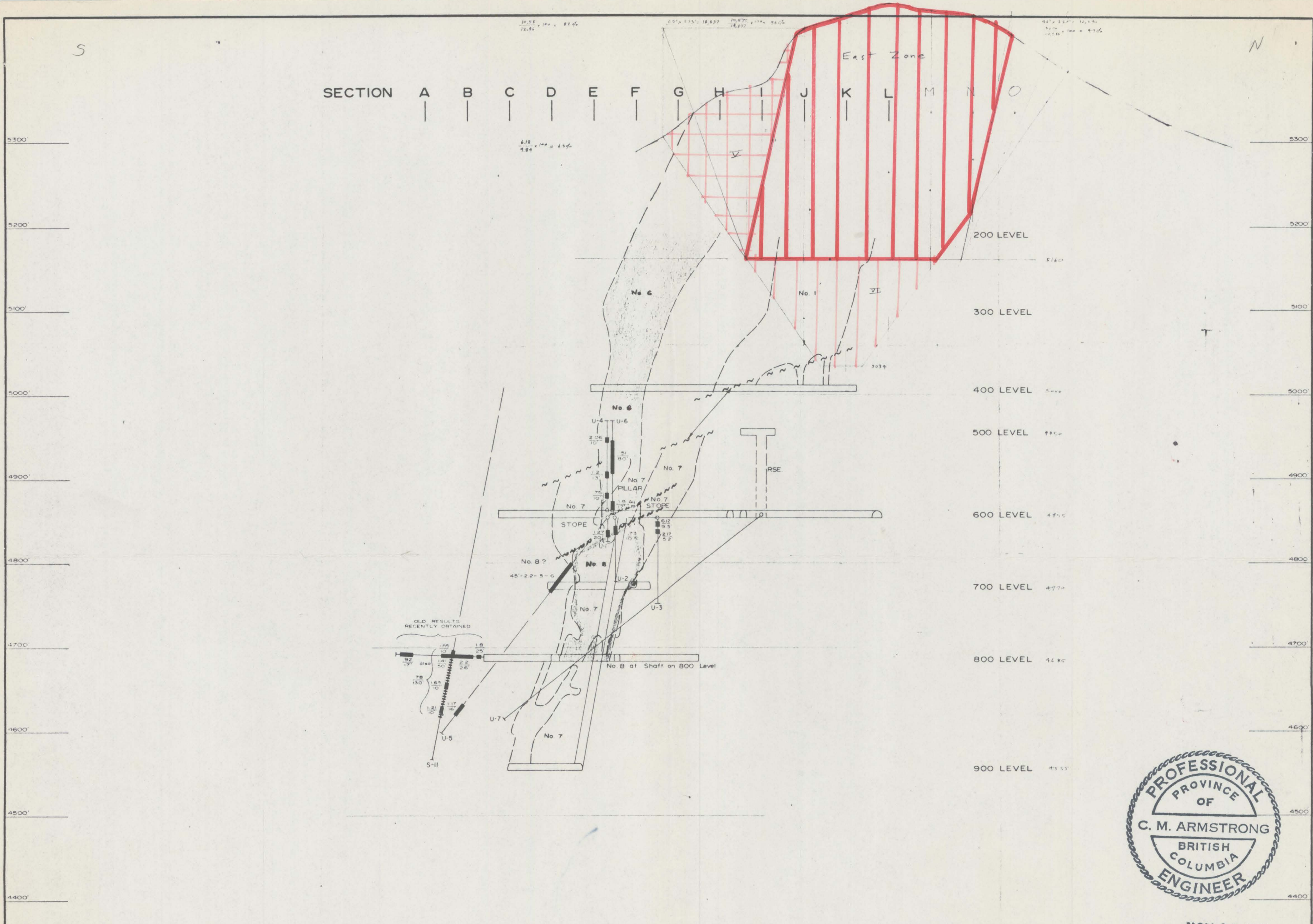
SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair MAY-1974

F


CUTLASS EXPLORATIONS
LTD. (N.P.L.)
DE SOTO MINE
SECTION D-D'
YAVAPAI COUNTY, ARIZONA
SCALE
FEET 50 25 0 50 100 150 200 FEET
Altair MAY-1974
PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
C. M. ARMSTRONG
NOV 29 1974
West Zone
East Zone
DDH-11
DDH-12
DDH-5
P-36
H-3
P-51
U-5
WEST DRIFT
5160 200 Level
600 Level
700 Level
800 Level
900 Level
NUMBER 7 STOPE
NUMBER 8 STOPE
V. Poor Rec.
M.S.
0.41% / 20'
0.33 / 50'
1.1% / 22'
2.8 / 45'
0.78% / 130'
0.96 / 50'
5600'
5500'
5400'
5300'
5200'
5100'
5000'
4900'
4800'
4700'
4600'
Biotite sch - Metased or tuff
Dk grn, qtz, ~~biotite~~ chlorite, biotite sch. (Some massive) Andesite tuff? (Several types)
Dk grn, grn massive, mottled, qtz, biotite, chlorite - Andesite tuff? or metased
Qtz, sericite, (chlorite, talc) sch - rhyolite tuff
Qtz, chlorite, (sericite) sch - rhyolitic tuff?
Bedded chert (minor sericitic, chloritic sch)
Late porphyry dikes
West Zone
Category Block in² tpbf
O 20 3.59 747
W 21 7.35 1529
Σ O+W 2276



NOV 29 1974

LOOKING WEST

CUTLASS EXPLORATIONS LTD. (N.P.L.)

DE SOTO MINE

LONGITUDINAL SECTION East Zone

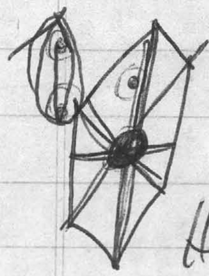
SEP 17 1974

YAVAPAI COUNTY, ARIZONA

SCALE

FEET 50 25 0 50 100 150 200 FEET

Altair April, 1974



Hutchinson 1973
Econ Geol V. 68
1223-1248

Sillitoe 1973
Ec Geol
V. 68 1321-1325

Hutchinson 1972
CIM Bull.
Possible Tectonic & Metallogenic Relationship between Porphyry & Massive

V 65 p 34-40

David Rabb
Solution mining
millerton

- CUPRITE
  No response from Horizon

- MAGNETICS
  In abeyance

- DESOTO
  Jim Knox spent time on property
  yavapai county
  410 420 x ?
  50 x 500 x 600 ft
  50 x 600 x 1000
  down dip

~~15000~~
1500

Sherwood Owens = underlying owner

Current deal too tough - renegotiate

Steve Radvick - Al talking to - Consultant

Patented + unpatented
30-40 claims

Concordant sulfide - volcanogenic
N 30 E

20
70
35
25
10

Swastika, Desoto, Bluebell, Binghamton

Andesite tuffs meta dirty sediments
locally siliceous, cherts

299,000 T 3.4% Cu production
1 oz Ag
.02 AU
minor Sphalerite & Galena

Interest: not in any present reserve.

Probably 2.5x10^6 T 0.9 oxide.
assuming west grade as good as east.

If everything went well = $500,000 profit
for 7 yrs.

Potential:
2-3x10^6 of 3-5% Cu
unknown Zn + Au & Ag.
NOT DEVELOPED TO DATE

Drill at greater depths to verify extension

Shaft operation c̄ water to pump.
Possibly small mill site.

Has ~$300,000 in operation – ~130,000 of admin
in in payments.

Radbaen thinks that owner can be dealt c̄.

Radbaen

Dean

$30,000 down Aug 73
30,000 Dec 73
30,000 every subsequent 4 months

until 200,000 $\bar{c}$ 10% NSR
then
2500/mo @ 10% NSR
to 1 ~~2~~ mil
then
2500/mo or 5% NSR

19 ~~un~~patented, ~~24~~ 4 unpatented
3 mile perimeter

Probably want give 3-4 months slack
under new terms:

1.5 $\times 10^6$ end price
@ 6% NSR
maybe give some continuing

Payment schedule
2,500 per month $\bar{c}$ 4th month = 4,000

May 5,000
June 6,000
July + = 10,000 per month
Then later to 2,000

First 4 months 11,500
Drilling 40,000
Roads 2,000
assays 3,000
Supervision 4,000
Legal 1,000
Geophysics 5,000

In 4 months 66,500

Cullan for carried net profits interest
Recover investment from production

30 million over short period ?

3 holes to 1500 feet

4-6 weeks drilling

2 rigs

Iron King produced 5×10^6 tons
7.34 Zn
2.50 Pb
3.7 oz Ag
0.123 Au
.19 % Cu

2,000 long
×
2500 deep
×
20-25' wide

principal
Cutlass 7½-8% carried
net profit interest
PKK 4% npi

NOTES ON DE SOTO MINE

YAVAPAI CO., ARIZONA

Source: U.S. GS. BUL. 782 pp 162-164.

FROM: LONGITUDINAL SECTION OF ORE CAUSES

STRIKE LENGTH OF CAUSES FROM SOUTH TO NORTH

MEASURED ALONG ~~200~~ 200 LEVEL

	200 LEVEL ~~700~~
over an 1100 ft length	30
→ 36.8 % stopes ~~ore~~ along strike length	90
	35
	90
	80
	80
	405
avg =	67.5

MEASURED ALONG 260 LEVEL

	100
	10
over a 1,100 ft length	90
	~~50~~
→ 40 % stopes along strike length	60
	130
	440
avg =	73.33

MEASURED ALONG 450 LEVEL

	70
over a 1100 ft length	10
	70
	50
→ 26.36 % stopes along strike length	30
	60
	290
avge =	48.33

(2)

DESOTO NOTES

MAIN FAULT ZONE = 500-650 LEVEL

MEASURED ALONG 700 LEVEL

over 1100 ft length

~ 16% stoper

$$\begin{array}{r} 90 \\ 85 \\ \hline 175 \\ \text{avg} \sim 87.5 \\ \hline \end{array}$$

- Number 1 ore body is up to 150 ft wide
  a 600 level (may ~~to~~ be widened by faulting!
  pinches to 20' wide @ 900 feet.

~~Georgia Policy~~

Iron King

Ore: @ contact ~~between~~ of a unit made up of rhyolitic tuff + interbedded andesite and a structurally underlying series consisting of andesitic tuffs, minor rhyolitic tuffs and argillaceous sediments.

Sequence may be overturned

~~a parallel~~

a parallel zone of copper mineralization occurs within the hanging wall rocks.

Ore = ~~0.123 oz~~ Au

Au = 0.123 oz

Ag = 3.69 oz

Pb = 2.50

Zn = 7.34

Cu = 0.19

Keratophyric:

Salic (light-colored - silica or magnesium rich i.e. qtz, feldspar, feldspathoids) characterized by albite or albite/oligoclase + chlorite, epidote + calcite.

Some c ~~...~~ sodic orthoclase + sodic amphibole + pyroxene

SWASTIKA MINE

A mile east of the Peck along the road to Peck siding are the Black Warrior and Silver Prince veins, now owned by Frank W. Giroux, of Mayer, under the name of the Swastika Silver & Copper Co.

The Silver Prince is mentioned in Raymond's report of 1877 with the statement that the cost of packing the ore to Prescott was $50 a ton. The Mint report for 1883 mentions both veins, stating that the Black Warrior was 2 to 3 feet wide, that $40,000 in silver had been extracted so far, and that 8 tons a day was milled in a 4-stamp mill for a yield of 113 ounces of silver to the ton. About 1885 the mine was considered exhausted, and it was idle until reopened by F. W. Woods in 1910. From 1910 to 1915 the mine produced 600,000 ounces of silver. The total production is stated to be about 1,000,000 ounces. Since 1915 the mine has been in intermittent operation. Mr. Woods states that from 1875 to 1908 the Silver Prince had yielded $480,000 and the Black Warrior $385,000. The later production came wholly from the Silver Prince.

The country rock consists of Yavapai schist, mostly fissile and sericitic, with lenses of quartzite, but the outcrops are deeply oxidized. The two parallel veins strike due north and dip 60° W., with the schist. The Prince lies 300 feet west of the Black Warrior. Between the two there is a 50-foot dike of light-colored porphyry.

The Silver Prince is developed by tunnels and a 400-foot shaft about 600 feet to the north. The vein is at most a few feet wide and carries dark-brown limonitic ore. There is a little quartz, but the principal gangue mineral is a sideritic carbonate, with native silver, chloride, and some sulphides. The sulphides consist of a partly decomposed tetrahedrite rich in silver and a little chalcopyrite. The ore, which contains a little lead, was sold to El Paso and the lead smelter at Needles in 1914 and later shipped to Salt Lake City.

The shaft on the Black Warrior is said to be only 125 feet deep. The lowest levels were not visited, but it is evident that the ore on them is poorer. Here, too, the conditions are similar to those at the Peck, namely, an extraordinary concentration in the oxidized zone and impoverishment below. Considering the history of this mine, it would be rash to say that it is exhausted. More comments on the concentration in the oxidized zone of these deposits are found on page 49.

DE SOTO MINE

The outcrops of the De Soto copper mine lie 2 miles northeast of the Peck mine, on the summit of the high ridge separating Peck Canyon from Crazy Basin. The altitude is about 5,800 feet. The main tunnel is 600 feet below the outcrop, and from it an incline

leads down to Middleton station on the Crown King branch road. The property is owned by the same interests that control the Humboldt smelter (Southwest Metals Co.), to which the ore has been shipped. Work was discontinued in 1922, the ore bodies being considered exhausted. Most of the information given below was obtained from Mr. J. L. White, of the staff of the smelter.

The Yavapai chloritic schists strike N. 23° E. at the mine and dip 70° NW. The ore bodies, which carry pyrite-chalcopyrite ore and are contained in a chloritic schist, have yielded a total of 180,000 tons, averaging about 3.75 per cent of copper with 1 ounce of silver and 0.02 ounce of gold to the ton. The last ore treated contained 2.25 per cent of copper. There is less pyrite than at the Blue Bell

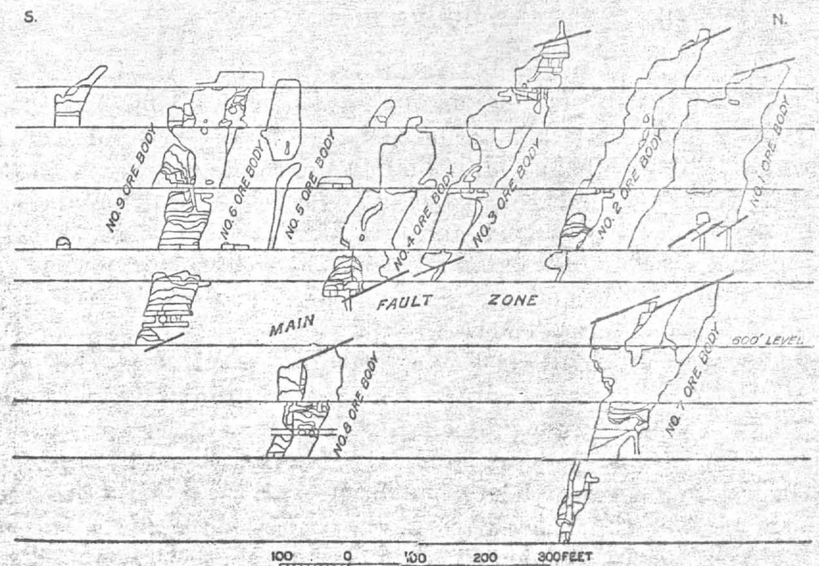


FIGURE 10.—Longitudinal section of ore lenses in the De Soto mine

mine, some sphalerite and galena, and occasional specimens of tetrahedrite. (See pl. 17, *A*.) A few prisms of arsenopyrite were observed.

The gangue is fine-grained quartz. There are also lenses of coarser quartz, much of it crushed and showing undulous extinction. Gangue and sulphides replace the schist, which is mainly chloritic with a little biotite.

The ore occurs in overlapping lenses. (See fig. 10.) On the upper levels there were seven such lenses close together, with an individual width of as much as 50 feet and a greatest length of 250 feet. Exploration extended to a depth of 300 feet below the main tunnel level, and at this depth only one small lens persisted. The total length of the ore zone is 350 feet; the total width 200 feet.

The ore bodies are said to have been cut off in depth by a flat fault 250 feet below the surface. The small bodies found below this depth are believed to represent the continuation of the ore below the fault. The ore bodies are shown in Figure 10, each separately, in a longitudinal projection, for they overlap so that they can not be indicated in their correct relative position.

OTHER PROPERTIES

The veins of the Gold King group, in the southern part of the district, are said to be the extension of the Gladiator vein, which is in the Pine Grove district. Near by is the Blue Bird vein. Both these deposits are in Yavapai schist.

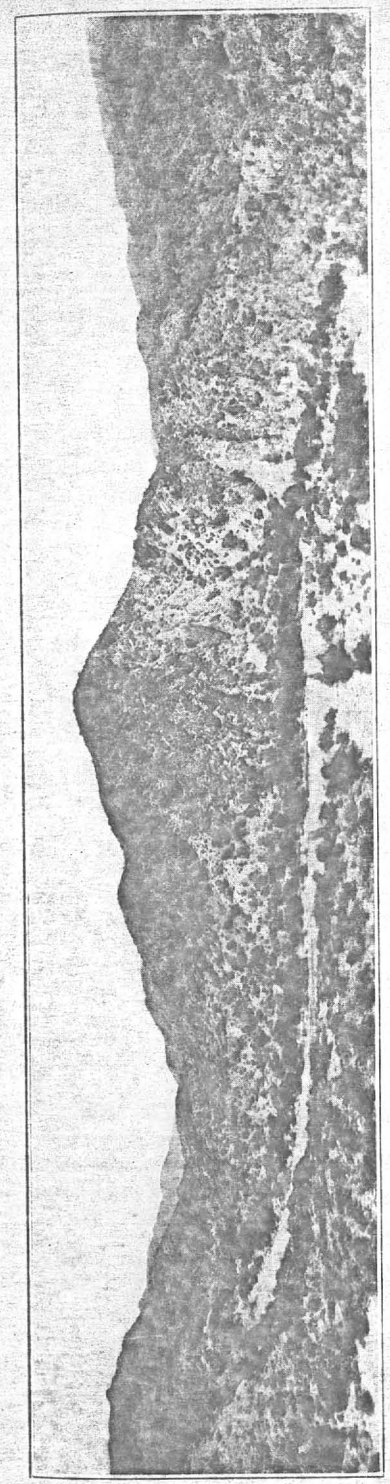
PINE GROVE DISTRICT

The Pine Grove district lies in the heart of the Bradshaw Mountains, in a well-timbered region, at an altitude of 6,000 to 7,500 feet. (See pl. 21, *B*.) Its highest point is the diorite mass of Towers Mountain. Most of the claims lie in a basin-like depression on the east side of the ridge. The district is reached by an automobile road from Prescott, 40 miles long, and by a branch railroad from Mayer, which ascends Crazy Basin and Poland Creek in a series of switchbacks. It is an old mining region, and many of the veins were very rich near the surface. The earliest properties worked were the Del Pasco, Gladiator, and War Eagle. The ores carry silver and gold.

Most of the mines are situated in granodiorite (quartz diorite, according to Jaggar and Palache), which forms a rounded mass 3 to 4 miles in diameter, intruded into Bradshaw granite and still earlier Yavapai schist. The granodiorite is cut by a series of dikes which trend north-northeast across the center of the area. In part these dikes are rhyolite porphyry, in part granite porphyry. There are also some light-colored granitic dikes which seem to be affiliated with the granodiorite; the others just mentioned appear to represent a distinctly later intrusion.

There are three prominent vein systems, which trend north-northeast and generally dip about 60° WSW. The shoots have a tendency to pitch northward. They occur mostly in the granodiorite but continue also to the north in Yavapai schist, diorite, and mixed areas (Wildflower mine), though these harder rocks are as a rule less favorable. Few of the veins are more than 5 feet in width, and they contain a filling of predominant quartz, with some ankerite and calcite.

Much of the quartz is drusy and contains more or less pyrite, chalcopyrite, zinc blende, and galena, with some tetrahedrite. In places free gold occurs in the primary ore. Most of the ore extracted



A. VIEW LOOKING UP BLACK CANYON

Mine of Howard Copper Co. in the distance

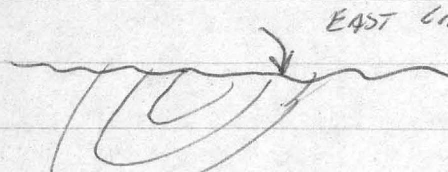
B. VIEW LOOKING NORTHEAST FROM CROWN KING STATION TOWARD CRAZY BASIN

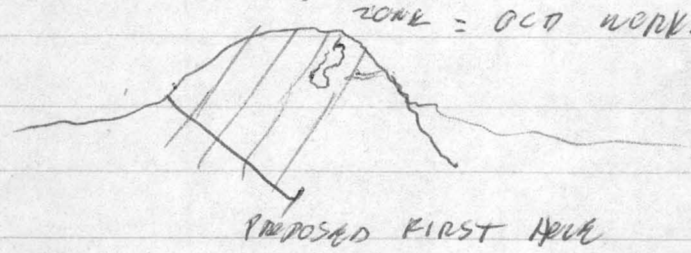
DESOTO MINE

• LOCATION: BRADSHAW MTS. - CENTRAL ARIZONA
  .. BLUEBELL = 4 MILES NORTHEAST ON TREND
  1½ MILLION TONS OF ~3.5% Cu TO 1500 ft
  .. IRON KING 16 MILES TO NORTH
  5 MILLION TONS of 13-16% Pb/Zn TO 3,000 ft

• SETTING:
  .. PC VOLCANOGENIC STRATABOUND MASSIVE SULPHIDE IN YAVAPAI SCHIST

  .. TWO PARALLEL ZONES OF COPPER MINERALIZATION IN ALTERED SILICEOUS, TUFFACEOUS, RHYOLITIC SCHISTS HAVE BEEN DEFINED BY SURFACE + SUBSURFACE EXPLORATION OVER STRIKE LENGTHS UP TO 1500 FEET, (WEST ZONE) AND TO DEPTHS OF 6-900 ft. SUBSTANTIAL WIDTHS OF 0.5-1% HAVE BEEN SHOWN TO 200 feet (ON EAST LIMB





NO DEEP DRILLING - DRILLING ALL SHALLOW ON EAST OXIDE ZONE

.. PRODUCTION:
  290,000 TONS OF 3.3% Cu

(2)

B:

- PROPOSAL:
  - GEOLOGIC MAPPING
  - TURAM E.M. SURVEY
  - 3 DIAMOND DRILL HOLES TO TEST BOTH ZONES BELOW OLD WORKINGS AND POSSIBLY TO NORTH ALONG STRIKE.

- ESTIMATED COST
  - 4 MONTH PROGRAM
    2 MONTH EVALUATION
    6 MONTHS

DIAMOND DRILLING	40,000
ROADS + SITES	2,000
ASSAYS	3,000
TURAM EM	3,000
LEGAL	1,000
SUPERVISION	4,000
TOTAL	$53,000

- ASKED FOR TERMS: OWNERS

2 500/mo FOR 6 MONTHS = $15,000

AT END OF 6 MONTHS 30,000

2,000/MONTH THEREAFTER
OR 5% NSR TO 1,500,000

2% NSR THEREAFTER

CUTLASS 7½-10% NET PROFITS WITH $350,000 TO ~~COME OUT~~ BE RECOVERED ON A PROPORTIONAL BASIS

(3)

.05 g Au
1.00 g Ag

BASED ON 2MM TONS @ 3.5% Cu.

- POTENTIAL
  - LOW CAPITAL ~ $8,900,000
  - RAPID PAYOUT ~ 2 year
  - NET PROFIT / YR ~ $1,206,000
  - ANNUAL CASH FLOW ~ 4,596,000
  - ACCOUNTING RATE OF RETURN (UNDISCOUNTED) 13.5 %

  - LIFE @ 1,000 TPD = 5.7 yrs

Year		Capital	Interest on Capital	Capital Plus Interest	Cash Flow	Discounted ~~Net Cash~~ Cash Flow	Net Discounted Cash Flow	Cumulative DCF	% DCFROI
Explor + Dev. Feasibility	1	(700,000)	(70,000)	(-770,000)	(-700,000)	(-700,000)	(-770,000	(-700,000)	-0.0954 -0.0965
Shaft Mill Infra	2	(6,300,000)	707,000 (7,070,000)	(-7,777,000)	(-5,206,612)	(-5,206,612)	(-7,777,000	(-5,906,612)	-.7097 -0.8735
Working Capital	3	(1,903,000)	968,000 (9,600,000)	(-10,648,000)	4,682,378	2,088,188 3,517,940	(-7,130,060)	(-3,818,424) (-7,130,060)	.2846 -0.8009
	4		713,006 (7,130,060)	(-7,843,066)	4,682,378	3,198,127 3,198,127	(-4,644,939)	(-620,297) PAYOUT	.1359 -0.5217
	5		464,494 (4,644,939)	(-5,109,433)	4,682,378	2,907,388	(-2,202,045)	2,287,092	.3963 -0.2173
	6		220,204 (2,202,045)	(-2,422,249)	4,682,378	2,643,080	220,831	4,930,172	.3603 +0.0248
	7		PAID OUT	PAID OUT	4,682,378	2,402,800	2,402,800	7,332,972	.3275 +0.2699
	8				3,277,665	1,529,055	1,529,055	8,862,027 3,432,055	+0.4678 +0.3855 +0.1717
	9	RETURN WORKING CAPITAL					1,903,000	1,903,000	Average DCFROI = -23.12 %
									average = 0.1834
					26,689,555	16,198,390	6,055,686	10,765,027	

Total Capital
8,903,000

NET PRESENT VALUE = 6,055,686

NET PROFIT FOR 9 YEAR = 7,295,390

PV Capital = 7,336,364

18.34% DCFROI

350 D/Y OPERATIONS STANDARD PROPERTY EVALUATION FORM 2 MM TONS @ 3.5% Cu

Reserves 2 X 10^6

Milling Rate 1000 TPD 350,000 TPA 3.25% NSR

Property Life 5.7 10% NET PROFITS

Capital Investment 8,903,000 Payments Royalty

Recoverable Value $36/T

Average Grade

Mining Method Cost/T

Milling Method Cost/T

Administrative Cost/T Total Direct Cost/T 13.50

1. Annual Gross Sales 12,600,000

Less Post Milling Costs		
Smelting		
Smelting Depreciation		
Concentrate Handling		
Selling		
Royalties avg = 3.25% NSR	409,000	
2. Depletion Base		12,190,500
Less Cost		
Direct Cost	4,725,000	
Depreciation-Amort.	1,561,930	
Local Taxes	630,000	6,916,930
3. Operating Income		5,273,570
Depletion 15 % of 2 .50	1,828,575	
Investment Credits		1,828,575
4. Taxable Income	2½% inventory of mine	3,444,995
State Tax	~~5%~~ 2½% NET PROCEEDS net profit	172,250 ~~3,272,745~~
Federal Tax	50%	1,722,500
5. Profit		1,205,745 *
Add:		
Smelter Depreciation		
Depreciation-Amort.	1,561,930	
Depletion	1,828,575	3,390,505
6. Annual Cash Flow	6th year (844,021)	1,205,745

Payout Time 2 yrs Years Life 5.7

Cash Flow Rate of Return (Discounted)

Net Present Value

Accounting Rate of Return (Undiscounted)

ACCOUNTING ROI = 13.5% UNDISCOUNTED

STANDARD PROPERTY EVALUATION FORM

Reserves ____________
Milling Rate ____________ TPD ____________ TPA
Property Life ____________
Capital Investment ____________ Payments ____________ Royalty ____________
Recoverable Value ____________
Average Grade ____________
Mining Method ____________ Cost/T ____________
Milling Method ____________ Cost/T ____________
Administrative Cost/T ____________ Total Direct Cost/T ____________

1. Annual Gross Sales ____________

   Less Post Milling Costs
   Smelting ____________
   Smelting Depreciation ____________
   Concentrate Handling ____________
   Selling ____________
   Royalties ____________
   ____________ ____________ ____________

2. Depletion Base ____________

   Less Cost
   Direct Cost ____________
   Depreciation-Amort. ____________
   Local Taxes ____________
   ____________ ____________ ____________

3. Operating Income ____________
   Depletion ____% of 2 ____________
   .50 ____________
   Investment Credits ____________
   ____________ ____________ 1,828,575

4. Taxable Income — TI = 3,444,995
   172,250
   less 10% NTI Royalty 3,272,745
   State Tax 172,250
   ~~1,636,573~~
   Federal Tax 1,636,373 — 1,636,373
   344,500

5. Profit — NET ATP → ~~1,291,873~~
   Add:
   Smelter Depreciation ____________
   Depreciation-Amort. 1,561,930
   Depletion 1,828,575 — 3,390,505

6. Annual Cash Flow — 6th year (3,277,665) — 4,682,378
   Payout Time 1.9 year Years Life 5.7
   Cash Flow Rate of Return (Discounted) ____________
   Net Present Value ____________
   Accounting Rate of Return (Undiscounted) ____________

STANDARD PROPERTY EVALUATION FORM

Reserves ____________
Milling Rate ____________ TPD 350,000 TPA
Property Life ____________
Capital Investment ____________ Payments ____________ Royalty ____________
Recoverable Value $36/T
Average Grade ____________
Mining Method ____________ Cost/T ____________
Milling Method ____________ Cost/T ____________
Administrative Cost/T ____________ Total Direct Cost/T 13.50

1. Annual Gross Sales 12,600,008

Less Post Milling Costs
Smelting ____________
Smelting Depreciation ____________
Concentrate Handling ____________
Selling ____________
Royalties 409,000

2. Depletion Base 12,190,500

Less Cost
Direct Cost 4,725,000
Depreciation-Amort. 1,561,930
Local Taxes 630,000
6,273,570

3. Operating Income 5,273,570
Depletion 15 % of 2 1,828,575
.50
Investment Credits ____________
1,828,575

4. Taxable Income 3,444,995
State Tax 172,250 172,250
~~3,272,745~~
Federal Tax 1,636,373 - 1,636,373
~~1,636,372~~

5. Profit
Add:
Smelter Depreciation ____________
Depreciation-Amort. ____________
Depletion ____________

6. Annual Cash Flow
Payout Time ____________ Years Life ____________
Cash Flow Rate of Return (Discounted)
Net Present Value
Accounting Rate of Return (Undiscounted)

	Year	Capital	Gross Revenue	Operating Costs includ depreciation + amortization ?	Undivided Income before taxes + royalty	Overriding royalty 5% NSR to 1.5×10⁶ then 2% NSR	Cash Flow	Discounted Cash Flow	Cumulative DCF
Exploration + feasibility Development	1	(700,000)						(-770,000	(-770,000)
Shaft, mill, surface	2	(6,300,000)						(-7,007,000)	(-7,777,000)
working capital	3	(1,903,000)	12,600,000	5,950,000	6,650,000	630,000	4,682,378	1,385,707	(-6,391,293)
	4		12,600,000	5,950,000	6,650,000	630,000	4,682,378	3,617,578 ~~4,256,707~~	(-2,773,715
	5		12,600,000	5,950,000	6,650,000	396,000	4,682,378	3,979,335	(1,205,620
	6		12,600,000	5,950,000	6,650,000	252,000	4,682,378	4,256,707	5,462,327
	7		12,600,000	5,950,000	6,650,000	252,000	4,682,378	4,256,707	9,719,034
	8		9,000,000	4,250,000	4,750,000	180,000	3,277,665	2,979,695	12,698,730
Recover Working Capital	9						1,903,000		14,~~600~~,
		8,903,000	72,000,000			2,340,000		NET PRESENT VALUE =	14,601,730

6,375,639.00
1,903,000.00

$$\frac{8,903,000}{14,601,730} = 164\%\ \text{DCFROI}$$

DISCOUNTED NET PRESENT VALUE = 8,278,639

$$\text{DCFROI} = \frac{8,278,639}{8,903,000}\ 0.93$$

De Soto Mine

Capital Requirements

NOTE: CUTLASS ALREADY HAS $350,000 INVESTED.

• 1,000 foot, 2 compartment shaft + related facilities	1,300,000
• preproduction expense (exploration, etc.)	700,000
• mining equipment, miscellaneous surface installations	1,000,000
• 1,000 TPD flotation mill	4,000,000
• Working capital	

(2)

De Soto Mine

Capital Requirements

<u>Working capital</u>

- Mining costs:

  ~~3~~ 2 months × 30 days/mo × 1000 T/D × ~~$17/T~~ $13.50/T = ~~1,530,000~~ 810,000

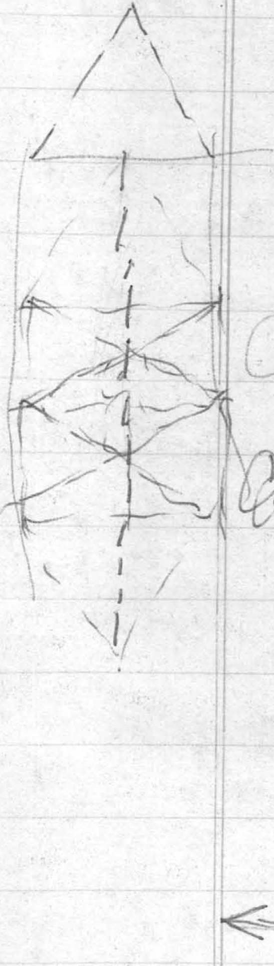
  COST = $17/T − 3.50 depreciation + depletion = $13.50

- 6 MONTHS TAXES 936,000

  1,746,000

- Interest on working capital @ 9% 157,140

TOTAL WORKING CAPITAL 1,903,000



[illegible] Net Proceeds of Mines tax

Corporate income tax

Bullion tax

Net Off Road Vehicle

Corp

State Tax Commission: 882-5404

415 N. Carson

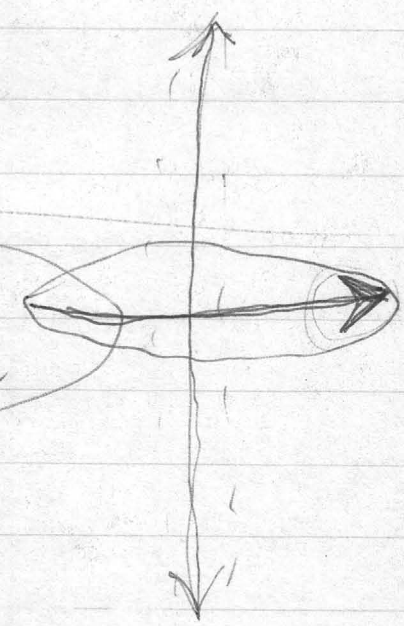
Gross Revenue

$1{,}000\ T/D \times 350\ D/Y \times \$36^{00}/T = \$12{,}600{,}000/Y$

Severance tax — use 5% — maybe — 4% — 6%

State Corporate income tax

Reserve



Discriminated suppliers

350 D/4 OPERATION

STANDARD PROPERTY EVALUATION FORM

Reserves 2×10^6

Milling Rate 1000 TPD 350,000 TPA avg = .0325 { 5% NSR to 1.5x10^6 / 2% NSR }

Property Life 5.7 yrs

Capital Investment 8,903,000 Payments ______ Royalty 10% NET PROFITS

Recoverable Value $36/T

Average Grade ______

Mining Method ______ Cost/T ______

Milling Method ______ Cost/T ______

Administrative Cost/T ______ Total Direct Cost/T 13.50

1. Annual Gross Sales 12,600,000

Less Post Milling Costs

Smelting		
Smelting Depreciation		
Concentrate Handling		
Selling		
Royalties avg .0325 NSR	409,500	

2. Depletion Base 12,190,500

Less Cost

Direct Cost	4,725,000	
Depreciation-Amort.	1,561,930	
Local Taxes	630,000	
		6,916,930

3. Operating Income 5,273,570

15% DEPLETION ON Cu

Depletion ~~.22~~ of 2 .50 .15 [illegible]	1,828,575	
Investment Credits		
		1,828,575

4. Taxable Income 3,444,995

? State Tax	? 5%	172,250
less 10% net profits to [illegible]		344,500
Federal Tax	50%	1,722,500

5. Profit 1,205,745

Add:

Smelter Depreciation		
Depreciation-Amort.	1,561,930	
Depletion	1,828,575	3,390,505

6. Annual Cash Flow 4,596,250

Payout Time 2 yrs. Years Life 5.7

Cash Flow Rate of Return (Discounted) ______

Hoskold Rate of Return ______

Present Worth (15 risk 2% safe) ______

Accounting rate of return = 13.5% undiscounted

(26) Anderson, C.A. + Creasey, S.C. (1958)
Geology + Ore Deposits of the Jerome area, Yavapai Co., Ariz.
USGS Prof. Paper 308

(2)

Iron King

Spilitic:
An altered basalt, characteristically amygdaloidal or vesicular in which feldspar has been albatized + ~~then~~ usually accompanied by chlorite, calcite, epidote & chalcedony + prehnite or other low temp hydrous crystalization products characteristic of greenstone. Often as submarine flows c pillow structure.

Spud Mtn. Tuff
chlorite - sericite schist
qtz - sericite schist

Ore Horiz

{ ankerite: a white, red or grayish iron-rich mineral related to dolomite: $Ca(Fe, Mg, Mn)(CO_3)_2$. Related to iron ores.

DE SOTO PROPERTY

PAST PRODUCTION = 290,000 TONS of 3.3% Cu
c̄ Au, Ag + Zn
0.053/t 1.2/3/t 2.1% Zn

MINERALIZED ZONES = 1,500 feet on strike + 6-900 ft dow

0.5 - 1.0% Cu encountered in testing to date

TARGET = 2-3 × 10^6 TONS

PROPOSED WORK:

- GEOLOGIC MAPPING
- TURAM EM SURVEY
- ~3,000 ft of diamond drilling
- " ~ 2 MONTHS

COST ESTIMATE = $53,000 + PROPERTY PAYMENTS.

GENERAL - location in central Arizona - Bradshaw Mtns
(Prescott National Forest.

REFERENCE:

U.S.G.S. BUL 782
" " 1345
" MAP GQ 996
" " " 997
" AIR MAG MAP GP 758

GEOLOGICAL SURVEY OF CANADA PAPER 72-22
SANGSTER

DE SOTO

GEOLOGICAL SETTING -

EAST FLANK OF WEST DIPPING
OVERTURNED SYNCLINE

ROCK TYPES

PRECURSOR ROCK

BIOTITIC, CHLORITIC, FELDSPAR SCHISTS ← ANDESITIC TUFFS + GRAYWACKES

QTZ-SERICITE
QTZ
CHLORITE
SERICITE

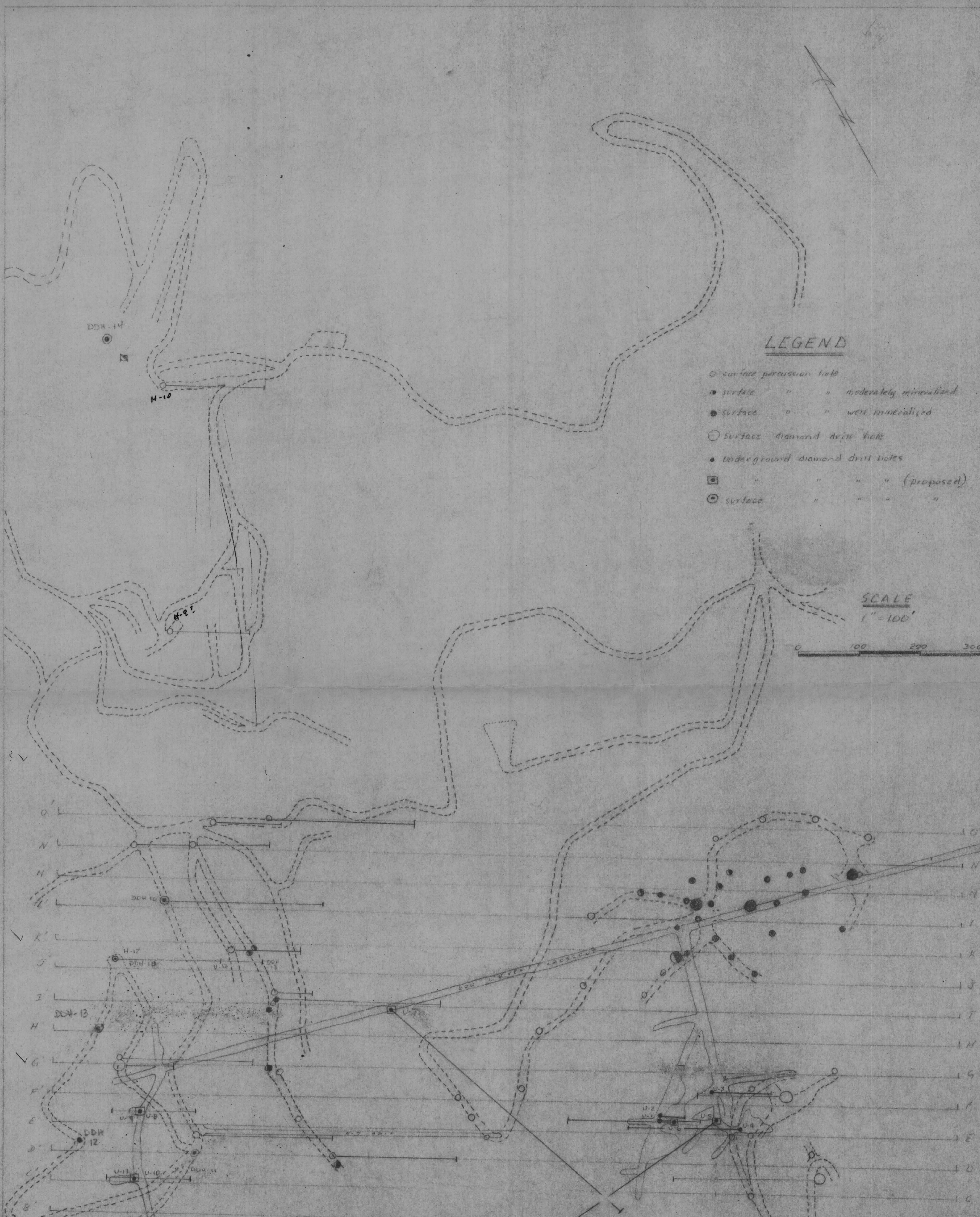
} SCHISTS ← { RHYOLITE PYROCLASTICS
LOW Fe, TUFFACEOUS CHERT

RHYOLITIC CHERT UNITS FORM LENTICULAR BAND ~~2000~~
~2,000 FT. ON STRIKE.

CU MINERALIZATION

- OCCURS IN TWO SILICEOUS SCHISTOSE LENSES UP TO 250 FEET WIDE = (EAST ZONE) SEPARATED BY ~500 FEET OF CHERT + BARREN SCHIST.

- BETTER GRADE MINERALIZATION = 5-600 FT ALONG STRIKE
- LOWER GRADE TRACED 1,500 FT ALONG WEST ZONE



CUTLASS EXPLORATION INC.

PROPOSED DRILLING FEB. 74

OFFICES:

TUCSON, ARIZONA (BUSINESS)
2343 E. BROADWAY, SUITE 206
P. O. BOX 12754, ZIP 85732
TELEPHONE (602) 622-0582

PERRY, KNOX, KAUFMAN, INC.

MINERAL EXPLORATION AND DEVELOPMENT

SPOKANE, WASHINGTON
NORTH 20 PINES ROAD, SUITE 21
P. O. BOX 14336, ZIP 99214
TELEPHONE (509) - WA 4-0878

Spokane, Washington
January 27, 1975

To: J. B. Imswiler
International Minerals and Chemical Corporation

From: Perry, Knox, Kaufman, Inc.

Subject: DeSoto Mine Evaluation, Yavapai County, Arizona

I. Summary and Recommendation

The DeSoto property is a Precambrian, volcanogenic, stratabound massive sulfide prospect from which approximately 290,000 tons of 3.3% copper ore, with nuinor gold, silver, and zinc values, have been mined. Two parallel zones of copper mineralization in altered siliceous, tuffaceous, rhyolitic schists have been defined by surface and subsurface exploration over strike lengths up to 1,500 feet (west zone) and to depths of 6-900 feet. Substantial widths of 0.5-1.0% copper mineralization have been shown to exist to the depths tested. It is not unreasonable to assume that other lenses of ore might exist at greater depth along this favorable rhyolitic horizon. Two to three million tons of economic grade copper ore, with associated gold and silver values, would support a very profitable operation.

It is recommended that an attempt be made to option the property on a reasonable basis; renegotiation of an underlying agreement will be required to obtain an acceptable payment schedule during the initial exploration period. Minor geologic mapping and a limited Turan E. M. Survey will be required. Three diamond drill holes are proposed to test the two zones at greater depth, approximately 3,000 feet of drilling. Estimated cost of this program is on the order of $53,000, exclusive of property payments.

II. General

The DeSoto property is situated in the Bradshaw Mountains (Prescott National Forest) approximately 8 1/2 miles S. SW. of Mayer and 3 miles west of Cleator in Sections 31 and 32, T. 11N, R. 1 E., Yavapai County, Arizona. Local topography is rather rugged, though not extreme, and poses no serious problem to exploration or possible development. The elevation in the immediate prospect area is approximately 5,500 feet. Current access is by several miles of rather second-rate road from the Mayer-Crown King road.

Past production is reportedly on the order of 290,000 tons of 3.3% copper, .05 oz./T gold, 1.2 oz/T silver, and less than 1% zinc. Part of this production undoubtedly was derived from higher grade semi-massive sulfide lenses that also contained better zinc values.

III. Geology

Some geological information on the general area can be found in several U.S.G.S. publications and maps:

U.S.G.S. Bulletin 782
U.S.G.S. Bulletin 1345
U.S.G.S. Geologic Map GQ 996
U.S.G.S. Geologic Map GQ 997
U.S.G.S. Air Mag Map GP 758

Of interest, also, are a report on the Iron King Mine in the A.I.M.E. volume, Ore Deposits of the U.S., and a general massive sulfide review by Sangster in Geological Survey of Canada Paper 72-22.

A review of the recent U.S.G.S. geologic mapping for the Mt. Union and Mayer quadrangles, which is at a scale of 1" = 1 mile and rather generalized, shows the DeSoto area to be underlain by Iron King Volcanics (Precambrian Archean Yavapai Schist) which consist mainly of andesitic and basaltic flows and rhyolitic flows and tuffs. The rhyolitic rocks exposed at the DeSoto are rather localized and are not shown on the U.S.G.S. map. The U.S.G.S. mapping, however, does show the DeSoto to be situated on the east flank of a west-dipping, overturned syncline; this indicates that the steeply-dipping bedding at the DeSoto is stratigraphically right side up.

The rocks exposed at surface and intersected by drilling at The DeSoto are believed to be mainly biotitic, chloritic, feldspar schists derived from andesitic tuffs and graywackes; quartz, sericite and quartz, chlorite, sericite schists derived from rhyolite pyroclastics, and very low iron, partially tuffaceous chert. The rhyolite-chert units apparently form a lenticular band at least 850 feet thick in the vicinity of the copper mineralization that extends along strike over lesser widths for perhaps 2,000 feet. At surface, the rhyolitic schists appear to possibly grade southward into the chert; their relationship at depth is unknown. Foliation, which probably conforms very closely to bedding, strikes approximately N 15-20° E. and dips 70-75° west.

The copper mineralization observed occurs mainly in two siliceous, schistose lenses up to 250 feet wide (east zone) separated by approximately 500 feet of chert and relatively barren schist. The better grades of mineralization exposed at surface and by underground workings extend perhaps 5-600 feet along strike, though lower grade mineralization can be traced for 1,500 feet along the west zone. Within the east zone primary sulfide mineralization, principally pyrite and chalcopyrite, occurs as streaks, disemenations and small semi-massive lenses, roughly parallel to foliation in economic, lenticular shoots to 40 feet in width, 150 feet in strike length, and several hundred feet in plunge length; these shoots have been followed in a zone down-dip approximately 900 feet. Low grade (less than 1% copper) pyrite-chalcopyrite mineralization in disseminations and streaks also occurs between the main shoots ~~on~~ in the east zone; mineralization in the west zone consists largely of this type also.

Within the higher grade zones of copper, black chloritic schist is associated with the quartz, sericite, chlorite schists. The black chlorite and nature of the sulfide mineralization is somewhat suggestive of stringer-type ore, though no cross-cutting relationships of the mineralization and bedding can be demonstrated; shearing, of course, can produce such a situation in stringer-type ore.

IV. Potential

The DeSoto property is a fairly typical Precambrian, volcanogenic, massive sulfide-type occurrence, although the percentage of sulfides present, for the most part, is insufficient to qualify the ore as massive sulfide. Mineralization is associated with acid volcanic pyroclastics and chert, the chert repre-senging a volcanic exhalite situated within an intermediate to basic vol-canic and volcanic-derived metasediment pile. Ore bodies in this type of environment often occur in clusters, usually near the top of the rhyolitic units. Exploration for this type of occurence on a strategraphic basis has proven successful in many instances in the Canadian Shield.

In the Bradshaw Mountain area, at least two other ore bodies have been found in similar geologic settings in the Iron King Volcanics. At the Blue Bell Mine 4 miles northeast, the situation is identical to that at the DeSoto with the exception that the thick lens of chert or weak sulfide iron formation is lacking; the Blue Bell has produced 1 1/2 million tons of copper ore from steeply plunging lenses extending to a depth of 1,500 feet; the grade is comparable to The DeSoto production. At the Iron King Mine 16 miles to the north, 5 million tons of 13-16% zinc-lead ore was mined; at the Iron King Mine, the early production was derived from the "footwall series," a series of short, small, massive sulfide lenses in andesite -- exploration in the hanging wall (Stratigraphic foot wall) finally found the main "I Series" that extended to a depth of 3,000 feet; the I Series was poorly exposed at surface, and where exposed, gave little indication of the magnitude of the ore zone below.

It is not unreasonable to assume that additional ore grade mineralization might occur at depth or along strike near the upper contact of the siliceous, schistose, rhyolitic rocks at the DeSoto. If present, this ore could be expected to be at least of comparable grade to that mined in the past; a larger lens or series of lenses might very possibly contain better grade. The zones of mineralization are still on the order of 100 feet wide and well mineralized at the deepest points explored.

Although it is believed that we are dealing primarily with stratabound, volcanogic-type mineralization occurring along a favorable rhyolitic horizon, there is the possibility that this is footwall zone, stringer-type mineralization. This is not particularly significant, however, since a number of very profitable orebodies consist entirely or in part of stringer ore.

The dimensions of a zone of mineralized rock required to constitute an at-tractive orebody are not great. A zone 40 feet in width, 750 feet long, and 1,000 feet in plunge length would provide a reserve of 2 1/2 million tons.

(USING 12 FT^3/T)
LOW

Although of relatively minor importance, the near-surface (open pit), acid soluble copper potential should be considered. Although incompletely evaluated, the presence of approximately 2 1/2 million tons of ±0.9% copper mineralization, part of which is leachable, can be demonstrated. A rough economic analysis of this potential indicates that it might be possible to realize a $500,000/year after tax profit over a 7 year life; royalties might reduce this ~~their~~ return considerably.

V. Past Exploration Efforts

Prior to the work of Cutlass Explorations Ltd., all exploration efforts were apparently confined to an evaluation of the acid soluble copper potential and to work immediately adjacent to the old workings; this exploration consisted of a number of short percussion and diamond drill holes.

Cutlass Explorations devoted most of its effort again to a study of the leaching possibilities; a series of percussion, rotary, and diamond drill holes was completed. The last stage of Cutlass' work apparently involved an attempt to test the down-plunge potential of the mineralized zones as shown on the enclosed cross sections. As will be noted, this work failed to reach any greater depth than the old workings in the east zone and only tested the upper 700 foot plunge length of the west zone (400 feet below surface due to topographic configuration). It should be noted that diamond drill holes in this area tend to be deflected perpindicular to foliation; it is probable that the Cutlass holes flattened somewhat and therefore tested less depth potential than indicated by the cross sections.

No geophysical or geochemical surveys have ever been run in the DeSoto area.

VI. Land Status

The DeSoto property consists of 19 patented and 24 unpatented claims. Although a claim map has not yet been provided us, it is assumed that these form a contiguous block with the exception of several patents situated in the valley bottom to the southeast. Sherwood Owens of Tucson is the owner of this ground.

Cutlass Explorations Limited of Vancouver, B.C., holds an option on this property from Owens. The terms of this agreement are not favorable. These terms are summarized as follows:

1. Advance payment schedule (as of January, 1975)
   (All payments due on 22nd of each month)
   $2,500/month - Jan-March, 1975
   30,000 - April, 1975
   2,500/month - May - July, 1975
   32,500 - August, 1975
   2,500/month minimum payments thereafter

2. 10% NSR royalty on production to $1.7 million (all payments applicable), then 5% NSR or $2,500/month minimum payments.

3. Any claims staked within a 3 mile perimeter area are included in this agreement.

Steve Radvak, Consultant for Cutlass, has been dealing with Owens for some time and feels that these terms can be renegotiated to some degree. The following amended schedule is suggested if IMC is interested in pursuing this matter:

1. 5% NSR royalty on production to a $1.5 million end price. (Radvak feels that Owens will not settle for an end price, and a 2% NSR royalty may be required after payments total $1.5 million.)

2. The advance payment schedule must obviously give us adequate exploration time at a reasonable price. At least six months time is needed, after which substantial payments might be justified if economic mineralization has been found.

   $2,500/month - 1st 6 months
   30,000 at end 6 months
   2,000/month thereafter.

3. Work commitments might be considered to make the proposed amendments more acceptable to Owens.

With respect to a deal with Cutlass, it seems obvious they would accept a 10% net profits arrangement. It is suggested that an attempt be made to acquire their interest for 7 1/2-8% net profits, all costs to be recovered on a proportional basis prior to any distribution of net profits. Cutlass claims to have expended approximately $350,000 to date including $125,000 in payments to Owens.

VII. Economic Considerations

Orebodies of the type that might occur on the DeSoto property provide a very attractive return on investment. Although it is not possible to accurately predict the economics of what might be found due to the variables involved, a rough approximation can be made based on reasonable assumptions. A briefly summarized evaluation is as follows:

Reserves: 2 million tons @ 3.5% copper, .05 oz/T gold, 1.0 oz./T silver.

Production Rate: 1,000 TPD; 6 years life

Capital Requirements:

$1.3 million - 1,000 foot, 2 compartment shaft and related facilities
0.7 million - preproduction expense (exploration, etc.)
1.0 million - mining equipment, misc. surface installations
4.0 million - 1,000 TPD flotation mill
$7.0 million - Total Capital

Estimated Operating Costs:

$8.00/T - Mining (Sublevel Stoping)
4.50/T - Milling
1.00/T - Overhead
3.50/T - depreciation and amortization
$17.00/T - Total

Mill Recovery: 90%

Concentrate Grade: 24%

Metal Values: 60¢ copper, $150 gold, $4.00 silver.

NSR Value per ton ore: $36.00/T
Pre-Tax Profit per ton: $19.00/T
Total Pre-Tax Profit after estimated royalty: ~~$3.6~~ $36 million

Estimated Total Net after Tax: $22 million
Estimated Annual Net: $3.6 million
Estimated Annual Cash Flow: $4.7 million
Payback Period: 1 1/2 years

Variations in metal prices, tax rates, depletion, and smelter schedules can obviously affect these numbers appreciably.

VIII Proposed Exploration Program

Exploration of the Desoto property would, at least initially, be quite simple. The down plunge extensions of the known mineralization and the favorable rhyolitic units should be tested at depth. The following program is proposed.

1. Geologic mapping - several days only.

2. Turam E. M. Survey - Although it is doubtful that penetration would exceed 600 feet, this is an inexpensive item (estimate $3,000 for 5 line miles) and should be run on the chance that an otherwise unknown target might be defined.

3. Diamond drilling - 3 holes, 3,000 feet; the initial hole would test the down plunge extension of both mineralized zones; the other two holes would test only the westernmost zone, probably the zone with the highest potential for deep economic mineralization. These holes would be drilled from the valley to the west, several hundred feet below the mineralized outcrops.

Estimated exploration costs are itemized below:

$40,000	Diamond drilling
2,000	Road and drill site construction
3,000	Assays
3,000	Turam Survey
1,000	Legal
4,000	Supervision
$53,000	Total

As you will note, no provision has been made for property payments.

Consideration must be given to the fact that since any agreement entered into on The DeSoto will require substantial monthly payments, any significant encouragement in the above described program will necessitate immediate subsequent efforts.

James A. Knox

James A. Know
Perry, Knox, Kaufman, Inc.

OFFICES:

TUCSON, ARIZONA (BUSINESS)
2343 E. BROADWAY, SUITE 206
P. O. BOX 12754 ZIP 85732
TELEPHONE (602) 622-0582

PERRY, KNOX, KAUFMAN, INC.

MINERAL EXPLORATION AND DEVELOPMENT

SPOKANE, WASHINGTON
NORTH 20 PINES ROAD SUITE 21
P. O. BOX 14336, ZIP 99214
TELEPHONE (509) - WA 4-0878

Tucson, Arizona
March 8, 1975

Mr. J. B. Imswiler
Manager of Exploration- W. USA
IMC
Suite 12, 390 Freeport Blvd.
Sparks, Nevada 89431

James A. Knox of this Company, operating under terms of our July 1, 1975 Agreement (IMC Arizona Project), recently studied the mineral potential of a massive sulfide copper occurrence situated in Yavapai County, Arizona -- <u>De Soto Mine</u>. Knox reported to IMC (his evaluation dated Jan. 27, 1975) indicating the property to be of possible economic worth. He briefly analyzed the return from such a property, making certain necessary assumptions, including the discovery of additional ore. In our view, the properties estimated return was substantial.

On March 7, after reviewing the De Soto with IMC management in Sparks, you indicated a "no interest" in pursuing the De Soto further.

Although no specific plan was submitted by PKK pertaining to the DeSoto as provided by our Agreement it was the intent of PKK to submit such a plan once we received a show of general interest on the part of IMC management.

PKK considers the De Soto a viable exploration target--assuming renegotiation with the owners is possible, as discussed by Knox.

PKK now asks IMC to release PKK from terms of the Agreement, with respect to the De Soto and the area of interest surrounding as described below -- as provided by Section 11, the Agreement.

Would you please forward this request to the officers of IMC for their approval. Please return one executed copy to PKK, the above Tucson address.

A. J. Perry
President
PERRY,KNOX,KAUFMAN,INC.

Approved: ______________________________ (IMC)

Dated: ______________________________

Area of Interest (De Soto) to include the following:

All of Sections 31 and 32, T11N-R1E
and
All of Sections 5 and 6, T10N-R1E., G&SRM,
Yavapai County, Arizona.

SWASTIKA MINE

A mile east of the Peck along the road to Peck siding are the Black Warrior and Silver Prince veins, now owned by Frank W. Giroux, of Mayer, under the name of the Swastika Silver & Copper Co.

The Silver Prince is mentioned in Raymond's report of 1877 with the statement that the cost of packing the ore to Prescott was $50 a ton. The Mint report for 1883 mentions both veins, stating that the Black Warrior was 2 to 3 feet wide, that $40,000 in silver had been extracted so far, and that 8 tons a day was milled in a 4-stamp mill for a yield of 113 ounces of silver to the ton. About 1885 the mine was considered exhausted, and it was idle until reopened by F. W. Woods in 1910. From 1910 to 1915 the mine produced 600,000 ounces of silver. The total production is stated to be about 1,000,000 ounces. Since 1915 the mine has been in intermittent operation. Mr. Woods states that from 1875 to 1908 the Silver Prince had yielded $480,000 and the Black Warrior $385,000. The later production came wholly from the Silver Prince.

The country rock consists of Yavapai schist, mostly fissile and sericitic, with lenses of quartzite, but the outcrops are deeply oxidized. The two parallel veins strike due north and dip 60° W., with the schist. The Prince lies 300 feet west of the Black Warrior. Between the two there is a 50-foot dike of light-colored porphyry.

The Silver Prince is developed by tunnels and a 400-foot shaft about 600 feet to the north. The vein is at most a few feet wide and carries dark-brown limonitic ore. There is a little quartz, but the principal gangue mineral is a sideritic carbonate, with native silver, chloride, and some sulphides. The sulphides consist of a partly decomposed tetrahedrite rich in silver and a little chalcopyrite. The ore, which contains a little lead, was sold to El Paso and the lead smelter at Needles in 1914 and later shipped to Salt Lake City.

The shaft on the Black Warrior is said to be only 125 feet deep. The lowest levels were not visited, but it is evident that the ore on them is poorer. Here, too, the conditions are similar to those at the Peck, namely, an extraordinary concentration in the oxidized zone and impoverishment below. Considering the history of this mine, it would be rash to say that it is exhausted. More comments on the concentration in the oxidized zone of these deposits are found on page 49.

DE SOTO MINE

The outcrops of the De Soto copper mine lie 2 miles northeast of the Peck mine, on the summit of the high ridge separating Peck Canyon from Crazy Basin. The altitude is about 5,800 feet. The main tunnel is 600 feet below the outcrop, and from it an incline

leads down to Middleton station on the Crown King branch road. The property is owned by the same interests that control the Humboldt smelter (Southwest Metals Co.), to which the ore has been shipped. Work was discontinued in 1922, the ore bodies being considered exhausted. Most of the information given below was obtained from Mr. J. L. White, of the staff of the smelter.

The Yavapai chloritic schists strike N. 23° E. at the mine and dip 70° NW. The ore bodies, which carry pyrite-chalcopyrite ore and are contained in a chloritic schist, have yielded a total of 180,000 tons, averaging about 3.75 per cent of copper with 1 ounce of silver and 0.02 ounce of gold to the ton. The last ore treated contained 2.25 per cent of copper. There is less pyrite than at the Blue Bell

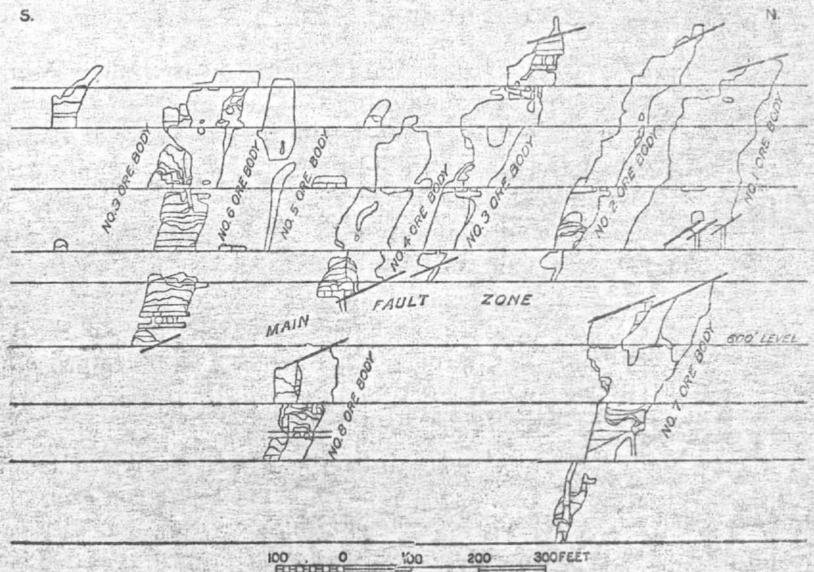


FIGURE 10.—Longitudinal section of ore lenses in the De Soto mine

mine, some sphalerite and galena, and occasional specimens of tetrahedrite. (See pl. 17, *A*.) A few prisms of arsenopyrite were observed.

The gangue is fine-grained quartz. There are also lenses of coarser quartz, much of it crushed and showing undulous extinction. Gangue and sulphides replace the schist, which is mainly chloritic with a little biotite.

The ore occurs in overlapping lenses. (See fig. 10.) On the upper levels there were seven such lenses close together, with an individual width of as much as 50 feet and a greatest length of 250 feet. Exploration extended to a depth of 300 feet below the main tunnel level, and at this depth only one small lens persisted. The total length of the ore zone is 350 feet; the total width 200 feet.

The ore bodies are said to have been cut off in depth by a flat fault 250 feet below the surface. The small bodies found below this depth are believed to represent the continuation of the ore below the fault. The ore bodies are shown in Figure 10, each separately, in a longitudinal projection, for they overlap so that they can not be indicated in their correct relative position.

OTHER PROPERTIES

The veins of the Gold King group, in the southern part of the district, are said to be the extension of the Gladiator vein, which is in the Pine Grove district. Near by is the Blue Bird vein. Both these deposits are in Yavapai schist.

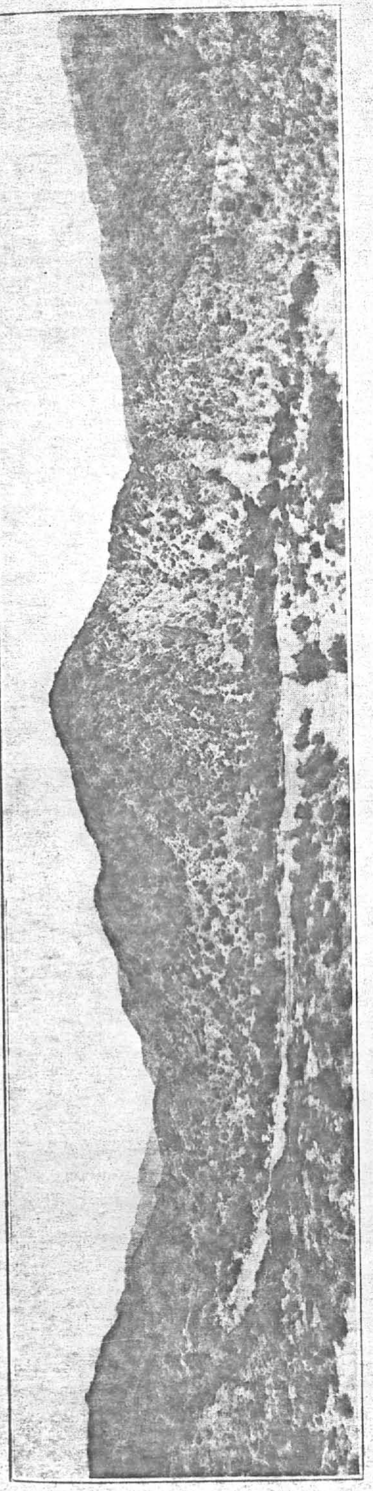
PINE GROVE DISTRICT

The Pine Grove district lies in the heart of the Bradshaw Mountains, in a well-timbered region, at an altitude of 6,000 to 7,500 feet. (See pl. 21, *B*.) Its highest point is the diorite mass of Towers Mountain. Most of the claims lie in a basin-like depression on the east side of the ridge. The district is reached by an automobile road from Prescott, 40 miles long, and by a branch railroad from Mayer, which ascends Crazy Basin and Poland Creek in a series of switchbacks. It is an old mining region, and many of the veins were very rich near the surface. The earliest properties worked were the Del Pasco, Gladiator, and War Eagle. The ores carry silver and gold.

Most of the mines are situated in granodiorite (quartz diorite, according to Jaggar and Palache), which forms a rounded mass 3 to 4 miles in diameter, intruded into Bradshaw granite and still earlier Yavapai schist. The granodiorite is cut by a series of dikes which trend north-northeast across the center of the area. In part these dikes are rhyolite porphyry, in part granite porphyry. There are also some light-colored granitic dikes which seem to be affiliated with the granodiorite; the others just mentioned appear to represent a distinctly later intrusion.

There are three prominent vein systems, which trend north-northeast and generally dip about 60° WSW. The shoots have a tendency to pitch northward. They occur mostly in the granodiorite but continue also to the north in Yavapai schist, diorite, and mixed areas (Wildflower mine), though these harder rocks are as a rule less favorable. Few of the veins are more than 5 feet in width, and they contain a filling of predominant quartz, with some ankerite and calcite.

Much of the quartz is drusy and contains more or less pyrite, chalcopyrite, zinc blende, and galena, with some tetrahedrite. In places free gold occurs in the primary ore. Most of the ore extracted



A. VIEW LOOKING UP BLACK CANYON

Mine of Howard Copper Co. in the distance

B. VIEW LOOKING NORTHEAST FROM CROWN KING STATION TOWARD CRAZY BASIN