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GEOLOGY & URANIUM RESOURCES OF THE ANDERSON MINE PROJECT YAVAPAI COUNTY, ARIZONA

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Report MXU-76-2

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GEOLOGY AND URANIUM RESOURCES

OF THE

ANDERSON MINE PROJECT

YAVAPAI CO., ARIZONA

CONCLUSIONS AND RECOMMENDATIONS

The bulk of the uranium mineralization at the Anderson Mine property is associated with fine-grained carbonaceous lacustrine sediments. Drilling to date has established the following uranium resource potential:

Cutoff (%eU_3O_8)	Average grade (%eU_3O_8)	Average thickness (feet)	eU_3O_8 (millions of lbs.)
.02	.046	20.6	27.7
.03	.061	13.8	23.0
.05	.090	8.4	16.8
.07	.117	6.4	13.3

A drill program on a 200-foot grid pattern is recommended as the next step in development of the resources. Additional milling characteristic studies should be conducted.

Mineralization extends south under claims held by Urangesellschaft. It is also recommended that joint venture possibilities with Urangesellschaft be investigated.

INTRODUCTION

The Anderson Mine property is located approximately 50 miles northwest of Wickenburg, Arizona on the south side of the Santa Maria River approximately 20 miles west of State Highway 93.

Access from this highway is via both improved and unimproved dirt road (Fig. 1). The property consists of 88 unpatented mining claims and one 640-acre state lease located in portions of sections 2 and 9 through 16, T. 11 N., R. 10 W., Yavapai County, Arizona (Fig. 2).

Anomalous radioactivity was first detected in the area by Mr. T. R. Anderson of Sacramento, California using an airborne scintillometer in January, 1955. After ground checking disclosed uranium oxide in outcrop, several hundred claims were located. The property was drilled and a few small shipments of ore were made. In 1958, approximately 4,300 tons of ore averaging 0.21% U_3O_8 were shipped (verbal communication, M. Jones). It is thought that during this period a group, including Messrs. Jones and Jacobs, obtained control of the acreage. During 1967-68, Getty Oil Company obtained an option on the area. The property was subsequently dropped by Getty after drilling delineated several small pods of uranium mineralization. The uranium prices in 1968 of approximately $6.50 per pound probably influenced Getty's decision to relinquish the property. They did, however, retain uranium property in the vicinity of Artillery Peak, approximately 18 miles to the northwest.

In 1968, Minerals Exploration Company's Tucson office received a submittal on the area. It was forwarded to the Casper office in 1969 where, after initial turndown, it remained in the files until 1974 when the increasing price of uranium created a

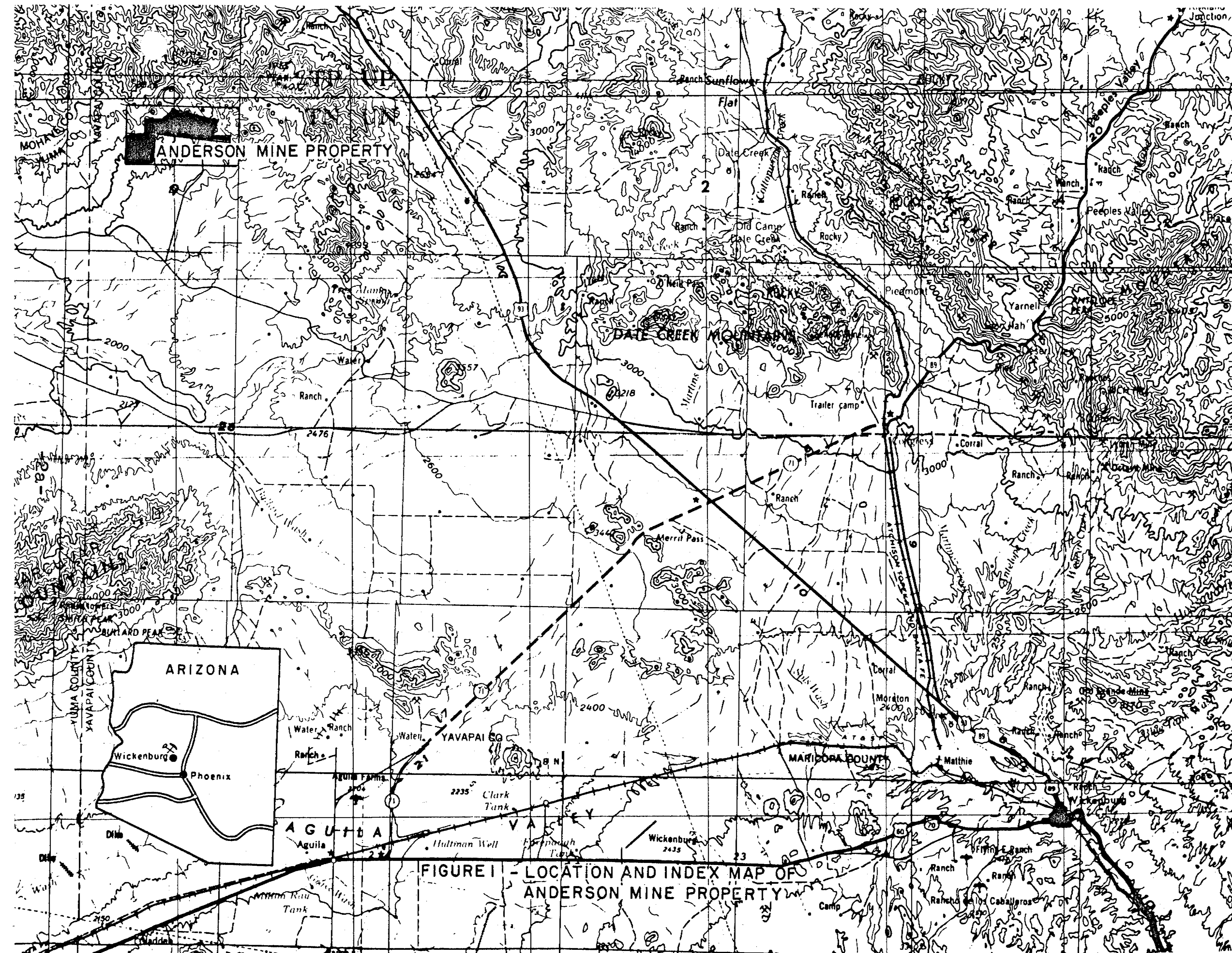
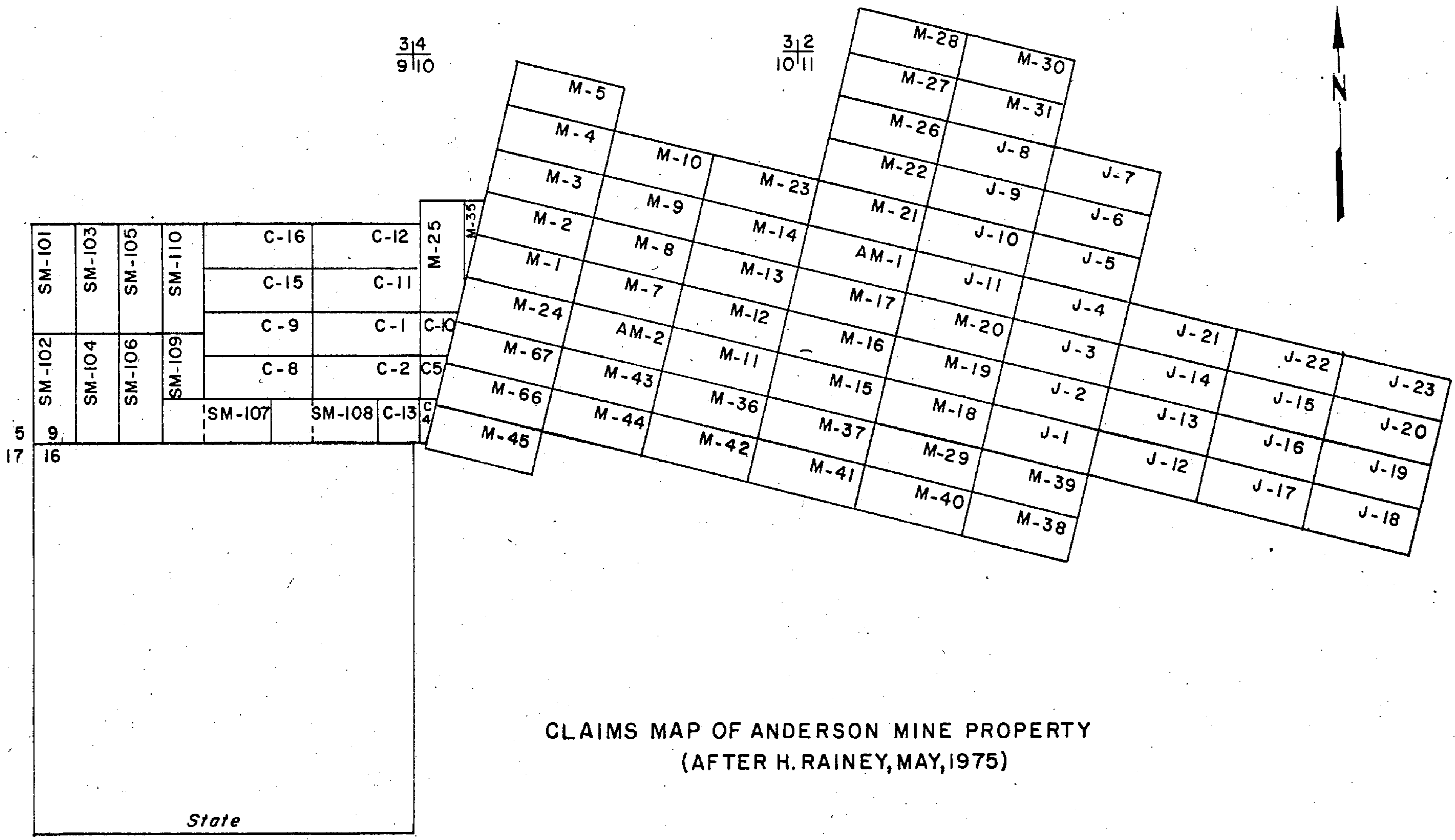


FIGURE I - LOCATION AND INDEX MAP OF ANDERSON MINE PROPERTY



CLAIMS MAP OF ANDERSON MINE PROPERTY
(AFTER H. RAINEY, MAY, 1975)

FIGURE 2

renewed interest in the property. Following a field check, discussions with Mr. Jones, and evaluation of the 1968 Getty drill data, an option was taken on the property in late 1974. Minerals Exploration Company purchased the property in 1975 after a 53 hole, 19,000 foot drilling program on 800-foot centers confirmed the much greater uranium resource for the area that had been interpreted from the 1968 Getty gamma log data. A 180 hole, 74,000 foot drill and core program on 400-foot centers conducted from November 1975 through February 1976 further delineated the uranium resources. To date, a total of 211 holes have been drilled and 15 holes cored by Minerals Exploration Company. This report summarizes the geology and uranium resources of the Anderson Mine property.

GEOLOGY

Stratigraphy

As interpreted from drill hole data and surface geologic mapping, six major informal stratigraphic units are recognized. In ascending order, they are: 1) a 'basement' volcanic unit of andesitic composition; 2) a succession of tuffaceous lacustrine strata; 3) a sandstone-conglomerate unit; 4) a basaltic flow unit; 5) a capping conglomerate; and 6) recent alluvium (Fig. 3).

TERTIARY VOLCANIC ROCKS (Tv)

A succession of volcanic rocks of intermediate (andesitic) composition form the 'basement' upon which younger lacustrine and subaerial clastic sediments were deposited. The volcanics, which

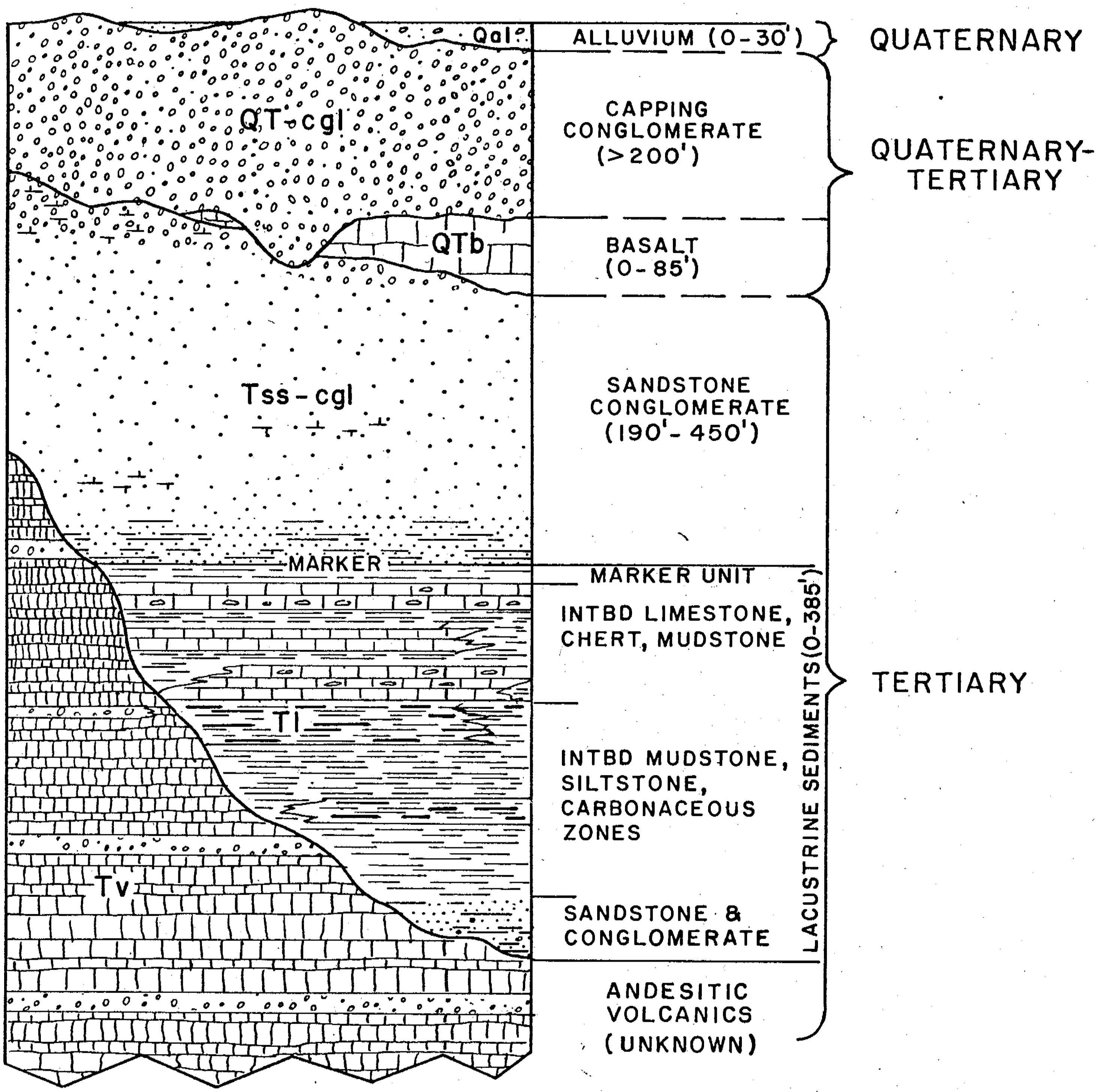


FIGURE 3 STRATIGRAPHIC COLUMN SHOWING PRINCIPAL STRATIGRAPHIC UNITS. UNIT THICKNESSES (NOT DRAWN TO SCALE) SHOWN IN PARENTHESES

PREPARED BY
STEPHEN J. PAVLAK

JUNE 1976

crop out in the northern portion of the area (Plate 1), form irregular peaks and boulder strewn slopes.

The unit consists of a sequence of red-brown, gray, and black volcanic flows and interbedded light-colored volcaniclastic sediments. Reyner and others (1956) have described the unit as a fine-grained, vesicular augite andesite which locally contains calcite-filled amygdules.

The 'basement' terrain dips gently (10°-15°) in a southerly direction and is characterized by an irregular upper erosion surface upon which locally thick paleosols developed. Faulting and folding have further modified the 'basement' topography (Plate 2, 2a). Interpretation of drill hole data indicates that the irregular paleotopography of the 'basement' may be responsible for much of the local thinning and thickening and apparent onlap relationships in the overlying sediments.

LACUSTRINE STRATA (Tl)

Up to 385 feet of predominantly lacustrine strata unconformably overlie the basal volcanic complex. The strata contain an appreciable amount of volcanic material and may, in large part, be considered waterlain tuffaceous sediments.

The sequence of lacustrine strata is subdivided into: 1) a basal coarse clastic unit; 2) a mudstone-siltstone unit containing intercalated carbonaceous zones; 3) a succession of interbedded limestones, silicified limestones, cherts, mudstones, and siltstones; and 4) a thin fissile, fossiliferous marker bed which has been designated the top of the lacustrine unit.

These strata appear to have been deposited in a northwest trending, fault-controlled (?) depression. To the west, the unit thins and grades laterally into coarse-grained tuffaceous sediments in the vicinity of a west bounding normal fault. To the north and east, it thins and appears to lap onto the volcanic complex. The unit thickens to the south.

Other than the organic material contained within the carbonaceous zones mentioned above, abundant plant remains, including twigs, reeds and small rough-walled cylinders resembling roots are present throughout much of the lacustrine unit. Fresh water mollusks, up to $1\frac{1}{2}$ inches in length, are locally common, especially in calcareous beds. Reyner and others (1956) recognized two zones containing abundant silicified palm-type wood. The leg bone of a duck found in this unit has been dated as Miocene by the Los Angeles County Museum.

Basal Coarse Clastic Unit

Arkosic sandstone and conglomerate, which occur principally along the southern margin of the area, rest unconformably on the volcanic 'basement', and interfinger with and grade laterally and vertically into the mudstone and siltstone unit. These sediments are typically gray, fine to coarse-grained, and contain abundant quartz, white to a gray feldspar, biotite, and common granitic, metamorphic, and red-brown volcanic clasts. The immature nature of these sediments suggests transportation from a nearby source (possibly the Pre-cambrian crystalline terrain to the north). Thinly-laminated to very thinly-bedded and locally cross-bedded carbonaceous siltstones occur in the basal coarse clastic sediments. Minor uranium mineralization occurs in this unit.

Mudstone-Siltstone Unit

The mudstone-siltstone unit unconformably laps onto the irregular erosion surface of the andesitic volcanics and locally rests conformably on the underlying coarse clastic unit. The unit is predominantly fine-grained. It consists of a thick succession of olive green, gray-green, and brown tuffaceous mudstones and siltstones intercalated with black, gray-brown, and blue-green carbonaceous to lignitic tuffaceous mudstones and siltstones. The non-carbonaceous sediments are thin to thick-bedded (many appear massive in cores), weakly to moderately indurated, and are locally silicified. The thickness of the unit is variable, attaining a maximum in excess of 250 feet.

Two persistent carbonaceous zones have been recognized from drill hole cuttings. The zones are interbedded with, and grade laterally into the non-carbonaceous sediments, and thus constitute a paludal facies of the mudstone-siltstone unit. The nature of the carbonaceous material is quite variable. It is expressed as disseminated carbon trash in fine-grained sediments, as organic films on stratification planes, and as lignite. These zones crop out as carbonaceous partings and disseminated carbonaceous material in thin-bedded calcareous mudstones at two locations near the inferred northern carbonaceous/non-carbonaceous facies boundary: approximately 200 feet northwest of drill hole AM 39 and 200 feet northwest of drill hole AM 8. Limestones and silicified mudstones and siltstones are locally common in the carbonaceous sediments.

The bulk of the uranium mineralization occurs in the mudstone-siltstone unit in association with the carbonaceous material.

Limestone-Chert-Mudstone Unit

A succession of limestones, cherts, mudstones, and siltstones conformbably overlies the mudstone-siltstone unit. The base of the unit is arbitrarily chosen on the bottom of the lowermost major limestone-chert bed. The unit is characterized by both marked thickening and thinning and by rapid facies changes. The unit grades laterally and vertically from massive, nearly pure limestones to thinly-bedded calcareous and silicified mudstones and siltstones.

The limestones, which are typically white, gray and gray-green, commonly contain fine-grained detritus. The interbedded olive-green tuffaceous siltstones and mudstones contain relict pumice shards, are thin to thickly-bedded, and commonly have a punky character. Much of this unit is partially to intensely silicified with vari-colored chert (including red, brown, pink, white, orange, and green). A diagenetic origin for the silicification is suggested. Silica, released from the alteration of volcanic glass, is prevalant as chalcedonic and opal cement, as veins, pods, and nodules, and as silicified plant remains. Bentonite, another product of the devitrification process, is common. Subordinate amounts of uranium mineralization are associated with this unit.

Calcareous Marker Bed Unit

The unit consists of approximately 20 to 30 feet of light brown, very fine-grained sandstone to olive green siltstone capped by a thin (1 to 2 feet) fissile, fossiliferous marlstone. It conformably overlies the limestone-chert-mudstone unit except in the vicinity of Hill 2027, where the contact appears unconformable in the subsurface. The rapid thinning of the underlying limestone-chert-mudstone unit observed in drill holes AM 6, 18, 26, and 160 suggests the presence of a local diastem.

The marlstone, which is laterally persistent across much of the area, serves as an excellent marked bed and is designated as the top of the lacustrine strata. Small (up to 2 inches) fish fossils are contained within the marker.

SANDSTONE-CONGLOMERATE UNIT (Tss-cgl)

The sandstone-conglomerate unit rests conformably on the underlying lacustrine strata except southeast of Hill 2079 in the SW$\frac{1}{4}$ sec. 10, T. 11 N., R. 10 W. where it disconformably laps onto the andesitic volcanics. The unit, which is exposed in the slopes beneath the resistant capping basalt, is laterally continuous for long distances and can be mapped approximately seven miles to the west where it overlies a succession of red beds in the vicinity of Palmerita Station.

The unit is homogeneous laterally and is characterized by a coarsening upward sequence of clastic rocks, which in ascending order consist of: 1) green and brown siltstones; 2) very fine to medium-grained friable arkosic sandstones; and 3) an arkosic pebble to cobble conglomerate. The sandstone is loosely consolidated but locally contains abundant calcareous ribs. The conglomerate is typically light brown and contains abundant rounded granitic and metamorphic clasts in a fine to coarse-grained arkosic sand matrix. The conglomerate contains a minor amount (approximately 5%) of volcanic clasts. The lower part of the unit is well-sorted and is distinctly bedded. Locally, the coarser clastics are calcite-cemented forming white cliff faces in outcrop.

The unit is variable in thickness due to a pronounced unconformity at the top. Its maximum observed thickness is approximately 450 feet in drill hole AM 26.

BASALT (QTb)

Unconformably overlying the sandstone-conglomerate unit is a gently dipping basalt flow which is exposed discontinuously at or near the top of the cliffs along the central and southern portions of the property. The basalt attains a maximum thickness of approximately 85-90 feet in the vicinity of the west bounding fault, thins abruptly from about 40 feet to 5 feet near drill hole AM 122, and pinches out to the east in the vicinity of drill hole AM 175. As suggested by Reyner and others (1956) and by a subsurface oxidized zone within the basalt, two flows are probably present in the western part of the area.

The basalt is black, fine-grained to aphanitic, and contains calcite-filled amygdules. The basalt is commonly jointed parallel to the flow surface.

CAPPING CONGLOMERATE (QTcgl)

A conglomerate unconformably overlies the basalt flows and where the latter are absent, rests unconformably on the sandstone-conglomerate unit. It crops out along the southern portion of the area where it forms the cap rock of the cliffs and rolling upland which extend westward to Palmerita Station. Total thickness of the unit is unknown, but up to 200 feet is either intercepted in drill holes or exposed in outcrop.

This unit is brown to pinkish-brown and consists predominantly of sub-rounded, silicic to intermediate volcanic clasts up to three feet in diameter. Pebbles and cobbles from $\frac{1}{4}$ to 3 inches in diameter are common. The matrix consists of a medium to coarse-grained sand. The unit is weakly to moderately indurated and is thinly to thickly-bedded.

The conglomerate appears to have been deposited by streams and mudflows on an erosion surface developed on the basalt. Locally, the conglomerate fills relatively deep channels dissected through the underlying basalt and into the sandstone-conglomerate unit (Plate 6).

ALLUVIUM (Qal)

Unconsolidated sand and gravel derived from the aforementioned units are found in present-day drainages. Caliche has formed where these deposits have been calcite-cemented.

Structure

The most conspicuous structural feature of the area is the gentle (5^{o}-15^{o}) southerly regional dip. Reyner and others (1956) suggest that this has resulted from recurring faulting before, during, and after sedimentation.

All major faults are normal and trend N. 35° W. to N. 55° W. The largest fault, located near the western claims boundary in sections 9 and 16, dips steeply (approximately 80°) to the southwest. Comparison of projected basement elevations in drill holes AM 11-10-16-1 and AM 64 suggests a vertical displacement of approximately 1,000 feet. In the vicinity of drill hole AM 11-10-16-1, a paralleling fault dips approximately 50° SW and offsets the capping basalt at least 200 feet.

A large hinge fault located in the central part of the claims group (hereafter referred to as Fault 1878) dips steeply (75°-80°) to the southwest. It appears to offset the capping conglomerate at least 200 feet vertically, with displacement gradually diminishing to nothing approximately one-half mile northwest along strike in the vicinity of drill hole AM 119.

Another fault occurs at the eastern edge of section 11. Little is known regarding its displacement since no holes have intersected the fault.

Minor faults and shear zones were noted but are believed to have a minimal ? amount of displacement. Such features may represent either fracturing and slight offset of strata during differential compaction of the underlying sediments or local adjustment to major faulting. One such zone, located in the southeast corner of section 10, can be extended northwest where it is expressed as a fault zone in the volcanic basement rocks. The fault cannot, however, be traced in the subsurface any distance south of the central portion of section 10.

The largest fold in the area is a broad gentle northwest trending syncline in the SE$\frac{1}{4}$ sec. 9, T. 11 N., R. 10 W. (see Plates 1, 2, 2a, 4, 5, 6, and 7). Dips attain a maximum of 13^{o} except where modified by shearing. Many smaller folds with amplitudes of several feet are present in the lacustrine strata, particularly in the limestone-chert-mudstone unit.

URANIUM MINERALIZATION

The major portion of the uranium resources at the Anderson Mine occurs in the tuffaceous mudstone-siltstone unit of the Tertiary lacustrine strata in close association with carbonaceous material. Subordinate amounts of mineralization occur within the overlying limestone-chert-mudstone unit and within the underlying coarse clastic unit.

Two grade thickness maps (Plates 17 and 18) delineate a blanket type deposit with dimensions of approximately 5,000 feet by at least 3,000 feet. Information obtained from Urangesellschaft via a log exchange indicates that the mineralization extends southwest of our claims boundary for a distance of approximately 10,000 feet beneath the Urangesellschaft claims.

Mineralization in the Mudstone-Siltstone Unit

The vertical and lateral distribution of most of the mineralization appears to be coincident with the paludal facies of the lacustrine strata. In general, where carbonaceous sediments occur, mineralization is present. However, barren carbonaceous sediments occur in some areas. Non-carbonaceous siltstones and mudstones adjacent to the paludal facies are often mineralized. The carbonaceous zones attain a thickness of up to 170 feet in the southwest corner of section 10. Mineralized zones (greater than or equal to .02% eU_3O_8 grade) within this interval range up to 64 feet in aggregate thickness.

The ore mineralogy within the black carbonaceous units below the water table has not been determined. Yellow and yellow-orange uranium minerals described as carnotite by Reyner and others (1956) crop out in association with carbonaceous material. An unidentified green mineral is sometimes present in association with the yellow uranium minerals. Limonite and hematite staining commonly occurs in the mineralized zones. Black (manganese?) coatings are frequently found within hematite-stained zones.

Mineralization in the Limestone-Chert-Mudstone Unit

Uranium mineralization within the limestone-chert-mudstone unit occurs as blooms of yellow uranium minerals and as coatings on fracture surfaces. The bulk of the mineralization in this unit is located in the SW¼, sec. 11, T. 11 N., R. 10 W., although scattered occurrences are present throughout the area. Thickness of individual mineralized intervals range up to 23 feet and average three feet. More than one mineralized zone can be observed in many areas.

Mineralization in the Coarse Clastic Unit

A minor amount of mineralization is associated with carbonaceous zones within the coarse clastic unit. Traces of hematite staining and bright orange feldspar present in yellow-gray sandstone intervals may indicate sandstone alteration. No solution front type trend has been identified. Mineralization in this unit is confined to the south-central portion of the claims group in the vicinity of drill hole AM 26.

Structural Control of Mineralization

Subsurface interpretations suggest that uranium mineralization occurred prior to faulting and folding. Mineralized zones have been displaced as much as 180 feet by Fault 1878. Mineralized intervals appear to have been folded at the northwestern corner of the area in the vicinity of Hill 2079.

Distribution of Mineralization

The distribution of the mineralization can be summarized as follows:

1) The bulk of the Anderson Mine resources is associated with carbonaceous material of the mudstone-siltstone unit. It is generally confined to the area between Fault 1878 and the eastern boundary of sections 9 and 16.
2) The bulk of the mineral within the limestone-chert-mudstone unit is located in an area northeast of Fault 1878.
3) Mineralization in the vicinity of Hill 2079 at the west end of the property shows no preference for a particular lithology.

4) Mineralization associated with carbonaceous material in the coarse clastic unit is limited to the south-central area in the vicinity of drill hole AM 26.

5) Scattered occurrences of uranium are found in all lithologic subdivisions of the lacustrine strata, except the calcareous marked bed unit.

Origin of Mineralization

Reyner and others (1956) suggest three possible origins of the uranium at the Anderson Mine property: hypogene, ash leach, and bog deposition. Interpretation of Minerals Exploration Company data favors a variation of Reyner's ash leach - bog deposition theory. Uranium was probably derived from silicic volcaniclastic lacustrine material with subsequent precipitation in reduction traps.

Reyner and others (1956) cite field evidence in favor of a hypogene source and state that: 1) uranium ore has not been observed beyond the boundary faults; 2) intense silicification has altered mudstone and limestone; 3) limonite and hematite staining occurs on bedding and fracture planes; 4) calcite, chalcedony, sepiolite, and manganese are found associated with the west bounding fault. This field evidence can be interpreted differently. Our drilling data indicates that the carbonaceous sediments also have not been observed beyond the boundary faults. This may explain why the mineral is localized within the boundary faults. Further, if uraniferous solutions migrated up faults, one would expect mineral and grade to be concentrated along the faults. Our subsurface interpretations indicate no such association. Data indicates that faulting offsets mineralization. Intense silicification is most probably a result of devitrification of silicic volcaniclastic sediments. Bentonite, common in the area, is also an alteration product of tuffaceous material. Hematite and limonite stain on bedding and fracture planes was possibly derived from pyrite associated with carbonaceous material. Calcite, sepiolite, chalcedony, and manganese deposited along the western fault may indicate movement of fluids along this zone; but without associated uranium, such deposits can not significantly be cited as evidence that uraniferous solutions migrated up the fault zone.

Reyner and others (1956) speculate that two other types of origin are possible: ash leach and bog type deposition. Both leaching of ash and deposition in bog type reduction traps are conceivable. Reyner and others (1956) allude to a vitrophyric andesite source. Andesite is not commonly a uranium source. Silicic volcanic rocks are known to contain anomalous amounts of uranium (Love, 1961; Turekian and Wedepohl, Table 2, 1961). Surface and subsurface interpretations suggest a modified ash leach-bog deposition sequence as the origin of uranium at the Anderson Mine property.

The presence and diagenesis of the tuffaceous component of the lacustrine sediments in combination with adjacent geochemically favorable paludal sediments, provides a possible model for the origin and deposition of uranium at the Anderson Mine property.

The age of mineralization was probably early. Diagenesis, occurring during compaction and dewatering of the lake sediments, released uranium from volcanic material. The known affinity of uranium for carbonaceous material (Breger, 1974) accounts for the fixation of uranium within the paludal unit. Contact of uranium bearing solutions with the reducing environment produced by the abundant carbonaceous material resulted in the precipation of uranium within and adjacent to the carbonaceous sediments. Some remobilization of uranium in recent geologic time has resulted in uranium, silica, and carbonate deposition in fractures.

URANIUM RESOURCES

All of the gamma ray logs have been digitized. This data has been run through a gamlog computer program which produced a thickness and mineral grade report for each mineralized interval. The area of influence of each mineralized hole has also been calculated. This area is a function of the density of holes in the drill pattern. A maximum area of influence was arbitrarily set at approximately 283,000 square feet. Utilizing this data, tonnages and pounds of uranium ore were tabulated for each mineralized interval in each hole. This information is summarized in the following table:

No. of holes	Cutoff grades	Average thick.* (feet)	Average grade % eU_3O_8	Mineralized area ($ft^2 x 10^6$)	Avg. depth to top of mineral (ft)	eU_3O_8** ($lbs x 10^6$)
185	.02	20.6	.046	24.6	259	27.7
174	.03	13.8	.061	22.9	268	23.7
143	.05	8.4	.090	18.2	283	16.8
114	.07	6.4	.117	14.5	303	13.3

*The thicknesses and grades for each mineralized interval have been diluted by one-half foot of waste at the top and bottom of each interval.

**Tonnage factor data are presented as Appendix 2.

These summaries should be viewed as approximate resource estimates of indicated pounds of uranium in the ground. Actual recoverable reserves have not been determined.

The resource total may be increased by extending the mineralization into the southeast corner of section 16. Drill hole data in the area indicates grade thicknesses of up to 0.68 (Plate 17) at depths greater than 650 feet. A north-south line of barren holes limits the western extent of mineralization to the E½, sec. 16, T. 11 N., R. 10 W. Drill data indicates that the paludal facies of the lacustrine strata is absent in this area. Extension of resources eastward is unlikely as geologic mapping and reconnaissance indicate that little or no lacustrine strata are present.

Exchanged logs indicate that the mineralization extends up to 10,000 feet south-wouthwest of our southern boundary under Urangesellschaft property. These logs indicate aggregate mineralized thicknesses of up to 38 feet and grades of up to 0.08% eU_3O_8 at depths ranging from 550 to 1760 feet. A joint venture agreement may be beneficial to us provided that future drilling by Urangesellschaft develops sufficient uranium reserves.

Chemical analyses of 15 core holes for vanadium indicate grades of up to 0.28% V_2O_5. In general, the grade of vanadium increases proportionally with uranium grade. No vanadium resources have been calculated. Such a value would be tied to the uranium resource value by a vanadium/uranium ratio which has not yet been calculated.

A semi-quantitative X-ray fluorescence analysis for minor elements was performed by Hazen Research Inc. on two core pulps from core hole AM 16C. No potentially commercial quantities of any other elements besides uranium are evident from the results (see Appendix 3). Similar analyses are planned on other cores.

Uranium amenability data are presented as Appendix 1.

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APPENDIX I

Uranium Amenability Studies

The following data regarding ore amenability tests conducted on selected core samples by Hazen Research Inc., Golden, Colorado summarizes all testing to date by Minerals Exploration Company:

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4501

September 25, 1975

Mr. G. E. Marrall
Minerals Exploration Company
P. O. Box 2674
Casper, Wyoming 82601

Re: HRI Project 1833
Uranium Amenability Studies, Anderson Mine

Dear Mr. Marrall:

The sample of split core, "AM-16-c," delivered by you on September 4, 1975, was crushed through 6-mesh and blended to form our sample HRI 8630. An analytical pulp taken from the composite contained 0.107% U_3O_8.

Agitation leaches were performed on portions of the composite ground to 28-mesh. A carbonate leach at 80°C and 33% solids, for 24 hours with 160 lb/ton Na_2CO_3, 80 lb/ton $NaHCO_3$, and 8 lb/ton $KMnO_4$ added, solubilized 86% of the uranium. The resulting leach pulp was very slow filtering. A sulfuric acid leach at 80°C and 33% solids, for 24 hours required the addition of 472 lb/ton sulfuric acid and 36 lb/ton sodium chlorate to solubilize 87% of the uranium into a 1.5 pH leach solution. Data sheets and material balances for these two tests are attached. It would appear that this particular sample is more suited to carbonate leaching.

Two simulations each were made of both the UKAEA and Holmes and Narver acid-bake/cure processes. In the first series, only 200 lb/ton of sulfuric acid was used, whereas, the second series used 400 lb/ton. The UKAEA procedure was to pug 28-mesh ore with the acid and chlorate at 80% solids and hold it at 100°C for three hours. The moist ore was then pulped at 25% solids for 30 minutes, filtered, washed, and dried. Only 7% of the uranium was solubilized in the test with 200 lb/ton sulfuric acid and 2 lb/ton sodium chlorate. Increasing the reagent additions to 400 lb/ton and 4 lb/ton sodium chlorate raised the uranium extraction to 67%.

The Holmes and Narver simulation consisted of wetting 6-mesh ore in a flask with about 13% water. Concentrated sulfuric acid, 96%, was added and the mixture was pugged. The temperature rose to a maximum of 60°C upon the addition of the acid. The flask was then loosely stoppered and was placed in a 60°C oven for 48 hours. The cured ore was then pulped at 25% solids for 30 minutes, filtered, washed, and dried. About 9% of the uranium was solubilized in the test with 200 lb/ton sulfuric acid and only 32% was solubilized in the test with 400 lb/ton sulfuric acid.

It would appear that the shale and clay in the ore are too reactive with the acid to allow good uranium extractions when using low acid addition techniques. The Holmes and Narver technique is also limited by the coarse ore size. Assuming that this sample is representative, it is my opinion that additional tests be made to attempt optimization of the carbonate leaching technique. The poor filterability of the carbonate-leached ore indicates that the uranium recovery from the leach solution would be by a resin-in-pulp technique.

Unless I hear to the contrary, we shall do further carbonate leaches next week.

Yours truly,

John

John E. Litz
Project Manager

JEL:nd
Encls.

Uranium Carbonate Leach Amenability, 472-71
HRI-8630, 0.107% U_3O_8

160 lb/ton Na_2CO_3
80 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	$KMnO_4$ Cum lb/ton
0	45	9.7	-130	8
1	78	9.2	-170	
2	80	9.1	-180	
3	80	9.2	-190	
4	81	9.2	-190	
6	82	9.3	-190	
12	80	9.4	-180	
24	81	9.6	-200	
	25	10.1	-40	

Metallurgical Balance

Sample Time Hours	Filtrate + Wash			Residue			U_3O_8 Extraction %
	Volume ml	U_3O_8 g/l	U_3O_8 g	Weight g	U_3O_8 %	U_3O_8 g	
6				17.5	0.019	0.003	82
12				14.5	0.014	0.002	87
24	1260	0.197	0.248	214	0.015	0.032	86

Overall calculated head, 0.114% U_3O_8

Uranium Acid Leach Amenability, 472-72

HRI-8630, 0.107% U_3O_8

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	72	6.2 / 1.3	340	384	9.6
1	78	1.2	460		13.6
2	80	1.35	450		21.6
3	80	1.6 / 1.3	470	440	25.6
4	81	1.3	480		29.6
6	82	1.5 / 1.3	440	472	36.0
12	80	1.4	430		
24	80	1.8	380		
	25	1.5	310		

Metallurgical Balance

Sample Time Hours	Filtrate + Wash			Residue			U_3O_8 Extraction %
	Volume ml	U_3O_8 g/l	U_3O_8 g	Weight g	U_3O_8 %	U_3O_8 g	
6				17	0.018	0.003	83
12				14	0.017	0.002	84
24	1270	0.197	0.250	217	0.015	0.003	87

Overall calculated head, 0.115% U_3O_8

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4501

October 28, 1975

Mr. G. E. Marral
Minerals Exploration Company
P. O. Box 2674
Casper, Wyoming 82601

Re: HRI Project 1833
Uranium Amenability Studies, Anderson Mine

Dear Mr. Marral:

Additional carbonate leaches were performed on 28-mesh portions of composite "AM-16-C," our sample HRI-8630. Leach tests are summarized in Table 1.

Table 1

Uranium Amenability Studies, Anderson Mine

Conditions: 0.107% U_3O_8, 33% solids, 40 g/l Na_2CO_3
20 g/l Na_2HCO_3, 80°C

		Uranium Extraction		
Test No.	Oxidant	6 Hours	12 Hours	24 Hours
472-75	32 1/	88	88	88
472-76	0	79	81	81
472-77	Air 2/	84	86	85
472-78 3/	0	79	81	82
472-79 4/	Air	-	84	-
472-80	Air	-	77	-
472-81 5/	Air	-	-	90

1/ lb/ton $KMnO_4$
2/ With copper amine catalyst
3/ 80 g/l Na_2CO_3, 20 g/l $NaHCO_3$
4/ 20 g/l Na_2CO_3, 10 g/l $NaHCO_3$
5/ Temperature, 65°C

Leach Test No. 472-81, at 65°C for 24 hours and 10-20 cc/min air bubbling through, solubilized 90% of the uranium. From this particular test it would appear that air is an efficient oxidant. The resulting leach pulp was very slow filtering. On Tests 472-75 and 472-77 a chemical oxidant was added, 88% and 85% of the uranium was solubilized.

The carbonate leach Test No. 472-76, with no oxidant, gave a low uranium extraction. Only 81% was soluble.

Detailed data sheets and material balances for each amenability test are attached.

Thank you for the work and we hope we can assist you further in this project.

Yours truly,

Robert Balderrama JEK

Robert Balderrama
Research Engineer

RB:mk
Enclosures

URANIUM CARBONATE LEACH AMENABILITY, 472-75
HRI - 8630, 0.107 % U_3O_8

160 lb/ton Na_2CO_3
80 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv	$KMnO_4$ Cum lb/ton
0	70	9.4	-90	32
2	76	9.4	-140	
4	80	9.8	-150	
6	80	9.9	-170	
12	80	10.0	-190	
24	79	10.1	-190	

Metallurgical Balance

Sample	Filtrate+wash			Residue			U_3O_8
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 g	weight g	U_3O_8 %	U_3O_8 g	Extraction %
6				20	.013	.003	88
12				17.4	.013	.002	88
24	1070	.247	.264	209.5	.013	.027	88

URANIUM CARBONATE LEACH AMENABILITY, 472-76
HRI-8630, 0.107% U_3O_8

160 lb/ton Na_2CO_3
80 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv
0	72	9.4	-370
2	76	9.3	-180
4	80	9.75	-200
6	80	9.8	-218
12	81	9.92	-240
24	80	10.0	-215

Metallurgical Balance

Sample	Filtrate+wash			Residue			U_3O_8
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 g	weight g	U_3O_8 %	U_3O_8 g	Extraction %
6				18.7	.023	.004	79
12				16.0	.020	.003	81
24	1030	.243	.250	211	.020	.042	81

LABORATORY WORKSHEET
HAZEN RESEARCH, INC.

URANIUM CARBONATE LEACH AMENABILITY, 472-77
HRI-8630, 0.107% U_3O_8

160 lb/ton Na_2CO_3
80 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv	$CuSO_4$ Cum. lb/ton	NH_3 Cum lb/ton	Air cc/min
0	74	9.3	-160	8	8	10
2	77	9.3	-130			
4	81	9.7	-150			
6	80	9.8	-135			
12	80	9.92	-130			
24	80	10.0	-155			

Metallurgical Balance

Sample Time Hours	Filtrate + wash Volume ml	U_3O_8 g/l	U_3O_8 g	Residue weight g	U_3O_8 %	U_3O_8 g	U_3O_8 Extraction %
6				19	.017	.003	84
12				16.5	.015	.003	86
24	990	.258	.255	209.3	.016	.034	85

LABORATORY WORKSHEET
HAZEN RESEARCH, INC.

URANIUM CARBONATE LEACH AMENABILITY, 472-78
HRI-8630 0.107 % U_3O_8

320 Lb/ton Na_2CO_3
80 Lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv
0	75	9.6	-390
2	77	9.6	-210
4	81	9.95	-240
6	80	10.0	-215
12	81	10.1	-230
24	81	10.1	-225

Metallurgical Balance

Sample Time Hours	Filtrate + wash Volume ml	Filtrate + wash U_3O_8 g/l	Filtrate + wash U_3O_8 g	Residue weight g	Residue U_3O_8 %	Residue U_3O_8 g	U_3O_8 Extraction %
6				19.2	.023	.004	79
12				16.5	.020	.003	81
24	1060	.241	.255	210.6	.019	.040	82

LABORATORY WORKSHEET
HAZEN RESEARCH, INC.

URANIUM CARBONATE LEACH AMENABILITY, 472-79
HRI- 8630 0.107 % U_3O_8
80 lb/ton Na_2CO_3
40 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv	Air cc/min
0	66	9.2	-160	10-20
2	79	9.0	-190	
4	79	9.1	-200	
6	79	9.3	-180	
12	81	9.4	-170	

Metallurgical Balance

Sample	Filtrate+wash			Residue			U_3O_8
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 g	weight g	U_3O_8 %	U_3O_8 g	Extraction %
12	1080	.219	.236	247	.018	.044	84

URANIUM CARBONATE LEACH AMENABILITY, 472-80

HRI-8630 0.107 % U_3O_8

160 Lb/ton Na_2CO_3

80 Lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv	Air cc/min
0	67	9.4	-220	10-20
2	79	9.2	-240	
4	79	9.3	-225	
6	80	9.4	-200	
12	81	9.5	-202	

Metallurgical Balance

Sample Time Hours	Filtrate + wash Volume ml	Filtrate + wash U_3O_8 g/l	Filtrate + wash U_3O_8 g	Residue weight g	Residue U_3O_8 %	Residue U_3O_8 g	U_3O_8 Extraction %
12	1130	.229	.259	248	.025	.062	77

URANIUM CARBONATE LEACH AMENABILITY, 472-81

HRI-8630 0.107% U_3O_8

160 lb/ton Na_2CO_3

80 lb/ton $NaHCO_3$

Elapsed Time Hours	Temp °C	pH	emf mv	Air cc/min
0	68	9.98	-140	10-20
2	64	9.98	-130	
4	67	9.4	-100	
6	66	9.4	—	
12	66	9.3	—	
24	63	9.2	-25	

Metallurgical Balance

Sample	Filtrate + wash			Residue			U_3O_8
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 g	weight g	U_3O_8 %	U_3O_8 g	Extraction %
24	1120	.250	.280	245.5	.011	.027	90

HAZEN RESEARCH, INC.

4601 INDIANA STREET
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TELEPHONE 303/279-4501

January 24, 1975

Mr. G. E. Marrall
Minerals Exploration Company
P. O. Box 2674
Casper, Wyoming 82601

Re: Uranium Amenability Studies
HRI Project 1684

Dear Mr. Marrall:

Three samples from the Anderson Mine property were received on December 11, 1974. The samples were crushed and portions were split out for head analyses. The analyses of the samples are listed below:

Upper Lense	0.033% U_3O_8
Lower Lense	0.022% U_3O_8
Stockpile	0.075% U_3O_8

Portions of each sample were dry ground to a nominal minus 14-mesh. Acid leach amenability tests were performed at 1.3 pH and both 20 and 80°C. A sodium carbonate leach amenability was also performed on the stockpile sample. The sodium carbonate amenability was at 80°C using a lixiviant containing 40 g/l sodium carbonate and 20 g/l sodium bicarbonate.

The uranium extractions after 24 hours of leaching are listed below:

Sample	Temperature °C	Acid Added lb/ton	Sodium Chlorate Added lb/ton	Uranium Extraction Percent
Upper lense	20	272	4.2	31
Lower lense	20	63	3.0	78
Stockpile	20	108	3.2	67
Stockpile	80	241	2.4	98
Stockpile*	80	-	-	95

*Carbonate leach

The uranium extraction rate curves for the leach tests are depicted in Figure 1. The detailed data for each test are attached.

The samples tested formed very thixotropic mixtures when pulped with water. Therefore, the amenabilities were done at a pulp density of 33% solids. The leached pulps also demonstrated very poor settling and filtering characteristics. Part of this is due to the very fine particle size of the leach residue.

The acid additions per pound were quite high. Over 100 pounds of acid were added per pound of solubilized uranium at 20°C and over 160 pounds at 80°C. It would appear that agitation leaching with acid would not be the most practical or economic method of recovering the uranium from this ore.

The reagent additions for the carbonate leach were high and would require recovery and recycle steps that, because of the poor settling characteristics of the ore, would be inefficient. It is possible that a heap leaching or acid bake process would permit lower acid additions while still obtaining satisfactory uranium extraction.

Yours truly,

John E. Litz

John E. Litz
Project Manager

Enclosures
JEL:mk

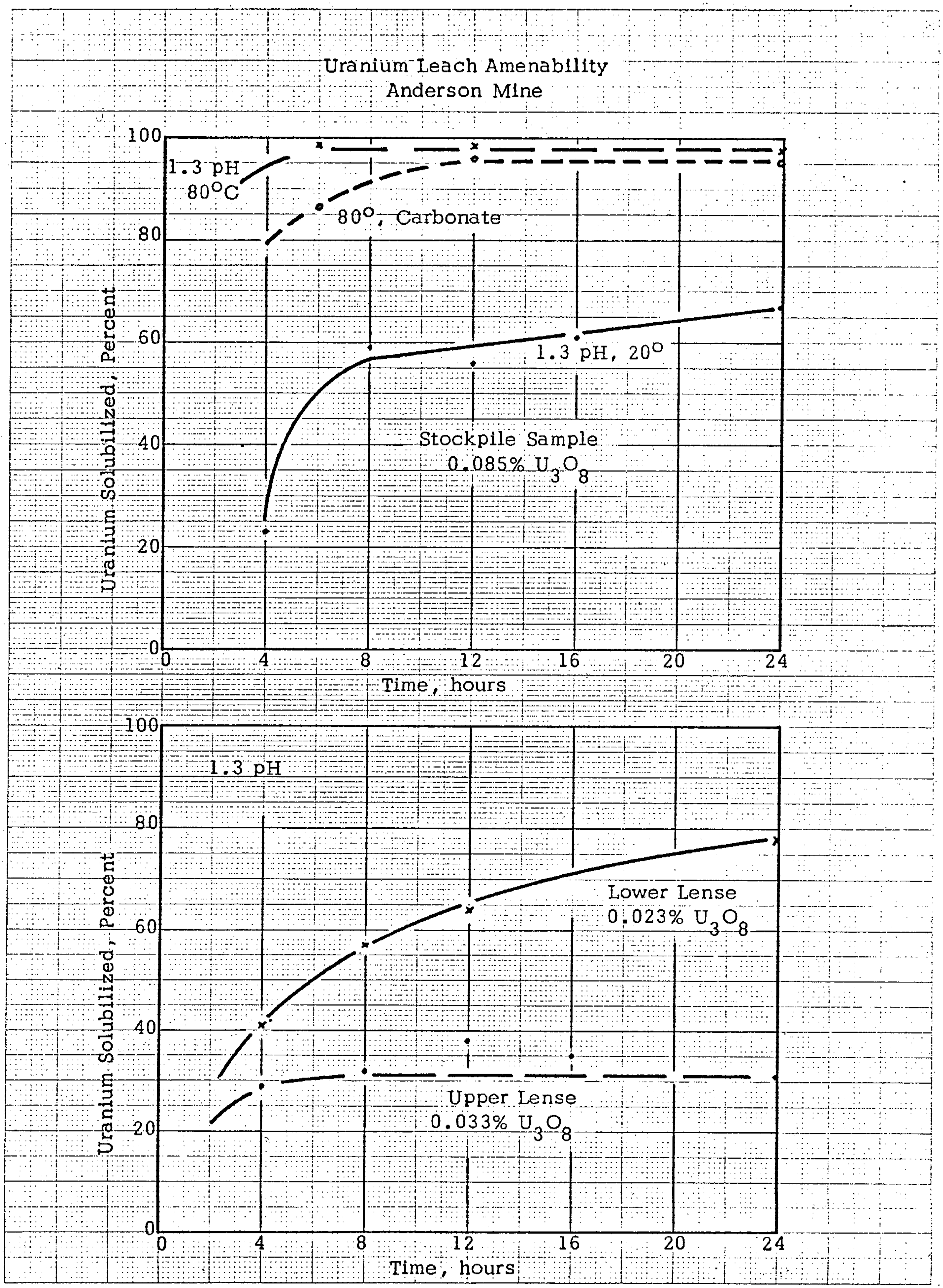
Uranium Leach Amenability
Anderson Mine
1.3 pH
80°C
80°, Carbonate
1.3 pH, 20°
Stockpile Sample
0.085% U_3O_8
Uranium Solubilized, Percent
Time, hours
1.3 pH
Lower Lense
0.023% U_3O_8
Upper Lense
0.033% U_3O_8
Uranium Solubilized, Percent
Time, hours

Figure 1

hri

Acid Leach Amenability of Upper Lense: HRI Sample 7708-1

0.033% U_3O_8

Elapsed Time Hours	Temp °C	Leaching Conditions pH Read/Adjust	emf	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	22	8.1/1.3	+270	211	0.8
1	24	2.5/1.2	+240	245	2.4
2	23	1.5/1.25	+360	261	2.8
3	22	1.3	+375	261	3.6
4	22	1.35/1.3	+415	268	4.2
5	22	1.3	+470	268	4.2
6	22	1.4/1.3	+500	272	4.2
8	22	1.3	+515	272	4.2
12	21	1.4	+505	272	4.2
16	20	1.45	+505	272	4.2
24	20	1.5	+515	272	4.2

Metallurgical Balance

Sample Time Hours	Filtrate + Wash Volume ml	U_3O_8 b/l	U_3O_8 Grams	Residue Weight Grams	U_3O_8 %	U_3O_8 Grams	U_3O_8 Calc Head %	Extraction %
4	238	0.0073	0.0017	19.3	0.021	0.0041	0.030	29
8	242	0.0069	0.0017	17.2	0.021	0.0036	0.031	32
12	239	0.0085	0.0020	17.6	0.019	0.0033	0.030	38
16	237	0.0074	0.0018	15.6	0.021	0.0033	0.033	35
24	1305	0.0326	0.0425	444	0.021	0.0932	0.031	31

Overall calculated head 0.031

Acid Leach Amenability of Lower Lense: HRI Sample 7708-2

0.022% U_3O_8

Elapsed Time Hours	Temp °C	Leaching Conditions pH Read/Adjust	emf	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	22	8.9/1.3	+335	41.2	0.8
1	22	1.55/1.3	+345	49.2	1.2
2	21	1.3	+380	49.2	1.6
3	21	1.4/1.3	+390	54.0	2.2
4	21	1.4/1.3	+410	58.8	2.8
5	21	1.3	+430	58.8	3.0
6	21	1.4/1.3	+450	62.8	3.0
8	21	1.3	+530	62.8	3.0
12	20	1.35	+545	62.8	3.0
16	20	1.4	+555	62.8	3.0
24	20	1.4	+565	62.8	3.0

Metallurgical Balance

Sample Time Hours	Filtrate + Wash Volume ml	U_3O_8 g/l	U_3O_8 Grams	Residue Weight Grams	U_3O_8 %	U_3O_8 Grams	U_3O_8 Calc Head %	Extraction %
4	228	0.0076	0.0017	16.2	0.015	0.0024	0.025	41
8	231	0.0105	0.0024	17.8	0.010	0.0018	0.024	57
12	237	0.0122	0.0029	18.0	0.009	0.0016	0.025	64
16	221	0.0153	0.0034	15.3	0.029	0.0044	0.051	33
24	1310	0.0581	0.0761	423	0.005	0.0211	0.023	78

Overall calculated head 0.024% U_3O_8

Acid Leach Amenability of Stockpile: HRI Sample 7708-3

0.075% U_3O_8

Elapsed Time Hours	Temp °C	Leaching Conditions pH Read/Adjust	emf	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	21	8.3/1.3	+330	75.2	0.8
1	22	1.95/1.3	+300	88.0	2.0
2	21	1.6/1.3	+370	97.2	2.4
3	21	1.35/1.3	+385	98.8	3.0
4	21	1.4/1.3	+445	102.0	3.2
5	21	1.35/1.3	+510	104.8	3.2
6	21	1.3	+540	104.8	3.2
8	21	1.35	+550	104.8	3.2
12	20	1.4	+555	104.8	3.2
16	20	1.5/1.45	+565	108.4	3.2
24	1.45		+565	108.4	3.2

Metallurgical Balance

Sample Time Hours	Filtrate + Wash Volume ml	U_3O_8 g/l	U_3O_8 Grams	Residue Weight Grams	U_3O_8 %	U_3O_8 Grams	U_3O_8 Calc Head %	Extraction %
4	236	0.0152	0.0036	18.1	0.062	0.0118	0.081	23
8	239	0.0266	0.0064	16.4	0.027	0.0044	0.066	59
12	230	0.0293	0.0067	15.3	0.034	0.0052	0.078	56
16	226	0.0340	0.0079	16.5	0.030	0.0050	0.078	61
24	1355	0.174	0.236	436	0.027	0.1177	0.081	67

Overall calculated head 0.081% U_3O_8.

Acid Leach Amenability of Stockpile: HRI Sample 7708-3

0.075% U_3O_8

Elapsed Time Hours	Temp °C	Leaching Conditions pH Read/Adjust	emf	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	77	8.1/1.3	+ 20	170	2.4
1	82	1.5/1.25	+655	182	2.4
2	81	1.3	+670	182	2.4
3	80	1.5/1.3	+675	191	2.4
4	80	1.3	+680	191	2.4
6	81	1.35	+675	191	2.4
12	83	2.0/1.5	+670	241	2.4
24	81	1.6	+685	241	2.4

Metallurgical Balance

Sample Time Hours	Filtrate + Wash Volume ml	U_3O_8 g/l	U_3O_8 Grams	Residue Weight Grams	U_3O_8 %	U_3O_8 Grams	U_3O_8 Calc Head %	Extraction %
6	237	0.079	0.0187	20.2	0.0016	0.0187	0.094	98
12	235	0.074	0.0174	18.0	0.0016	0.0174	0.098	98
24	1020	0.370	0.377	446	0.370	0.377	0.087	98
		Overall calculated head		0.085				

Carbonate Leach Amenability of Stockpile: HRI Sample 7708-3

0.075% U_3O_8

Elapsed Time Hours	Temp °C	Leaching Conditions pH Read	emf	Na_2CO_3 Cum lb/ton	$NaHCO_3$ Cum lb/ton
0	77	8.1	+ 20	160	80
1	82	9.9	-100	160	80
2	81	9.7	- 80	160	80
3	80	9.95	- 60	160	80
4	80	9.8	- 60	160	80
6	81	9.75	- 55	160	80
12	83	9.9	- 40	160	80
24	81	10.1	- 40	160	80

Metallurgical Balance

Sample Time Hours	Filtrate + Wash Volume ml	U_3O_8 g/l	U_3O_8 Grams	Residue Weight Grams	U_3O_8 %	U_3O_8 Grams	U_3O_8 Calc Head %	Extraction %
6	225	0.071	0.0160	20.6	0.0119	0.0025	0.090	87
12	230	0.096	0.0221	19.3	0.0051	0.0010	0.120	96
24	1020	0.356	0.363	457	0.0042	0.0192	0.084	95

Overall calculated head 0.085

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4501

June 2, 1975

Mr. G. E. Marrall
Minerals Exploration Company
P. O. Box 2674
Casper, Wyoming 82601

Re: HRI Project 1757
Uranium Amenability Studies

Dear Mr. Marrall:

The two samples received from you on May 16, 1975, were evaluated for their amenability to ambient and hot sulfuric acid leaching. Sample AM-16c-1 contained 0.101% U_3O_8 and sample AM-16c-2 contained 0.052% U_3O_8. The samples were leached at 33% solids for 24 hours using sulfuric acid and sodium chlorate to control the pH at 1.3 and emf at >400 mv for the first eight hours.

The results are summarized below and the detailed data for each test are attached. The high carbonate content of these samples does not permit economic acid leaching. Thief samples taken at eight hours show that the uranium is readily soluble; so it is probable that the samples would be amenable to carbonate leaching. Additional sample would be necessary for a carbonate leach test.

Sorry the results are not more encouraging.

Yours truly,

John E. Litz
Project Manager

JEL:mgp
Enclosures

Sample	Temp ^{o}C	Acid Added lb/ton	Sodium Chlorate Added lb/ton	Uranium Extraction Percent	Acid Addition lb/lb U_3O_8 Dissolved
AM-16c-1	25	165	4.8	73	116
	80	310	8.0	89	172
AM-16c-2	25	825	6.4	56	1313
	80	779	9.6	78	902

Acid Leach Amenability of AM-16c-1
at 25°C, 0.101% U_3O_8

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	25	6.9/1.3	+310	68	1.6
1	25	1.8/1.3	+390	98	3.2
2	25	1.6/1.3	+390	131	4.8
3	25	1.7/1.3	+490	165	4.8
4	25	1.0	+500	165	4.8
6	25	1.0	+520	165	4.8
8	25	1.0	+520	165	4.8
24	25	1.5	+510	165	4.8

Metallurgical Balance

Sample	Filtrate + Wash			Residue			Uranium
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 Grams	Weight Grams	U_3O_8 %	U_3O_8 Grams	Extraction %
8	75	0.098	0.0074	8.4	0.025	0.0021	78
24	710	0.239	0.1697	236.0	0.027	0.0637	73

Overall calculated head 0.097% U_3O_8

Acid Leach Amenability of AM-16c-1
at 80°C, 0.101% U_3O_8

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	25	7.0/1.3	+250	111	1.6
1	80	2.6/1.3	+350	202	3.2
2	80	2.0/1/3	+350	294	4.8
3	80	1.3	+400	294	4.8
4	80	1.3	+390	294	6.4
6	80	1.4/1.3	+420	302	6.4
8	80	1.4/1.3	+360	310	8.0
24	85	1.5	+350	310	8.0

Metallurgical Balance

Sample	Filtrate + Wash			Residue			Uranium
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 Grams	Weight Grams	U_3O_8 %	U_3O_8 Grams	Extraction %
8	70	0.106	0.0074	11.3	0.010	0.0011	87
24	600	0.363	0.2178	235.0	0.011	0.0259	89
	Overall calculated head			0.101% U_3O_8			

Acid Leach Amenability of AM-16c-2
at 25°C, 0.052% U_3O_8

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	25	6.7/1.3	+140	613	1.6
1	25	5.2/1.3	+170	705	3.2
2	25	5.0/1.3	+175	745	4.8
3	25	3.7/1.3	+400	771	4.8
4	25	3.3/1.3	+370	797	6.4
6	25	2.8/1.3	+450	818	6.4
8	25	1.4/1.3	+450	825	6.4
24	25	1.4	+450	825	6.4

Metallurgical Balance

Sample	Filtrate + Wash			Residue			Uranium
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 Grams	Weight Grams	U_3O_8 %	U_3O_8 Grams	Extraction %
8	75	0.0298	0.0022	8.4	0.028	0.0024	48
24	800	0.0955	0.0764	275.6	0.022	0.0606	56
	Overall calculated head			0.057% U_3O_8			

Acid Leach Amenability of AM-16c-2
at 80°C, 0.052% U_3O_8

Elapsed Time Hours	Temp °C	pH Read/Adjust	emf mv	H_2SO_4 Cum lb/ton	$NaClO_3$ Cum lb/ton
0	25	7.3/1.3	+250	585	1.6
1	80	2.6/1.3	+370	675	3.2
2	80	2.0/1.3	+365	755	4.8
3	80	1.3	+390	755	4.8
4	80	1.4/1.3	+390	763	6.4
6	80	1.4/1.3	+350	771	8.0
8	80	1.4/1.3	+360	779	9.6
24	80	1.5	+250	779	9.6

Metallurgical Balance

Sample	Filtrate + Wash			Residue			Uranium
Time Hours	Volume ml	U_3O_8 g/l	U_3O_8 Grams	Weight Grams	U_3O_8 %	U_3O_8 Grams	Extraction %
8	70	0.0417	0.0029	7.7	0.008	0.0006	82
24	665	0.158	0.1051	266	0.011	0.0293	78
	Overall calculated head			0.055% U_3O_8			

C O N F I D E N T I A L

LABORATORY TESTS CONDUCTED ON ANDERSON MINE URANIUM ORES USING TL LEACHING TECHNIQUES

by

Paul H. Johnson, PhD

Hydrometallurgical Engineer

February 13, 1974

HOLMES & NARVER, INC.

A Resource Sciences company

Advanced Technology / Engineering / Construction / Management / Maintenance & Operations

400 EAST ORANGETHORPE AVENUE • ANAHEIM, CALIFORNIA 92801

INTRODUCTION

A series of leach tests have been conducted on five different ore and waste samples from the Anderson uranium mine, near Wickenburg, Arizona using the new TL Leaching concept.

TL Leaching may briefly be described as a highly modified form of percolation leaching wherein crushed, unsized ore is efficiently leached by downward percolation in a relatively short period of time and with a minimum of leaching reagents. This leaching and rinsing would be achieved in a plant having low capital and operating costs. Previous testwork has indicated that this technique is far superior to any of the present means of leaching ores. For example, it is not uncommon to achieve above 95% recovery on a minus ½ inch oxide copper ore that would only yield about 80% recovery in a complicated and much more expensive vat leach. This new technique is protected by a patent application.

A previous agitation-percolation test on a high grade (0.18% U_3O_8) sample of this ore yielded a uranium dissolution of 92.7 % in three days of leaching (minus 3 mesh) at an acid consumption of 65 # H_2SO_4 per ton of ore.

ORE SAMPLES

AUHS 1 (0.06% U_3O_8 and 0.055% V_2O_5)

This sample was obtained from the large stockpile near the center of the main pit area. It was a mixture of shale, mudstones, and sandstones.

AUHS 2 (0.009 % U_3O_8)

AUHS 2 was a limey mudstone waste material from a pile near the edge of the main pit area.

AUHS 3 (0.045% or 0.065% U_3O_8)

This sample was obtained from the same pile as AUHS 1 and was of the same type.

AUHS 4 (0.0272 % U_3O_8)

This was a low grade mud and siltstone sample from a pile in the main pit.

AUHS 5 (0.10 % U_3O_8)

Sample AUHS 5 was a sample representative of the ore-grade shale material. It was obtained from a stockpile located below the main pit area.

The in-place density of these samples measured between 15 and 17 cu. ft. per ton. The crushed bulk density was about twice these values.

DESCRIPTION OF TESTS

The following table summarizes the test conditions of the five tests run on these ores.

TABLE 1. Anderson Mine Uranium Ore Tests - Summary of Test Conditions

Test No.	Ore Sm.	% U_3O_8	Size, in.	Days Cure	H_2SO_4 Cure Sol'n	Rinse Days	Ore Weight & Column Height, in.
TLAU 1	AUHS 1	.06	$-\frac{1}{2}$	2	100#/T @29%** Soln	2	6510 gms. 52" Final
TLAU 2	AUHS 2	.009	$-\frac{1}{2}$	"	"	2	2360 gms. 19.5 Final
TLAU 3	AUHS 3	.045 - .065	$-\frac{1}{2}$	"	" plus Soln 1-1	2 + 2*	3360 gms. 23 Start
TLAU 4	AUHS 4	.0272	$-1\frac{1}{2}$	"	100#/T @ 29% Soln	"	3263 gms. 30" Start
TLAU 5	AUHS 5	.10	-3/8	2	"	"	4000 gms. 44 Start 39 Final

*The column contents were turned over and rerinsed as in a face-of-heap final rinse cycle.

**Degree of wetting in curing stage.

TEST RESULTS

The uranium recoveries and the acid consumptions for these five tests are tabulated below in Table 2.

TABLE 2. Anderson Mine Uranium Ore Tests - Summary of Test Results

Test No.	% U_3O_8 Recovery	#/Ton of Ore Acid Consumption*
TLAU 1	93.6	150
TLAU 2	95 - 99	155
TLAU 3	Low **	120
TLAU 4	98.7	140
TLAU 5	97.8	111

* This acid consumption is a gross acid consumption and does not deduct acid generated in the solvent extraction step or equivalent acid in the form of ferric iron derived from leaching.

** This sample was accidently overwetted in its curing stage. This undoubtably resulted in poor percolation and poor recovery.

CONCLUSIONS

The principle conclusions to be drawn from these test data are that exceptionally high recoveries can be achieved on all grades and types of ore tested and that reasonable net (about 100# per ton) acid consumptions would be incurred in leaching each of the ore types. These recoveries and acid consumptions were obtained in short leaching and rinsing periods and in an efficient size plant. These factors should lend themselves to an efficient and low capital cost leaching- solvent extraction-precipitation plant.

A secondary rinse of the tailings on the face of the disposal heap would net an additional 3 to 4 % recovery over that obtained in the primary leaching cycle.

The one thing that could cause trouble in leaching is overwetting the ore in the acid-ore mixing or curing steps. It would be necessary to have good control over the amount of solution mixed with the ore in the initial stage of leaching and provide a plastic sheet for covering the curing piles in the case of a heavy rain.

It appears the the leach and rinse solutions can be recycled to give a clarified pregnant leach liquor of optimum acid, uranium, vanadium and molybdenum contents for the solvent extraction steps and also provide solutions for leaching that utilize the acid generated in solvent extraction and the ferric iron generated in leaching. It should be possible to conduct the solvent extraction steps in the storage resevoirs with these solutions.

About 40 % of the contained vanadium was dissolved along with the uranium. Although this metal would probably be more of a nuisance than an asset, it could be recovered as a separate product. More-than-likely however, it would be removed as a bleed stream product and evaporated down to a salt in a pond or discarded as a rinse solution in the final tailings.

The molybdenum content of the ore and the leach liquors was very low (about 1 part Mo in 30,000 parts U_3O_8). Hence, this troublesome metal should not contaminate the uranium product.

A large amount of ferric iron resulted in the leach liquors as a result of leaching iron oxides in the ore. If this iron can be utilized in leaching more ore it could materially reduce the acid consumption requirements.

This ore is comparatively very light in its weight.

It should also be pointed out that the optimum leaching conditions are not yet well established. Additional testwork would probably reduce the leaching and rinsing times and the acid consumptions without sacrificing the high level of recovery obtained in these tests.

Paul H. Johnson, PhD

TLAU 1   ORE: AU HS 1 (0.06% U_3O_8) ~17.5# - 1/2

1/22/73   Cure Soln: 100# H_2SO_4/Ton : 215 ml H_2SO_4 in 2324 ml Soln

29.3% Moisture Required

Rinse Soln: 10 gpl H_2SO_4

OPERATION	1 DATE	2 TIME	3 SM. NO.	4 ML. SM.	g/l U_3O_8	gms U_3O_8	g/l H_2SO_4	gms H_2SO_4	g/l Fe^T	Fe^{++}	gms Fe
MIX AND CURE	12/23	9 PM	CURE IN PLASTIC TUB					-410			
CURE	12/24										
CURE + LOAD	12/25	3PM	LOAD COLUMN - ~20% EXPANSION OF MATERIAL IN CO								
RINSE	12/25	5PM	Rinse with 10 GPL H_2SO_4 - Good Permeability.								
"	12/26	10AM	TLAU 1-1	1650*	1.50	2.48	18.0 Preg 10.0 Feed	+14.4	2.95	-0-	4.80
"	12/26	10 PM	TLAU 1-2	7820	NA		6.50 P 10.00 F	-27.4	.44	.11	3.44
"	12/27	8 PM	TLAU 1-3	7300	NA		6.92 P 10.00 F	-22.5	.14		1.02
OFF	12/28	10 PM	TLAU 1-4	6400	NA		6.30 P 10.00 F	-40.3	.09		.58
								485.8			9.90

* Soln TLAU 1-1 contained .590 g/l V_2O_5 + .00005 g/l Mo.

SOLIDS:	DESCRIPTION	gms	% U_3O_8	gms U_3O_8	% V_2O_5	gms V_2O_5
TAILS	TOP 2'	3168.5	.0042	.126		
TAILS	MIDDLE 1'-2'	1517.6	.0050	.076		
TAILS	BOTTOM 1-2'	1372.0	.0040	.058		
	TOTAL	6058.1	.0043	.260		
HEAD		CALC. 6510	.06	3.91	.055	3.585

U_3O_8 Recovery = $\frac{3.91 - .260}{3.91}$ = 93.6%

V_2O_5 Recovery in Soln 1-1 = $\frac{.973}{3.585}$ = 27%. $\frac{27}{63.5} \times 93.6$ = 39.4 ≈ V_2O_5 Re

Acid Consumption = $\frac{485.8\ (2000)}{6510}$ = 150#/TON

Column height at end of test = 52"

... WASTE MATERIAL .007 % U_3O_8 - 1/2

CURE SOL'N: 100# H_2SO_4 / TON SOLIDS. 30% MOISTURE

RINSE SOL'N: 10 gpl H_2SO_4 Soln.

COLUMN HEIGHT: 19 1/2" AT END OF TEST

OPERATION	DATE	TIME	SM. NO.	ML. SM.	g/l U_3O_8	gms. U_3O_8	Δ g/l H_2SO_4	Δ gms H_2SO_4 CONS	g Fe^T	Fe^{++}	gms. Fe^T
MIX + CURE	12/23	9 PM						125			
CURE	12/24										
CURE + LOAD	12/25	3 PM	- only part of solids put in column.								
Rinse	12/25	5 PM	start flow of solution				- Good Permeability				
OFF	12/28	10 AM	TLAU2-1	8000	.035	.280	7.1	56.7	.29	.04	2.32
			TOTAL			.280		181.7			

SOLIDS	DESCRIP.	gms	% U_3O_8	gms U_3O_8
TAILS	TOP 12"	1502	NA	
	Bottom 7 1/2"	760	NA	
HEADS		2360	.009	.212

U_3O_8 RECOVERY $= \frac{.280}{.212} \cong$ 95 - 100 %

ACID CONSUMPTION $= \frac{181.5}{2360} \times 2000 =$ 155 #/TON

EFFICIENCY® LINE No. 2536

TLAU 3 ORE: AUHS 3 (.045 % U_3O_8) -1/2 ~ 3000 gms.

CURE SOL'N: 100# H_2SO_4/TON + TLAU 1-1 Soln + 5 gms MnO_2 - 40m.
RINSE SOL'N: 10 gpl H_2SO_4 + 10 gpl in REVERSE RINSE*
COLUMN HEIGHT: 23" AT START

OPERATION	DATE	SM. NO.	ML. SM.	g/l U_3O_8	gms. U_3O_8	Δ g/l H_2SO_4	Δ gms. H_2SO_4 CONS	g/l Fe^T	gms Fe^T
Mix + Cure	1/3 8PM		CURE IN PLASTIC TUB						
Cure	1/4	RAIN WATER GOT IN PLASTIC TUB - OVERWET - NO AGGLOMERATION							
Rinse	1/5 8PM		POOR PERMEABILITY - LOW IN IRON.						
Rinse	1/6 3PM	TLAU 3-1*	2115	.35	.74			.96	
"	1/7 6PM	TLAU 3-2	5685	NA				.28	
"	1/7 9PM	TLAU 3-3	320	NA				.58	
	1/8 6PM	REVERSED COLUMN CONTENTS							
off	1/10	TLAU 3-4	4210	NA				.25	
	1/3	MIX SOLN	815.5	1.50	1.225				

* Soln TLAU 3-1 also contained .135 gpl V_2O_5 + .000015 g/l Mo.

			% U_3O_8		
TAILS		3211.5 gm	.065		MAY HAVE REVERSED HEADS + TAIL ASSAY
HEADS		3360	.045	1.51	
+	MIX SOLN			1.23	
	TOTAL			2.74	

U_3O_8 RECOVERY = VERY LOW BECAUSE OF OVERWETTING.
POOR ASSAYING OR MIXUP OF SAMPLES

ACID CONSUMPTION ≅ 120#/TON.

* TAILS AFTER TWO DAYS OF RINSING TURNED OVER + RE-RINSED.

TLAU 4   ORE: AU HS 4 $-1\frac{1}{2}''$ ~ 6.5# LOW GRADE MUDSTONE .0272% U_3O_8 (CALC.)

CURE SOLN: 150# H_2SO_4 / TON, - 30% MOISTURE

RINSE SOLN: 10 gpl H_2SO_4

COLUMN HEIGHT : 30" AT START

ITERATION	DATE	TIME	SM. NO.	ML. SM.	g/l U_3O_8	gms U_3O_8	g/l H_2SO_4	gms H_2SO_4 CONS.	g/l Fe^T	gms Fe^T
MIX + CURE	1/3/74	8PM		CURE IN	PLASTIC	TUB		225		
CURE	1/4									
RINSE	1/5	8PM	START	RINSING		GOOD	PERMEABILITY			
"	1/6	3PM	TLAU4-1	1715	.420	.740	F 10.00 P 30.0	+34.5	3.27	5.60
"	1/7	6PM	TLAU4-2	3815	.024	.092	F 10.00 P 5.90	15.6	.27	1.03
"	1/8	9PM	TLAU4-3	770	~.01	~.008	F 10.0 P 3.90	4.7	.11	.08
REVERSE	1/18	6PM	REVERSED	COLUMN		CONTENTS +		STARTED	RINSING	
RINSE	1/9									
Rinse	1/20	6PM	TLAU4-4	5875	.0053	.037	F 10.0 P 7.77	12.9	.16	.93
										7.64
		TOTAL				.883		223.8		

	gms	% U_3O_8	
TAILS	3108.4 gms	.0038	.012
HEADS	3108+5% = 3263	.0272 (CALC)	

$$U_3O_8 \text{ RECOVERY} = \frac{.883 - .012}{.883} = 98.7\%$$

$$\text{ACID CONSUMPTION: } \frac{223.8\,(2000)}{3200} = 140\#/\text{TON}$$

EFFICIENCY® LINE No. 2636

TLAU 5 ORE: AUHS 5 - 3/8 (SHALE ORE) 4000 GMS. (0.10% U_3O_8)

CURE SOL'N: 100 #/TON H_2SO_4 + 2 #/TON Fe^{+++} AS $Fe_2(SO_4)_3$ 29% Moistu[re]

RINSE SOL'N: 10 gpl H_2SO_4 + 2 gpl Fe^{+++}

COLUMN HT: 44" INITIALLY 39" FINAL

[OP]ERATION	DATE	TIME	BM. NO.	ML. SM.	g/l U_3O_8	gms. U_3O_8	g/l H_2SO_4	gms H_2SO_4 CONS.	g/l Fe^T	gms Fe^T
[M]ix	1/19/74	7PM						- 200		
[C]ure	1/20/74									
[C]ure [L]oad	1/21/74			Good agglomeration + Permeability						
Rinse	1/21/74	10 PM								
"	1/22	6 PM	TLAU5-1	8010			11.9	+ 15.2		
[Wa]sh off	1/23	6PM	TLAU5-2	5700			8.8	- 16.9		
[Wa]sh off	1/24	8PM	TLAU5-3	850			8.2	- 1.5		
[Re]verse	1/24	Bed Reversed + Rinse Re-started								
Rinse	1/25	[illegible]	~~TLAU 5-4~~	-0-			8.0			
[O]ff	1/27	1 PM	TLAU 5-5	9600			8.0	-19.2		
				TOTAL				-222.4		

	gms	% U_3O_8	gms U_3O_8
TAILS	3706	.0025	.093
HEADS	4000	.10	4.00

U_3O_8 RECOVERY $= \frac{4.00 - .093}{4.00} = 97.8\%$

ACID CONSUMPTION $\frac{222.4 (2000)}{4000} = 111$ #/ton

EFFICIENCY® LINE No. 2636

SKYLINE LABS, INC.
Hawley & Hawley, Assayers and Chemists Division
P.O. Box 50106, 1700 W. Grant Rd., Tucson, Arizona 85703

CERTIFICATE OF ANALYSIS

ITEM NO.	SAMPLE IDENTIFICATION	U_3O_8 %	U_3O_8 mg/l	V_2O_5 ppm	V_2O_5 mg/l	Mo mg/l
1	HSAU 1	0.061		550		
2	HSAU 2	0.0090				
3	HSAU 3	0.045				
4	TLAU 1 Bottom	0.0042				
5	1 Middle	0.005				
6	1 Top	0.0040				
7	3	0.065				
8	4	0.0038				
9	1-1		1500.		590	0.050
10	2-1		35.			
11	3-1		350.		135	0.015
12	4-1		420.			
13	4-2		24.			
14	TLAU 4-4		5.3			

TO:
Chemical Producers Corporation
104 East 40th Street
New York, N. Y. 10016
Attn.: Mr. George Ward

cc: Dr. Paul H. Johnson

I-40

REMARKS:

CERTIFIED BY: Edwin O. Bst

DATE REC'D: 1/16/74

DATE COMPL.: 2/5/74

JOB NUMBER: 740057

SKYLINE LABS, INC.
SPECIALISTS IN EXPLORATION GEOCHEMISTRY
12090 WEST 50TH PLACE • WHEAT RIDGE, COLORADO 80033 • TEL.: (303) 424-7718

REPORT OF ANALYSIS

Job No. M-2372
February 13, 1974

Mr. Paul H. Johnson
1532 West Chatea Avenue
Anaheim, California 92802

Analysis of 2 Rock Chip Samples

Item	Sample No.	U_3O_8 (%)
1.	TLAU 5 HS	.10
2.	TLAU 5 Tails	.0025

for Charles E. Thompson
Chief Chemist

CC: George Ward

APPENDIX II

Tonnage Factor

The following data regarding the tonnage factor conducted on selected core samples by Hazen Research Inc., Golden, Colorado summarizes all testing to date by Minerals Exploration Company:

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4501

March 22, 1976

Mr. G. E. Marrall
Minerals Exploration Company
P.O. Box 2674
Casper, Wyoming 82601

Re: HRI Project 1968
Anderson Mine-Tonnage Factors

Dear Gerry:

Here are the corrected tonnage factors on the five boxes of core. I suspect there was a gremlin in the calculator the first time.

Tonnage factors were measured on the top section of each two-foot interval of core. The values are listed below:

Hole No. AM-17 C

130'	2.05 SpG	15.6 ft^3/Ton
132	2.07	15.5
134	2.33	13.8
136	1.91	16.8
138	1.78	18.0
140	2.02	15.9
142	2.00	16.0
144	2.05	15.6
146	1.97	16.3
148	2.00	16.0

Hole No. AM-26-C

627	2.00 SpG	16.0 ft^3/Ton
629	2.07	15.5
631	2.10	15.3
633	2.14	15.0
635	2.19	14.6
637	2.24	14.3
639	2.44	13.1
641	2.00	16.0

Hole No. AM-135

460'	1.78 SpG	18.0 ft^3/Ton
462	1.99	16.1
464	2.09	15.3
466	2.15	14.9

Hope these will help your ore reserves.

Yours truly,

John E. Litz (mg)

John E. Litz
Project Manager

JEL:mhg

APPENDIX III

GEOLOGIC URANIUM RESERVES

By

J. E. Sherborne, Jr.

ANDERSON MINE GEOLOGIC RESERVES

An attempt was made to determine the geologic uranium reserves for three different cases on the Anderson Mine property. These cases were:

1) Maximum open pit depth of 200 feet with associated underground reserve.
2) Maximum open pit depth of 400 feet with associated underground reserve.
3) No open pit depth limitation with associated underground reserve.

The geologic reserve figures generated from these studies are not simply resources at various depths, but should approximate rough reserve estimates since reasonable economic parameters were used to arrive at these figures. For the open pit evaluation, the operating costs used to judge the merits of individual mineralized zones are shown in the following table:

Open Pit Cost Parameters

Cutoff grade	2.0 ft. @ 0.029% eU_3O_8
Tonnage factor	15.6 ft^3/ton
Primary stripping	$0.40/ton
Secondary stripping	$0.50/ton
Interior waste	$1.00/ton
Mining cost	$1.00/ton
Haulage	$0.20/ton
Milling cost	$6.00/ton
General & Admin.	$1.00/ton
Contingency	$0.50/ton
Mill recovery	90%
Mining recovery	100%
Product price	$40.00/lb. U_3O_8

The mineralization in each hole was evaluated in comparison to the above cost parameters in our UPR (unit profitability report) computer program. This analysis results in a value per ton of ore for each hole. Those holes showing a profit per ton of ore in a grouping of other profitable holes which could constitute an open pit configuration were considered as geologic reserves. By confining our geologic reserves to only that mineralization which could be extracted at a profit, it was felt that sufficient profit margin would be established to justify the capital costs necessary for this project. This procedure was used to calculate open pit geologic reserves for all three cases with the only variable, the maximum depth of the mineralization evaluated.

The mineralization which occurs at depths greater than maximum open pit was evaluated using a more rigid set of cost parameters so as to approximate an underground geologic reserve for each case. These underground parameters are shown in the following table:

<u>Underground Cost Parameters</u>

Cutoff grade	0.03% eU_3O_8
Minimum interval grade	0.048%
Minimum ore thickness	5.0 ft./interval
Mining cost	$18.00/ton ore
Interior waste	$18.00/ton
Milling cost	$ 6.00/ton
General & Admin.	$ 1.00/ton
Contingency	$ 0.50/ton
Mill recovery	90%
Mine recovery	80% of ore tons
Product price	$40.00/lb. U_3O_8

These cost parameters were used in the UPR computer program in a similar fashion to that used in the open pit evaluation.

Mineralized holes showing a profit in a contiguous area of profitable holes then constitute the underground geologic reserve.

The open pit and associated underground reserves for all three cases are shown on clear plastic overlays to plate 1. Geologic reserves for the three cases are shown in the following table:

Anderson Mine Geologic Reserves

	Ave. ore thick.(ft)	Ave. ore grade	Ave. GT prod.	Total tons ore (millions)	Recov. tons ore (millions)	Gross lb.eU_3O_8 (millions)	Recov. lb.eU_3O_8 (millions)
200' Open Pit	9.7	.060%	0.582	3.8	3.8	4.6	4.2
Associated Underground	8.4	.111%	0.933	4.2	3.4	9.3	6.7
			Totals	8.0	7.2	13.9	10.9
400' Open Pit	11.7	.065%	0.761	9.2	9.2	11.7	10.6
Associated Underground	8.1	.111%	0.900	2.7	2.2	6.1	4.4
			Totals	11.9	11.4	17.8	15.0
Unlimited Depth Pit	16.1	.063%	1.006	16.5	16.5	20.9	18.8
Associated Underground	6.4	.086%	0.551	0.5	0.4	0.8	0.6
			Totals	17.0	16.9	21.7	19.4

After the open pit configurations were determined for the three cases all of the NE-SW stick engineering sections were constructed. These sections attempted to show the correlation of the mineralization from hole to hole and to show the approximate configuration of the various open pit cases. An index for these sections is shown on plate 1.

APPENDIX IV

Semi-quantitative X-ray Flourescence Analysis

The following data conducted on our samples by Hazen Research Inc., Golden, Colorado summarizes all semi-quantitative X-ray flourescence analysis done by Mineral Exploration Company:

HAZEN RESEARCH, INC.

4601 INDIANA STREET
GOLDEN, COLORADO • 80401
TELEPHONE 303/279-4501

June 16, 1975

Mr. G. E. Marrall
Minerals Exploration Company
P. O. Box 2674
Casper, Wyoming 82601

Re: HRI Project 1757
Uranium Amenability Studies

Dear Mr. Marrall:

The two samples submitted by you on May 16, 1975, for uranium amenability studies were subjected to a semi-quantitative X-ray fluorescence analysis for minor elements. The results are as follows:

Element, %	AM-16c-1	AM-16c-2
Copper	0.009	0.010
Zinc	0.025	0.019
Lead	0.004	0.006
Arsenic	0.013	0.014
Iron	2.9	1.8
Cobalt	0.003	nd
Nickel	0.008	nd
Rubidium	0.025	0.016
Barium	0.070	0.035
Strontium	0.078	0.20
Titanium	0.15	0.068
Zirconium	0.029	0.046

Element, %	AM-16c-1	AM-16c-2
Vanadium	0.054	0.033
Molybdenum	0.010	0.021
Manganese	0.044	0.043
Yttrium	0.006	0.004

The above analyses indicate that the only element of value in the samples is the uranium, 0.101% U_3O_8 in Am-16c-1 and 0.052% U_3O_8 in AM-16c-2.

Yours truly,

John E. Litz

John E. Litz
Project Manager

JEL:mgp

AM 16C-1 sample is a combination of samples 4 through 13 assayed by Chemical & Geochemical Laboratories representing interval 294' through 299'.

AM 16C-2 sample is a combination of samples 44-47, 51, 55-57, 61, 62 assayed by Chemical & Geochemical Laboratories representing intervals 247.5'-249.5', 251'-251.5', 268.5'-270',288'-289'.

APPENDIX V

PRELIMINARY DISEQUILIBRIUM STUDY
FOR THE ANDERSON MINE
SEPTEMBER, 1976

By

T. S. Hellinger

Plates prepared by

J. R. Ljung

T. S. Hellinger

INTRODUCTION

The coring program at the Anderson Mine was initiated to resolve the relationship between the recorded subsurface gamma ray mineralization (eU_3O_8) and the actual chemical uranium content (cU_3O_8). Core hole locations were chosen from pre-existing drill holes which exhibited favorable gamma mineralization. These drill holes were offset approximately five feet and the anomalous zones were cored. To date, 15 core holes have been completed, with 14 core holes containing significant equivalent (eU_3O_8) and chemical (cU_3O_8) uranium mineralization.

Two Reid Drilling Company and one Universal Drilling Company rotary rigs were contracted to pull 925 feet of core, of which 94-95% was recovered. Various size core bits and core barrels were tried with the best recovery attained by Russel Sharpe of Reid Drilling Company using a three inch diameter core barrel set up. The core from each core run (usually ten feet) was carefully measured, labeled and boxed. The core was next described by a geologist using a 10X to 45X binocular scope, and finally shipped to the Casper office.

Upon receipt of the cores in Casper, they were split longitudinally and half of the core was dried and pulverized. Pulverized core, representing one-half or one foot intervals, was analyzed on the Blake Beta-Gamma scaler. Each interval was analyzed three times and an average was taken. Initially all samples with an average indicated analysis greater than .02% eU_3O_8 were sent to Chemical and Geological Laboratories in Casper for chemical and closed-can analyses. Subsequently, selected samples with an average beta-gamma analysis less than .02% U_3O_8 were sent out for chemical and closed-can analyses to fill in gaps in the assay intervals and to better delineate the ore zones (Table 1). A total of 448 core samples have been analyzed to date. In addition, 21 previously analyzed core samples, representing various grades of the mineralized lithologic units, were sent to Skyline Labs, Inc. in Wheat Ridge, Colorado for fluorimetric and closed-can uranium, chemical vanadium and spectrographic analyses (Table 4). Spectographic analyses were run to ascertain the presence of any element other than uranium which might constitute ore, or at least require consideration for secondary recovery during milling of the uranium ore (see Summary of Emission Spec. Results). Periodic cross check analyses were run on random samples throughout this study to verify the reproducibility of all the analyses.

DISEQUILIBRIUM AND CHEMICAL ASSAY RESULTS

Before a summary of the chemical analyses could be made and a subsequent disequilibrium factor computed, adjustments had to be made between the core assay footages and the digitized gamma log footage for each core hole. This adjustment was accomplished for each core hole by determining the best correlation between the closed-can gamma uranium assays and the digitized gamma log data. Plates 19 thru 33 graphically depict the relationship between eU_3O_8 and cU_3O_8. Only cored intervals with at least 2 feet of .03% eU_3O_8 from the gamma log, were considered in this disequilibrium study. The intervals in each core hole which met or exceeded this cutoff have been summarized in Table 2 along with all other analyses of the respective interval. A weighted average for each core was computed for eU_3O_8, cU_3O_8, V_2O_5, CO_2, and total sulfur by dividing the total thickness of all the intervals into the respective total grade thickness (Table 2). Disequilibrium was then computed for each hole by dividing the weighted average cU_3O_8 by the weighted average eU_3O_8.

Two methods were used to determine the uranium disequilibrium for the Anderson Mine property. The first method involved totaling the weighted eU_3O_8 and cU_3O_8 (Table 3), and then dividing the total cU_3O_8 by the total eU_3O_8:

$$\frac{\text{Total wt. } cU_3O_8}{\text{Total wt. } eU_3O_8} = \text{disequilibrium factor} \quad \frac{12.75 \; cU_3O_8 \text{ ft.}}{14.12 \; eU_3O_8 \text{ ft.}} \doteq .89$$

The first method yielded a disequilibrium factor of .89. The second method simply entailed dividing the total mineralized thickness into the total weighted disequilibrium for all of the core holes:

$$\frac{\text{Total wt. disequilibrium}}{\text{Total thickness}} \doteq \text{diseq. factor} \quad \frac{177.76 \text{ disequilibrium ft.}}{201 \text{ ft.}} = .88$$

This second method produced a disequilibrium factor of 0.88. The vanadium-uranium ratio (V_2O_5:cU_3O_8) of 1.39 was obtained by ratioing the appropriate weighted grade averages (Table 3). Average total CO_2 and sulfur were determined by dividing the total thickness (160 ft.) into the total weighted analyses of each. The average weighted grades for CO_2 and total sulfur were 6.38 wt. % and 0.57 wt. % respectively (Table 3).

SUMMARY OF EMISSION SPECTROGRAPH RESULTS

A rapid spectrographic scan of 21 uraniferous core samples was completed by Skyline Labs, Inc., Wheat Ridge, Colorado. The samples were selected to represent a cross section of the ore grades and mineralized lithologic units, recognized at the Anderson Mine. The spectrographic scan was run primarily for three reasons:

1. Identify elements other than uranium that might warrant consideration for secondary recovery during milling (i.e. V_2O_5).
2. Evaluate the concentration of those elements which might create milling problems (i.e. Mo).
3. Aid in geochemical exploration for similar uranium deposits in the Basin and Range area.

Before detailed evaluation of the data is made it should be pointed out that there were not enough samples analyzed to determine reasonable background values for all lithologic units. Therefore, average background values as tabulated by Turikian and Wedipohl (1961) were used when applicable.

The emission spectrographic scan provided semi-quantitative analysis of 31 elements. Elements which displayed the most interesting anomalies in at least one lithologic unit were: V, Mo, As, Co, Mn, and Sc (Table 4). Vanadium was the most pervasive anomally, present in all the mineralized units. Quantitative analyses (Table 1) indicate a high enough concentration of vanadium to at least warrant consideration for secondary recovery. The next most important anomalous element is molybdenum. Molybdenum was anomalous in seven samples, with significant concentration (50 ppm to 300 ppm) in three samples (Table 4). The molybdenum appears to represent rare isolated accumulations within the carbonaceous marls. However, because molybdenum can adversely affect milling of the uranium if an acid leach is used, further molybdenum analyses should be initiated to determine the actual concentration and distribution. The remaining elements; As, Cr, Mn, and Sc are of only minor importance as anomalous trace elements. None of these elements are concentrated enough to warrant secondary recovery or pervasive enough to be used as a pathfinder for uranium mineralization in other areas. However, the overall effects of these trace element concentrations with respect to milling is presently unknown. The lithologic unit with the most trace element anomalies is the sandstone (Table 4).

RECOMMENDATIONS

The disequilibrium factor and the weighted grade averages computed from the 14 mineralized cores are good first approximations. However, the rather complex lithology requires a greater detailed and more comprehensive coring program so that a better statistical evaluation for each mineralized unit can be made. Several units were not cored as frequently as they probably should be in the future. Disequilibrium for each mineralized lithologic unit has also not been computed due to the unequal distribution of the lithologic units cored. Our past experience in other areas indicates that the disequilibrium factor will probably improve as more coring is completed. More coring should provide a better statistical sampling of the area. Presently the disequilibrium factor neglects values with eU_3O_8 below the cutoff which also has correspondingly high cU_3O_8 (Example: Am-17c; 201 ft. - 202 ft., .076% cU_3O_8 vs. .011% eU_3O_8). Ranges of concentration for various elements via emission spectrograph should be determined for each lithologic unit. This may prove to be vital information for mill recovery. More quantitative work should be undertaken to better understand the concentration of at least U_3O_8, V_2O_5, CO_2, and Mo with respect to specific lithologies and areal distribution.

REFERENCES CITED:

Turekian, K. K. and Wedepohl, K. H., 1961, Distribution of the elements in some major units of the Earth's crust: G.S.A. Bull., v. 72, no. 2, pp. 175-191.

AM - 1C
Cored Interval
60'-140'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
60.5-61	0.056	0.071	0.076	-	-	sltstn.
63-64	0.003	0.010	0.034	-	-	"
64.0-64.5	0.008	0.023	0.021	-	-	"
64.5-65	0.021	0.015	0.036	-	-	"
65-65.5	0.029	0.037	0.044	-	-	calc.ls. & sltstns.
65.5-66	0.017	0.013	0.043	-	-	"
66-66.5	0.019	0.024	0.025	-	-	"
66.5-67	0.102	0.109	0.061	-	-	"
67-67.5	0.021	0.018	0.020	-	-	"
67.5-68	0.051	0.043	0.030	-	-	"
68-68.5	0.012	0.021	0.009	-	-	"
68.5-69	0.020	0.025	0.014	-	-	"
94.5-95	0.004	0.002	0.029	-	-	sltstn.
95-95.5	0.084	0.084	0.025	-	-	lignite
95.5-96	0.031	0.118	0.037	-	-	"
96-96.5	0.054	0.077	0.068	-	-	"
96.5-97	0.043	0.022	0.008	-	-	"
97-97.5	0.005	0.011	0.005	-	-	sltstn.
97.5-98	0.015	0.056	0.021	-	-	sltstn & mdstn.
98-98.5	0.004	0.011	0.005	-	-	"
98.5-99	0.007	0.014	0.020	-	-	"
99-99.5	0.013	0.054	0.021	-	-	"
99.5-100	0.021	0.075	0.036	-	-	"
100-100.5	0.214	0.170	0.133	-	-	sltstn.
100.5-101	0.121	0.121	0.110	-	-	"
101-101.5	0.012	0.038	0.050	-	-	"
101.5-102	0.009	0.015	0.049	-	-	"
103.5-104	0.016	0.030	0.125	-	-	"
104-104.5	0.098	0.096	0.168	-	-	lignite
104.5-105	0.040	0.028	0.023	-	-	"

*Closed can gamma only assay for eU_3O_8.

Table 1. Summary of core analyses from the Anderson Mine Property, September, 1976.

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
105-105.5	0.028	0.041	0.062	-	-	lignite
105.5-106	0.051	0.049	0.169	-	-	mdstn & slts
106-106.5	0.015	0.020	0.026	-	-	" "
106.5-107	0.013	0.021	0.020	-	-	" "
109-109.5	0.008	0.025	0.009	-	-	" "
109.5-110	0.037	0.021	0.026	-	-	" "
110-110.5	0.023	0.033	0.025	-	-	" "
110.5-111	0.028	0.026	0.026	-	-	" "
111-111.5	0.048	0.044	0.071	-	-	" "
111.5-112	0.029	0.036	0.039	-	-	lignite
112-112.5	0.106	0.113	0.035	-	-	"
112.5-113	0.048	0.110	0.041	-	-	"
113-113.5	0.016	0.021	0.034	-	-	"
113.5-114	0.017	0.021	0.088	-	-	sltstn
114-114.5	0.008	0.067	0.053	-	-	"
114.5-115	0.152	0.100	0.110	-	-	"
115-115.5	0.129	0.117	0.028	-	-	"
115.5-116	0.060	0.078	0.020	-	-	"
116-117	0.002	0.020	0.020	-	-	"

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 7C
Cored Intervals
15'-25'; 95'-118'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
17-18	0.009	0.018	0.128	0.32	0.25	mdstn
18-19	0.179	0.198	0.198	0.06	0.26	"
19-20	0.316	0.321	0.162	0	0.33	mdstn-sltstn
20-21	0.045	0.049	0.111	0	0.30	"
21-22	0.003	0.010	0.052	0.01	0.29	"
95-96	0.017	0.023	0.012	0	0.29	mdstn
96-97	0.001	0.024	0.018	-	-	"
97-98	0.009	0.020	0.018	0	0.31	"
98-99	0.016	0.034	0.015	0	0.30	"
99-100	0.011	0.029	0.012	0	0.26	sltstn
100-101	0.007	0.026	0.011	0	0.19	"
102-103	0.001	0.017	0.030	-	-	"
103-104	0.001	0.015	0.024	-	-	"
104-105	0.005	0.019	0.027	0	0.03	sltstn
105-106	0.010	0.017	0.055	-	-	"
106-107	0.005	0.019	0.037	-	-	"

*Closed can gamma only assay for eU_3O_8

Table 1. (Continued)

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
115-116	0.013	0.021	0.012	24.67	0.01	silty ls
116-117	0.006	0.008	0.008	-	-	ss
117-118	0.005	0.006	0.005	-	-	"
118-119	0.004	0.007	0.010	-	-	"
119-120	0.003	0.024	0.017	1.62	0.01	"
120-21	0.004	0.008	0.018	6.87	0.01	"
121-122	0.007	0.014	0.019	6.20	0.01	"
122-123	0.012	0.018	0.020	10.49	0.01	silty ls, cher
123-124	0.008	0.023	0.021	8.20	0.01	"
124-125	0.012	0.019	0.023	15.95	0.02	silty ls
125-126	0.008	0.019	0.026	14.84	0.01	" "
126-127	0.047	0.053	0.083	9.23	0.01	" "
127-128	0.018	0.025	0.029	11.45	0.01	" "
128-129	0.022	0.027	0.056	9.08	0.01	" "
129-130	0.001	0.007	0.004	16.62	0.01	ls & chert
130-131	0.007	0.012	0.006	26.22	0.01	" "
131-132	0.030	0.038	0.071	7.16	0.02	ls & mdstn
132-133	0.022	0.033	0.062	3.18	0.01	ss & mdstn
133-134	0.005	0.010	0.038	13.44	0.01	ls & mdstn
134-135	0.017	0.030	0.077	14.11	0.01	silty ls
135-136	0.024	0.031	0.059	14.77	0.01	" "
136-137	0.011	0.016	0.042	2.14	0.15	ss
137-138	0.024	0.028	0.062	9.67	0.01	"
138-139	0.023	0.031	0.107	[illegible]0.30	0.02	silicified ls
139-140	0.005	0.006	0.014	26.44	0.01	ls
140-141	0.013	0.015	0.014	18.31	0.01	ls & sltstn

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 16C
Cored Interval
240'-335'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
246.5-247	0.003	0.014	0.067	-	-	ss & mdstn
247-247.5	0.007	0.078	0.168	-		mdstn
247.5-248	0.014	0.058	0.048	-	-	"
248-248.5	0.055	0.051	0.107	-	-	"
248.5-249	0.078	0.103	0.099	-	-	"
249-249.5	0.425	0.444	0.257	-	-	mdstn & lignite
249.5-250	0.476	0.285	0.169	-	-	mdstn
250-250.5	0.447	0.416	0.159	-	-	"
250.5-251	0.181	0.122	0.278	-	-	mdstn & lignite
251-251.5	0.015	0.028	0.098	-	-	" "
251.5-252	0.003	0.008	0.115	-	-	sltstn
267.5-268	0.001	0.012	0.062	-	-	mdstn
268-268.5	0.004	0.014	0.030	-	-	"
268.5-269	0.141	0.150	0.109	-	-	lignite
269-269.5	0.125	0.102	0.071	-	-	mdstn
269.5-270	0.046	0.061	0.070	-	-	"
270-270.5	0.009	0.022	0.064	-	-	"
270.5-271	0.005	0.021	0.089	-	-	"
287.5-288	0.017	0.022	0.033	-	-	"
288-288.5	0.062	0.048	0.015	-	-	mdstn
288.5-289	0.033	0.053	0.031	-	-	"
289-289.5	0.005	0.020	0.050	-	-	"

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
292.5-293	0.007	0.015	0.164	-	-	sltstn
293-293.5	0.009	0.015	0.112	-	-	"
293.5-294	0.004	0.014	0.082	-	-	calc. mdstn
294-294.5	0.060	0.045	0.102	-	-	mdstn
294.5-295	0.074	0.066	0.151	-	-	lignite & mdstn
295-295.5	0.079	0.081	0.045	-	-	lignite
295.5-296	0.027	0.075	0.023	-	-	"
296-296.5	0.060	0.090	0.179	-	-	"
296.5-297	0.067	0.095	0.049	-	-	"
297-297.5	0.045	0.077	0.265	-	-	"
297.5-298	0.021	0.028	0.045	-	-	mdstn
298-298.5	0.027	0.035	0.023	-	-	sltstn
298.5-299	0.045	0.050	0.034	-	-	"
299-299.5	0.059	0.050	0.052	-	-	lignite & mdstn
299.5-300	0.042	0.051	0.102	-	-	" "
300-300.5	0.034	0.037	0.062	-	-	lignitic sltstn
300.5-301	0.025	0.025	0.031	-	-	lignitic sltstn
301-301.5	0.233	0.218	0.218	-	-	lignite
301.5-302	0.125	0.141	0.134	-	-	lignite & mdstn
302-302.5	0.022	0.025	0.023	-	-	calc mdstn
302.5-303	0.012	0.013	0.018	-	-	" "
303-303.5	0.013	0.014	0.015	-	-	" "
303.5-304	0.018	0.022	0.012	-	-	" "
304.5-305	0.019	0.040	0.071	-	-	lignite
305-305.5	0.071	0.054	0.054	-	-	"
305.5-306	0.027	0.035	0.015	-	-	mdstn
306-306.5	0.038	0.038	0.018	-	-	"
306.6-307	0.043	0.044	0.018	-	-	lignite
307-307.5	0.044	0.051	0.024	-	-	"
307.5-308	0.035	0.050	0.036	-	-	"
308-308.5	0.052	0.065	0.095	-	-	"
308.5-309	0.013	0.011	0.018	-	-	mdstn
309-309.5	0.015	0.020	0.035	-	-	"
309.5-310	0.016	0.014	0.033	-	-	"
310-310.5	0.050	0.036	0.045	-	-	"
310.5-311	0.039	0.045	0.033	-	-	"
311-311.5	0.039	0.037	0.030	-	-	"
311.5-312	0.038	0.031	0.042	-	-	"
312-312.5	0.012	0.011	0.052	-	-	"

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 17C
Cored Interval
100'-215'

Core Depth ft.	$_cU_3O_8$ % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
104-105	0.009	0.017	0.021	11.52	0.07	sltstn & Rs
105-106	0.004	0.014	0.070	0.40	0.02	sltstn
106-107	0.006	0.010	0.034	0.35	0.02	"
107-108	0.010	0.017	0.040	0.01	0.01	"
130-131	0.008	0.011	0.039	-	-	mdstn
131-132	0.040	0.037	0.046	0	0.02	"
132-133	0.012	0.007	0.034	0.07	0.02	"
133-134	0.030	0.026	0.035	11.49	0.02	"
134-135	0.008	0.013	0.016	-	-	"
135-136	0.007	0.010	0.008	-	-	"
136-137	0.028	0.037	0.018	0.07	0.01	"
137-138	0.006	0.008	0.008	-	-	"
138-139	0.017	0.025	0.015	0	0.01	"
139-140	0.010	0.020	0.021	0	0.02	"
140-141	0.009	0.013	0.061	0	0.02	sltstn
141-142	0.009	0.012	0.124	0	0.03	"
143-144	0.003	0.010	0.098	0	0.02	"
144-145	0.010	0.021	0.077	0	0.02	sltstn & mdst
146-147	0.011	0.024	0.249	0	0.01	sltstn
147-148	0.023	0.022	0.159	0	0.02	"
148-149	0.024	0.028	0.068	0	0.01	mdstn
149-150	0.039	0.039	0.092	0	0.01	mdstn
150-151	0.004	0.013	0.080	0	0.02	"
154-155	0.009	0.013	0.102	0	0.01	"
162-163	0.004	0.002	0.205	-	-	"
163-164	0.022	0.036	0.065	0	0.01	mdstn
164-165	0.004	0.011	0.031	-	-	"
165-166	0.024	0.023	0.146	0	0.01	"
166-167	0.005	0.005	0.026	-	-	"
192-193	0.012	0.011	0.039	-	-	"
193-194	0.017	0.020	0.130	0	0.04	mdstn
199-200	0.076	0.090	0.133	0	0.21	"
200-201	0.012	0.021	0.156	0	0.03	sltstn
202-203	0.003	0.008	0.049	0	0.01	mdstn
203-204	0.059	0.064	0.159	0.15	0.04	"
204-205	0.059	0.068	0.127	0	0.01	"
205-206	0.001	0.011	0.076	0	0.02	mdstn & sltst
198-199	0.003	0.007	0.079	-	-	mdstn

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 18C
Cored Interval
270'-320'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
278-279	0.007	0.011	0.021	- -	--	silty ls
279-280	0.055	0.063	0.181	23.12	0.56	" "
280-281	0.007	0.005	0.003	-	-	" "
281-282	0.095	0.115	0.205	22.82	0.73	" "
282-283	0.250	0.161	0.051	16.62	1.00	" "
283-284	0.042	0.035	0.015	22.23	0.88	" "
284-285	0.042	0.032	0.015	22.08	0.84	" "
285-286	0.019	0.025	0.009	21.56	0.77	" "
286-287	0.019	0.030	0.241	8.64	0.75	lignite
288-289	0.062	0.077	0.135	0.74	0.50	"
289-290	0.120	0.148	0.303	3.77	0.62	lignite & sltstn
290-291	0.038	0.040	0.083	6.94	0.75	sltstn & mdstn
291-292	0.005	0.005	0.024	-	-	" "
293-294	0.007	0.001	0.018	-	-	" "
294-295	0.059	0.058	0.036	13.07	0.89	" "
295-296	0.031	0.030	0.062	11.67	0.86	slstn & lignite
296-297	0.103	0.147	0.161	3.03	0.52	" "
297-298	0.058	0.059	0.003	12.48	0.44	" "
298-299	0.023	0.022	0.003	15.43	0.87	marl & lignite
299-300	0.012	0.008	0.025	-	-	" "
312-313	0.014	0.022	0.042	0.22	0.70	sltstn
313-314	0.029	0.017	0.074	0.15	0.63	"
314-315	0.007	0.006	0.049	-	-	"

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 26C
Cored Intervals
595'-649';705'-755'

Core Depth ft.	U308 % by wt.	eU308* % by wt.	V_2O_5 % by wt.	CO2 % by wt.	Total sulfur(s) % by wt.	Lithology
601-602	0.008	0.011	0.005			mdstn & sltst
602-603	0.024	0.029	0.302	8.24	0.67	slstn & lignite
603-604	0.032	0.027	0.072	3.72	0.62	" "
604-605	0.024	0.025	0.226	0.03	0.57	lignite
605-606	0.020	0.016	0.058	0.01	0.55	"
606-607	0.047	0.061	0.311	0.01	0.45	"
607-608	0.011	0.014	0.021	8.68	0.58	lignite & sltst
619-620	0.002	0.003	0.007	-	-	" "
620-621	0.011	0.021	0.013	-	-	" "
621-622	0.019	0.019	0.051	0.06	0.25	lignite, sltstn & mdstn
622-623	0.012	0.018	0.076	0.72	0.41	lignitic sltstn
623-624	0.009	0.010	0.045	4.95	1.57	" "
624-625	0.007	0.019	0.009	9.90	1.07	sltstn
625-626	0.043	0.048	0.003	15.88	0.27	silty ls
626-627	0.020	0.033	0.007	15.36	0.32	"
627-628	0.010	0.010	0.008	-	-	lignitic sltst
628-629	0.006	0.010	0.010	-	-	" "
629-630	0.014	0.026	0.016	9.16	0.70	" "
630-631	0.034	0.040	0.017	6.79	0.53	" "
631-632	0.034	0.070	0.009	22.01	0.18	silty, lignitic ls
632-633	0.008	0.006	0.026	-	-	" "
633-634	0.004	0.014	0.008	-	-	" "
634-635	0.106	0.108	0.008	36.93	0.25	" "
635-636	0.121	0.170	0.003	27.99	0.12	" "
636-637	0.248	0.175	0.015	26.00	0.62	ls & sltstn
637-638	0.017	0.057	0.001	29.91	0.27	" "
638-639	0.012	0.018	0.010	27.32	0.09	ls
639-640	0.052	0.048	0.010	18.09	0.50	"
640-641	0.008	0.015	0.010	-	-	ls & sltstn
718-719	0.004	0.005	0.012	-	-	sltstn & mdstn

*Closed can gamma only assay for eU308.

Table 1. (Continued)

AM - 26C (Con't)

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
719-720	0.008	0.019	0.015	-	-	mdstn
720-721	0.345	0.292	0.026	0.07	0.67	ss
721-721.3	0.067	0.081	0.071	0.71	0.08	"
722.5-723	0.007	0.009	0.071	-	-	"
723-724	0.022	0.038	0.054	0.68	0	ss & mdstn
725-726	0.006	0.012	0.013	-	-	sltstn
726-727	0.003	0.002	0.010	-	-	ss
733-734	0.003	0.003	0.010	-	-	sltstn
734-735	0.007	0.016	0.018	-	-	"
735-736	0.008	0.008	0.016	-	-	lignitic mdstn
736-737	0.007	0.011	0.016	-	-	" "
738-739	0.004	0.009	0.015	-	-	sltstn
740-741	0.003	0.002	0.008	-	-	marl
741-742	0.001	0.003	0.008	-	-	"
743-744	0.003	0.007	0.029	-	-	mdstn
737-738	0.034	0.037	0.086	0.09	1.74	"
739-740	0.045	0.029	0.089	30.06	0.19	slstn & marl
742-743	0.030	0.022	0.201	8.35	0.90	mdstn & ligni

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 49 C
Cored Interval
606'-650'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
609-610	0.007	0.013	0.063	-	-	mdstn
610-611	0.020	0.019	0.076	-	-	"
611-612	0.010	0.017	0.079	-	-	"
612-613	0.034	0.034	0.120	0.01	0.32	chert
613-614	0.008	0.027	0.155	1.99	0.36	mdstn&cherty ls
614-615	0.033	0.054	0.005	30.72	0.05	ls
615-616	0.058	0.051	0.039	26.51	0.01	"
616-617	0.061	0.058	0.026	25.33	0.04	"
617-618	0.077	0.091	0.026	25.63	0.01	"
618-619	0.040	0.051	0.048	6.65	0.55	ls&lignitic mdst
619-620	0.037	0.030	0.042	5.98	0.48	lignite & mdstn
620-621	0.032	0.036	0.196	0.22	0.63	mdstn
621-622	0.022	0.026	0.014	11.89	0.35	mdstn & ls
622-623	0.020	0.020	0.007	33.60	0.23	ls
624-625	0.024	0.024	0.015	2.73	0.98	mdstn
625-626	0.019	0.025	0.014	-	-	"
629-630	0.004	0.008	0.021	-	-	lignitic mdstn
630-631	0.019	0.027	0.053	0.06	0.63	lignite
631-632	0.114	0.089	0.336	0.09	0.73	"
632-633	0.016	0.054	0.039	0.22	1.38	"
633-634	0.020	0.027	0.049	0.65	0.94	mdstn
634-635	0.013	0.019	0.046	0.99	1.19	"
635-636	0.011	0.024	0.146	9.01	1.51	lignite ls & sltstn
636-637	0.078	0.046	0.089	2.29	2.07	mdstn
637-638	0.012	0.024	0.064	2.51	1.40	lignite & mdstn

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 51C
Cored Interval
377'-418';430-475'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
394-395	0.001	0.017	0.035	0	0.063	sltstn
395-396	0.004	0.016	0.038	0	0.037	"
396-397	0.040	0.036	0.064	0	0.429	sltstn & ss
397-398	0.013	0.020	0.016	0	0.386	" "
398-299	0.021	0.024	0.013	0.02	0.063	" "
399-400	0.008	0.023	0.013	0.01	0.316	mdstn
400-401	0.008	0.036	0.012	0.02	0.409	sltstn & ss
401-402	0.009	0.034	0.014	0.04	0.326	slstn
402-403	0.005	0.025	0.012	0.02	0.374	sltstn & mdst
403-404	0.012	0.025	0.025	0.04	0.506	lignitic slts
404-405	0.023	0.027	0.020	0	1.012	" "
405-406	0.011	0.031	0.0 0	0	0.848	sandy sltstn
406-407	0.011	0.030	0.017	0	1.386	mdstn
407-408	0.152	0.062	0.254	0.01	2.046	lignitic mdst
408-409	0.001	0.036	0.035	0	0.905	mdstn
409-410	0.005	0.017	0.020	0.10	0.848	"
410-411	0.010	0.032	0.045	0	1.402	"
411-412	0.007	0.026	0.067	0	0.094	"
412-413	0.001	0.007	0.057	0	0.334	"
438-439	0.001	0.011	0.121	3.63	0.027	sltstn
439-440	0.044	0.044	0.064	14.24	0.362	ls & mdstn
440-441	0.227	0.190	0.025	30.84	0.444	ls
441-442	0.063	0.116	0.018	23.25	0.516	"
442-443	0.020	0.028	0.029	14.02	0.227	ls & mdstn
443-444	0.011	0.823	0.089	7.81	0.721	mdstn
444-445	0.024	0.004	0.105	0.68	0.053	lignitic mds
445-446	0.075	0.096	0.199	0.17	0.053	" "
446-447	0.111	0.019	0.140	0.19	0.017	" "
447-448	0.001	0.005	0.200	0.11	0.651	" "
461-462	0.008	0.013	0.488	0.01	0.067	sltstn
462-463	0.007	0.017	0.281	0.02	0.035	"
463-464	0.011	0.018	0.303	0.01	0.060	lignitic sltst
464-465	0.259	0.280	0.312	0.01	0.569	" "
465-466	0.263	0.222	0.385	0.13	0.585	" "
466-467	0.845	0.567	0.382	0.02	0.711	sltstn
467-468	0.063	0.088	0.215	0.01	0.551	"
468-469	0.010	0.023	0.156	0.02	0.080	"
469-470	0.008	0.023	0.159	0.07	0.032	"
470-471	0.002	0.004	0.134	0.02	0.025	"

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 79C
Cored Interval
25'-70'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
40-41	0.001	0.007	0.008	-	-	mdstn
41-42	0.024	0.011	0.010	6.42	0.31	"
42-43	0.005	0.008	0.038	9.17	0.33	"
43-44	0.009	0.013	0.030	12.10	0.32	"
44-45	0.001	0.021	0.008	-	-	" "
45-46	0.022	0.026	0.044	12.36	0.11	"
46-47	0.035	0.037	0.012	32.35	0.32	mdstn & ls
47-48	0.021	0.024	0.008	31.68	0.08	" "
48-49	0.001	0.021	0.005	-	-	" "
49-50	0.001	0.006	0.003	-	-	" "
50-51	0.001	0.002	0.003	-	-	" "
58-59	0.010	0.010	0.018	30.28	0.13	silty ls
59-60	0.001	0.013	0.034	-	-	" "
60-61	0.020	0.021	0.060	23.48	0.07	"
61-62	0.004	0.009	0.018	-	-	sltstn
62-63	0.011	0.018	0.145	1.02	0.13	"
63-64	0.021	0.030	0.074	3.84	0.15	"
57-58	0.015	0.014	0.021	-	-	sltstn
64-65	0.008	0.014	0.011	-	-	ls
65-66	0.010	0.013	0.011	-	-	ls & sltstn

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 113C
Cored Interval
270'-345'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
272-273	0.013	0.012	0.036	7.75	1.75	lignite
273-274	0.028	0.022	0.030	0.44	2.17	"
274-275	0.005	0.015	0.026	-	-	"
275-276	0.007	0.010	0.029	-	-	mdstn
276-277	0.013	0.013	0.026	-	-	lignite
277-278	0.007	0.011	0.030	-	-	"
278-279	0.020	0.024	0.030	11.15	1.44	lignite, mdstn & ls
279-280	0.016	0.016	0.014	14.84	1.92	lignite,mdstn &ls
280-281	0.016	0.025	0.014	7.39	2.13	" " "
285-286	0.015	0.022	0.077	0	0.90	lignite
295-296	0.014	0.018	0.065	0	0.63	lignitic sltstn
296-297	0.004	0.012	0.132	0	0.47	" "
297-298	0.005	0.009	0.138	0	0.49	"
298-299	0.008	0.016	0.150	0	0.69	"
299-300	0.022	0.045	0.223	0	0.71	"
300-301	0.054	0.062	0.148	0	1.05	"
301-302	0.030	0.055	0.195	0	1.15	"
302-303	0.032	0.040	0.095	0	0.85	sltstn
303-304	0.004	0.016	0.024	-	-	"
316-317	0.004	0.011	0.029	-	-	"
317-318	0.070	0.036	0.036	0	1.69	lignitic sltstn
318-319	0.020	0.019	0.016	0	1.45	" "
319-320	0.012	0.017	0.021	0	1.36	" "
339-340	0.007	0.035	0.022	0	2.78	" "
340-341	0.156	0.117	0.036	0	0.47	" "
341-342	0.006	0.016	0.017	0	0.11	ss
342-343	0.008	0.022	0.021	0	0.13	"
343-344	0.057	0.076	0.028	0	1.42	lignitic mdstn
344-345	0.001	0.004	0.032	-	-	ss

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

AM - 119C
Cored Interval
26'-41';105'-135'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
30-31	0.001	0.005	0.055	-	-	mdstn
31-32	0.076	0.076	0.102	0	0.17	"
34-35	0.007	0.022	0.036	0	0.18	"
113.5-114	0.044	0.135	0.101	10.63	0.52	calc mdstn
114-115	0.017	0.029	0.011	20.86	0.99	ls & lignite
116-117	0.014	0.015	0.006	35.45	0.58	marl
118-119	0.016	0.017	0.012	26.81	0.45	"
119-120	0.090	0.083	0.030	27.03	0.93	"
120-121	0.045	0.040	0.065	31.24	0.63	marl & lignit
121-122	0.007	0.008	0.021	-	-	ls
122-123	0.007	0.008	0.024	-	-	ls & lignite
123-124	0.071	0.058	0.083	12.26	1.53	silicified ls & lignite
124-125	0.011	0.019	0.026	-	-	marl
130-131	0.007	0.012	0.029	-	-	
131-132	0.288	0.251	0.170	24.52	1.75	marl & lignite
132-133	0.194	0.118	0.077	0.37	1.85	lignite
133-134	0.011	0.020	0.027	0	2.13	lignite & sltst
117-118	0.015	0.018	0.017	-	-	marl

*Closed can gamma only assay for eU_3O_8.

Table 1. (Continued)

<u>AM - 135C</u>
Cored Interval
373'-399';452'-484'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
377-378	0.008	0.008	0.065	-	-	mdstn
378-379	0.027	0.027	0.095	0	0.47	"
382-383	0.020	0.017	0.077	0	0.96	"
383-384	0.024	0.025	0.030	0.34	1.00	"
384-385	0.028	0.038	0.027	0	1.10	lignitic mdstn
385-386	0.043	0.065	0.092	0	0.21	" "
386-387	0.135	0.085	0.123	0	2.22	" "
387-388	0.025	0.061	0.030	0.01	0.56	" "
388-389	0.033	0.038	0.062	0	0.67	" "
389-390	0.006	0.012	0.047	-	-	sltstn
457-458	0.011	0.033	0.344	0	0.21	lignitic sltstn
458-459	0.010	0.054	0.264	0.01	0.16	" "
459-460	0.033	0.051	0.456	0.01	0.33	" "
460-461	0.171	0.159	0.906	0.10	1.57	" "
461-462	0.176	0.183	0.373	1.09	0.51	" "
462-463	0.106	0.106	0.272	1.43	0.75	" "
463-464	0.090	0.122	0.064	4.61	2.42	" "
464-465	0.008	0.027	0.024	0.16	2.24	" "
465-466	0.006	0.010	0.018	-	-	" "
466-467	0.053	0.048	0.047	8.85	2.09	" "
467-468	0.138	0.125	0.117	6.99	2.24	" "
468-469	0.084	0.144	0.086	0.37	2.24	marl & lignite
469-470	0.045	0.052	0.056	28.06	0.21	marl
470-471	0.058	0.054	0.014	22.01	0.33	"
471-472	0.052	0.128	0.014	16.84	0.33	marl & lignite
472-473	0.361	0.370	0.067	15.58	0.21	" "
473-474	0.273	0.626	0.051	4.71	0.83	lignite sltstn
474-475	0.149	0.145	0.314	0.22	1.10	" "
475-476	0.059	0.051	0.071	0.01	2.09	" "
476-472	0.011	0.018	0.071	0.06	1.70	mdstn & sltstn
381-382	0.012	0.011	0.105	-	-	mdstn

*Closed can gamma only assay for eU_3O_8

Table 1. (Continued)

AM - 149C
Cored Interval(s)
340'-355';380-420'

Core Depth ft.	U_3O_8 % by wt.	eU_3O_8* % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur(s) % by wt.	Lithology
350-351	0.008	0.015	0.015	3.07	0.017	sltstn
351-352	0.005	0.013	0.022	3.38	0.133	"
352-353	0.001	0.004	0.076	0.17	0.007	"
380-381	0.008	0.008	0.064	0.07	0.498	"
381-382	0.014	0.022	0.016	0.20	0.534	"
382-383	0.061	0.076	0.012	0.14	0.658	"
383-384	0.072	0.081	0.014	0.13	0.215	"
384-385	0.032	0.048	0.011	0.38	0.075	"
385-386	0.022	0.028	0.013	0.26	0.316	"
386-387	0.012	0.027	0.015	0.06	0.848	"
387-388	0.003	0.015	0.017	0.04	1.216	"
388-389	0.011	0.013	0.021	0.04	1.009	"
389-390	0.012	0.012	0.024	0.01	0.852	"
390-391	0.009	0.011	0.041	0.01	0.892	"
391-392	0.003	0.013	0.070	0	1.169	"
392-393	0.016	0.045	0.099	0	1.921	"
393-394	0.050	0.050	0.209	0	0.159	lignite sltstn
394-395	0.039	0.028	0.143	0.02	0.077	" "
395-396	0.001	0.008	0.153	3.45	0.317	sltstn
396-397	0.001	0.016	0.163	0.10	0.094	"
397-398	0.039	0.033	0.571	0	0.939	"
398-399	0.009	0.018	0.153	0	0.885	"
399-400	0.011	0.016	0.115	0.01	1.216	"
400-401	0.005	0.021	0.131	0	0.721	"
401-402	0.014	0.017	0.184	0	0.631	sltstn & lignit
402-403	0.012	0.017	0.099	0	0.374	" "
403-404	0.005	0.007	0.077	0	0.090	mdstn
404-405	0.003	0.009	0.105	0	0.050	"
405-406	0.004	0.008	0.191	0.01	0.280	"
406-407	0.005	0.012	0.203	0.01	0.109	"
407-408	0.012	0.024	0.169	0	0.159	"
408-409	0.039	0.034	0.166	0	0.093	"
409-410	0.139	0.150	0.278	0	1.216	mdstn & lignite
410-411	0.032	0.036	0.080	0	1.979	sltstn & lignite
411-412	0.008	0.013	0.020	-	-	sltstn

*Closed can gamma only assay for eU_3O_8.

AM - 1c

Table 2. Core hole interval summary of assays from the Anderson Mine.

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
95.5-98.0	3.0	.061	.037	.029	-	-
100.0-102.5	3.0	.057	.065	.067	-	-
104.5-107.5	3.5	.042	.037	.085	-	-
111.5-116.5	5.5	.055	.058	.050	-	-
Total	15.0					
Weighted average		.054	.050	.057	-	-

cU_3O_8: eU_3O_8 = .926

AM - 7c

18.5-23.0	5.0	.095	.110	.130	.08	.34
98.0-99.5	2.0	.033	.014	.014	0	.28
Total	7.0					
Weighted average		.077	.083	.097	.06	.32

cU_3O_8: eU_3O_8 = 1.072

AM - 13c

125.5-138.5	13.5	.040	.019	.054	10.57	.02
Total	13.5					
Weighted average		.040	.019	.054	10.57	.02

cU_3O_8: eU_3O_8 = .475

AM - 16c

249.0-254.0	5.5	.135	.155	.142	-	-
273.5-276.0	3.0	.080	.055	.072	-	-
298.5-315.5	17.5	.052	.045	.060	-	-
Total	26.0					
Weighted average		.073	.069	.079		

cU_3O_8: eU_3O_8 = .945

AM - 17c

Table 2. (Continued)

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
149.0-150.5	2.0	.033	.032	.080	0	.01
204.5-207.0	3.0	.040	.040	.112	.03	.02
Total	5.0					
Weighted average		.038	.037	.099	.02	.02

cU_3O_8: eU_3O_8 = .974

AM - 18c

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
281.5-285.5	4.5	.051	.090	.059	21.06	0.84
289.5-293.0	4.0	.081	.060	.191	5.02	0.66
295.5-301.5	6.5	.065	.048	.048	11.14	0.72
Total	15.0					
Weighted average		.065	.063	.089	12.48	0.74

cU_3O_8: eU_3O_8: .938

AM - 26c

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
627.5-629.0	2.0	.042	.031	.005	15.62	0.30
631.5-640.0	9.0	.082	.064	.011	24.38	0.27
722.5-725.0	3.0	.111	.139	.037	0.39	0.38
Total	14.0					
Weighted average		.083	.075	.016	17.99	0.30

cU_3O_8: eU_3O_8 = .969

AM - 49c

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
615.0-622.5	8.0	.055	.045	.069	16.87	0.31
632.5-639.5	7.5	.038	.039	.108	1.90	1.21
Total	15.5					
Weighted average		.047	.042	.088	9.63	0.75

cU_3O_8: eU_3O_8 = .894

Table 2. (Continued)

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
396.0-401.0	5.5	.033	.018	.024	.01	0.32
403.0-411.5	9.0	.038	.026	.055	.02	1.01
442.5-450.0	8.0	.087	.072	.084	11.40	0.30
466.0-471.0	5.5	.205	.288	.319	0.04	0.50
Total	28.0					
Weighted average		.084	.089	.109	3.27	0.52

cU_3O_8: eU_3O_8 = 1.060

AM - 79c

45.5-47.5	2.5	.037	.029	.028	22.36	0.22
Total	2.5					
Weighted average		.037	.029	.028	22.36	0.22

cU_3O_8: eU_3O_8 = .784

AM - 113c

301.5-306.0	5.0	.049	.029	.150	0	0.89
342.5-347.5	5.5	.061	.047	.026	0	0.43
Total	10.5					
Weighted average		.055	.039	.085	0	0.65

cU_3O_8: eU_3O_8 = .709

AM - 119c

122.0-127.0	5.5	.045	.044	.045	23.51	1.03
132.0-134.5	3.0	.123	.163	.092	12.28	1.80
Total	8.5					
Weighted average		.073	.086	.062	19.55	1.30

cU_3O_8: eU_3O_8 = 1.178

AM - 135c

Table 2. (Continued)

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
386.0-391.0	5.5	.053	.053	.067	0.002	0.95
458.0-465.0	7.5	.115	.085	.337	1.06	1.14
468.0-477.5	10.0	.175	.127	.084	10.36	1.17
Total	23.0					
Weighted average		.126	.096	.162	4.85	1.11

$cU_3O_8 : eU_3O_8 = .762$

AM - 149c

Log depth ft.	Thickness ft.	eU_3O_8 % by wt.	cU_3O_8 % by wt.	V_2O_5 % by wt.	CO_2 % by wt.	Total sulfur % by wt.
382.5-388.0	6.0	.050	.036	.014	0.19	0.44
393.5-396.0	3.0	.044	.035	.150	0.01	0.72
398.0-399.5	2.0	.034	.024	.362	0	0.91
407.0-413.0	6.5	.061	.039	.066	0.002	0.59
Total	17.5					
Weighted average		.051	.036	.096	0.07	0.60

$cU_3O_8 : eU_3O_8 = .706$

Table 3. Calculation summary sheet

Hole No.	Thickness (ft.)	Average eU_3O_8 (wt.%)	W A (w	Weighted Average CO_2 (wt.% ft.)	Average total sulfur (wt.% ft)	Weighted Average total sulfur (wt.% ft.)
AM-1c	15.0	.054		-	-	-
AM-7c	7.0	.077		0.42	0.32	2.24
AM-13c	13.5	.040		142.70	0.02	0.27
AM-16c	26.0	.073		-	-	-
AM-17c	5.0	.038		0.10	0.02	0.10
AM-18c	15.0	.065		187.20	0.74	11.10
AM-26c	14.0	.083		251.86	0.30	4.20
AM-49c	15.5	.047		11.30	0.75	11.63
AM-51c	28.0	.084		91.56	0.52	14.56
AM-79c	2.5	.037		55.90	0.22	0.55
AM-113c	10.5	.055		0	0.65	0
AM-119c	8.5	.073		166.18	1.30	11.05
AM-135c	23.0	.126		111.55	1.11	25.53
AM-149c	17.5	.051		1.23	0.60	10.50
Total	201.0					
Weighted Totals				1020.00		91.73
Weighted Grade Average				6.38		.057

Disequilibrium Method

Disequilibrium Method

Table 3

Table 4. Summary of Emission Spectrographic Analyses, Anderson Mine, September, 1976.

Hole # / Core Depth / Lithology	AM-135c 460-461 lignite & sltstn	AM-135c 469-470 marl	AM-135c 472-473 marl & lignite	AM-119c 119-120 marl	AM-119c 132-133 lignite	AM-113c 300-301 lignite & sltstn	AM-7c 18-19 mdstn	AM-17 131-1 mdst
U_3O_8	.771	.045	.361	.090	.194	.054	.179	.040
V_2O_5	.906	.056	.067	.030	.097	.148	.198	.046
Fe	1.5%	.2%	1.5%	.5%	2%	2%	2%	2
Ca	.5%	15%	10%	20%	.7%	.2%	.3%	
Mg	.5%	.2%	.2%	1%	1%	.3%	1.5%	
Ag	<1	<1	<1	<1	<1	<1	<1	
As	<500	<500	500	<500	<500	<500	<500	<5
B	20	10	15	15	20	20	20	30
Ba	200	10	7	150	300	200	200	100
e	<2	<2	<2	<2	<2	<2	<2	2
Bi	<10	<10	<10	<10	<10	<10	<10	<10
Cd	<50	<50	<50	<50	<50	<50	<50	<50
Co	<5	<5	<5	<5	10	5	7	10
Cr	20	10	70	10	30	50	70	50
Cu	10	2	15	2	20	20	20	30
Ga	<10	<10	<10	<10	<10	<10	<10	<10
Ge	<20	<20	<20	<20	<20	<20	<20	<20
La	20	20	20	20	50	30	50	70
Mn	50	150	150	200	100	150	150	200
Mo	5	70 ●	300 ●	2	50 ●	30 ●	<2	<2
Nb	<20	<20	<20	<20	<20	<20	<20	20
Ni	15	<5	30	5	20	30	20	20
Pb	10	<10	10	10	20	15	20	10
Sb	<100	<100	<100	<100	<100	<100	<100	<100
Sc	<10	<10	<10	<10	10	<10	10	15
Sn	<10	<10	<10	<10	<10	<10	<10	<10
Sr	150	700	300	2,000	200	100	150	150
Ti	500	200	200	500	1,000	700	1,500	1,500
V	5,000 ●	200	500 ●	150	500 ●	700 ●	1,000 ●	200
W	<50	<50	<50	<50	<50	<50	<50	<50
Y	10	<10	<10	<10	15	10	20	20
Zn	<200	<200	<200	<200	<200	<200	<200	<200
	50	30	<20	30	50	50	50	50

● Anomalous value

Table 4. Summary of Emission Spectrographic Analyses, Anderson Mine, September, 1976.

	.039 .166	.095 .199	.111 .140	.259 .312	.845 .567
Hole #	AM-149c	AM-51c	AM-51c	AM-51c	AM-51c
Core Depth	408-409	445-446	446-447	464-465	466-467
Lithology	mdstn	lignite & mdstn	lignite & mdstn	lignite & sltstn	sltstn
Fe	1%	1%	1%	2%	1.5%
Ca	.3%	.5%	.2%	.2%	.2%
Mg	.3%	.5%	.5%	.2%	.5%
Ag	<1	<1	<1	<1	<1
As	<500	<500	<500	<500	700 ●
B	20	10	15	15	15
Ba	150	100	150	200	100
Be	<2	<2	<2	<2	<2
Bi	<10	<10	<10	<10	<10
Cd	<50	<50	<50	<50	<50
Co	<5	<5	<5	<5	5
Cr	70	30	50	50	30
Cu	20	10	7	20	15
Ga	<10	<10	<10	<10	<10
Ge	<20	<20	<20	<20	<20
La	30	30	20	20	20
Mn	70	50	50	50	100
Mo	15 ●	<2	<2	10 ●	<2
Nb	<20	20	<20	<20	<20
Ni	15	5	5	10	5
Pb	10	<10	<10	<10	10
Sb	<100	<100	<100	<100	<100
Sc	<10	<10	<10	<10	<10
Sn	<10	<10	<10	<10	<10
Sr	150	100	150	200	150
Ti	500	500	500	300	300
V	700 ●	500 ●	700 ●	1,000 ●	1,500 ●
W	<50	<50	<50	<50	<50
Y	15	10	<10	10	10
Zn	<200	<200	<200	<200	<200
Zr	30	70	20	20	20

● Anomalous value

Table 4. Summary of Emission Spectrographic Analyses, Anderson Mine, September, 1976.

	.058 .003	.248 .045	.345 .026	.067 .071	.022 .054	.034 .120	.058 .039	.050 .209
Hole #	AM-18c	AM-26c	AM-26c	AM-26c	AM-26c	AM-49c	AM-49c	AM-14
Core Depth	297-298	636-637	720-721	721-721.25	723-724	612-613	615-616	393-3
Lithology	lignite	ls & slstn	ss	ss	ss	chert	cherty ls	lignite
Fe	1.5%	.5%	3%	2%	5%	2%	.2%	1%
Ca	10%	15%	1%	1%	1.5%	.5%	10%	.2%
Mg	.5%	.2%	1.5%	1%	1.5%	.5%	.3%	.2%
Ag	<1	<1	<1	<1	<1	<1	<1	<1
As	<500	<500	500	<500	<500	<500	<500	<500
B	15	<10	20	30	15	15	<10	50
Ba	100	20	700	700	700	100	10	50
Be	<2	<2	2	2	2	<2	<2	<2
Bi	<10	<10	<10	<10	<10	<10	<10	<10
Cd	<50	<50	<50	<50	<50	<50	<50	<50
Co	<5	<5	15	5	15	<5	<5	<5
Cr	50	15	100 ●	50	100 ●	70	<10	150
Cu	20	5	50	30	30	20	2	10
Ga	<10	<10	10	<10	10	<10	<10	<10
Ge	<20	<20	<20	<20	<20	<20	<20	<20
La	20	20	50	50	50	30	20	30
Mn	500	300	200 ●	200 ●	200 ●	70	500	20
Mo	15 ●	<2	<2	<2	2	<2	<2	20
Nb	<20	<20	20	<20	20	<20	<20	<20
Ni	15	5	50	20	70	15	5	10
Pb	10	<10	30	20	20	10	<10	<10
Sb	<100	<100	<100	<100	<100	<100	<100	<100
Sc	<10	<10	15 ●	10 ●	20 ●	<10	<10	<10
Sn	<10	<10	<10	<10	<10	<10	<10	<10
Sr	1,000	200	1,000	1,000	1,000	70	500	100
Ti	500	100	2,000	1,000	3,000	500	100	500
V	50	30	200 ●	300 ●	300 ●	1,000 ●	100	700 ●
W	<50	<50	<50	<50	<50	<50	<50	<50
Y	<10	<10	30	10	20	20	<10	10
Zn	<200	<200	<200	<200	<200	<200	<200	<200
Zr	20	20	50	30	50	30	20	50

● Anomalous value