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AMOCO

1982 SUMMARY REPORT

GILT EDGE PROJECT, SOUTH DAKOTA

July, 1982

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SUMMARY

The Gilt Edge Mining District, located about 5 miles southeast of the Homestake gold mine and Lead, South Dakota, has produced gold irregularly since the late 1800's. In 1974, Cyprus Mines acquired a large land position from the Commonwealth Mining Company under an 80 percent Cyprus and 20 percent Congdon and Carey joint venture agreement. The partnership has continued to acquire property and currently holds 2 placer claims, 95 patented lode claims and 26 unpatented claims.

Exploration efforts since 1975 have developed extensive data on gold mineralization and distribution at Gilt Edge. A total of 214 shallow rotary drill holes and 47 diamond drill holes have been completed on the property. Exploration activity including drilling is currently underway.

Several studies have been made to define shallow, open-pit mining reserves and an update study is currently being done. Both manual and computer techniques have defined in excess of 8.5 million tons of mining reserves averaging close to .05 ounces of gold per ton and a 3-to-1 waste-to-ore stripping ratio. Current exploration activity is partly focused on expanding open-pit reserves in an untested adjacent area to the north, as well as continuing to drill for deeper, higher-grade gold mineralization.

Exploration efforts begun in 1980 and executed principally in 1981 and 1982 were designed to test for deeper, higher grade gold mineralization to be mined by underground methods. Amoco scoping studies in 1980 indicated that additional tonnage of higher grade gold would be necessary for economic development of the Gilt Edge property by Amoco. A gold value of $400 was used in the study.

An area near the southern limit of the shallow gold deposit, which also lies adjacent to the southernmost rhyolite stock, has been a target for deeper, higher-grade gold mineralization. This "cabin" area is partially drilled and is currently being explored by deep drilling. Several drill holes have intersected broad intervals where significant gold values have been obtained. This potential ore zone contains disseminated mineralization consisting of clots of pyrite, fluorite, hematite and pervasive silica alteration. This deeper deposit averages over 0.10 ounce per ton gold and may exceed 10 million tons.

Detailed core logging of the 1981 and earlier drill core has provided a basis for predicting where additional gold exploration might be successful. The best indicators for gold at Gilt Edge are intrusive breccias, crackle breccias and igneo-fragmental breccias that have been subjected to pervasive silica flooding and display a significant increase in pyrite and fluorite mineralization. No native gold has been seen in Gilt Edge core though it can be panned from the nearby creeks. Certainly, multiple siliceous porphyries intruded the host Precambrian schists and Cambrian quartzite units in the Gilt Edge area and gold mineralization is associated with one or more of these intrusive phases.

Continuing exploration at Gilt Edge will develop a poorly defined northern rhyolite stock which lies within Amoco's claim holdings. Scarce surface outcroppings contain gold that assays up to one ounce per ton. A modest, shallow rotary drilling program will evaluate this area for additional open-pit reserves. Deeper gold may exist there too, as at the "cabin" area now being explored. Geologic mapping and some geophysics are planned for this 1982 summer season.

Several metallurgical studies have been done at Gilt Edge and several comprehensive final reports are now being prepared. All but one study was done by Kappes, Cassiday and Associates of Reno, Nevada. A cyanidation study has recently been completed by the Cyprus Metallurgical Processes Corporation in Tucson.

The cyanidation study involved leaching tests on agitated ore pulps, leaching of flotation concentrates and leaching of roasted concentrates. Agitated leaching yielded 75 to 80 percent of the gold content and leaching of flotation concentrates yielded a net of 75 percent. Roasted concentrates gave significantly greater recovery of 97 percent of the gold.

The Kappes-Cassiday work concentrated on examining Gilt Edge ores for their amenability to heap leaching. The Sunday Zone and Dakota Maid Zone are both part of the shallow, open-pit reserve, well defined by rotary drilling. Oxidized ores from these zones are heap leachable with recoveries averaging 70 percent of the contained gold when the ore is crushed to 2 inches or less.

The heap leach behavior of unoxidized ore is highly variable and at best test results were 50 percent gold recovery. These conclusions are based on 90 bucket leach tests, four 40-foot column leach tests (25 ton samples), 500 cyanide bottle roll tests and one 1,700 ton field leach test.

Total expenditures on the Gilt Edge Project is $2.609 million since 1975, shared 80 percent by Amoco (Cyprus) and 20 percent by Congdon and Carey. Exploration costs peaked at $1.1 million in 1981 and have diminished to $581,200 for 1982. A proposed 1983 budget will be less than this current year expenditure.

INTRODUCTION

INTRODUCTION

This report updates the status of the Cyprus Gilt Edge gold project in Lawrence County, South Dakota and includes preliminary information on some of the on-going 1982 exploration activities. Included herein are several individualized summary reports for ore reserve calculations and metallurgical studies. At present several studies are in progress and summary reports have not been completed. They are therefore not included. Whitney and Whitney, Inc. of Reno are utilizing computer techniques to update the geologic gold reserves at Gilt Edge by including all of the 1981 drilling data. None of the Whitney and Whitney information is included. Kappes and Cassiday, metallurgical consultants in Reno, have provided a short summary report covering three separate studies related to heap-leaching shallow oxide and mixed oxide-sulfide ore at Gilt Edge. The three studies involve bottle roll tests, a 40-foot column leach test and a 1,700 ton trial heap study. Separate detailed tests for each of these studies are in preparation.

Detailed geologic logs for all of the 1981 drilling are contained in the appendix of this report. They are the basis for evaluating the total geologic system at Gilt Edge and for providing an exploration guide. Also included in this section are all of the assay results per drill hole for the 1981 drilling.

LOCATION AND ACCESS

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The Gilt Edge project is located in Lawrence County, South Dakota, about 5 miles southeast of Lead and the Homestake Mine. The project area is within Black Hills National Forest in sections 5, 6, 7, and 8, T.4N., R.4E. (Index Map).

The property may be reached by U.S. Highway 385 south from Lead, and thence along a gravel road 4.5 miles toward the old town of Galena. The Gilt Edge property is located near Galena at the headwaters of Strawberry Creek. Four-wheel-drive vehicles are necessary for all-weather access. Elevations on the property range from 5,000 feet along Strawberry Creek to 5,600 feet on top of Gilt Edge Hill. Water flows in lower Strawberry Creek most of the year, and water is found within 250 feet of the surface in mine workings. The entire area is moderately timber covered.

HISTORY AND PREVIOUS WORK

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General History

The Gilt Edge project area comprises the bulk of the Gilt Edge mining district. Most of the individual workings on the property were developed as small, independent gold mines during the late 1800's and early 1900's. Modest production of some 13,000 ounces of gold from at least five different mines is recorded prior to about the turn of the century. The Gilt Edge property currently consists of four different parcels of land with different histories: the Commonwealth Mining Company property, the Northwestern Metals Company land, the Kosel claims (Magdalena Waggoner Estate), and claims staked by Cyprus (Figure 1). An up-to-date property map is located at the end of this report.

Most of the old Gilt Edge mines are located on the property now owned by Commonwealth. Major producers were the Rattlesnake Jack mine and the overlying Sunday workings, the Dakota Maid Pit and the underlying King workings and Gilt Edge Incline. In the earliest years of mining, all of the workings were operated as small independent mines. The production records prior to 1900 show 1,050 ounces of gold was produced from 5,000 tons of ore at the Oro Fino mine. Oro Fino ore occurred at the contact zone between Precambrian rocks and Gilt Edge porphyries The mine shaft collar is near the Strawberry Creek channel and the mine flooded in 1894. The Golden Crest mine, now called the Commonwealth mine, lies about one mile west of the Dakota Maid pit. It produced 4,840 ounces of gold prior to 1906 from the basal Cambrian Deadwood Formation and from Tertiary dikes.

The mines in the central Gilt Edge area were consolidated into the Gilt Edge Maid Mining Company in 1900 and produced approximately 40,000 ounces of gold and 20,000 ounces of silver between 1905 and 1916. The mining company remained idle until 1935. From 1937 to 1940, it produced approximately 20,000 ounces of gold-silver bullion, chiefly from the Rattlesnake-Sunday workings. Written reports indicate that the mine was shut down in 1941 because of War Order L-208, but local people who worked in the mine at that time say it was shut down in 1940 because of metallurgical problems.

The Commonwealth Mining Company purchased all of the above properties and the Strawberry Creek Placer in the late 1940's but never operated any of them. Commonwealth holdings total 75 patented lode claims, one patented placer claim, and 17 unpatented lode claims (see land map).

The Northwestern Metals property consists of 16 claims of the once-extensive Branch Mint Mining Company. Prior to 1912, the Branch Mint claim group was larger and more productive than the Gilt Edge, with a large mill and a spur track connecting the mine to the main railroad line.

The 16 Northwestern claims include the Hoodoo, Union Hill, and Eureka workings. The Hoodoo and Union Hill mines connected, crossing the northern extension of the Rattlesnake-Sunday rhyolite dike-breccia system in the

Tertiary intrusive. Approximately 7,000 ounces of gold were produced in these workings prior to 1900. It is not known whether this figure includes production from the Eureka workings.

Four claims once held by the Kosel family include the portals of the Ora Bella (Rattlesnake Jack extension) and the Hoodoo workings, but no major production has been obtained from the area within the claim boundaries.

In 1967, a major portion of the project area was explored by the partnership of Congdon and Carey as a copper-molybdenum target. Eleven core holes totalling 9,960 feet were drilled, reaching a maximum depth of 1,500 feet. Discouraging results forced abandonment of the project in 1971.

Amoco (Cyprus) Program

Property Acquisition - In 1974, Congdon and Carey reacquired the Commonwealth Mining Company portion of the Gilt Edge property, and they promoted it to Azcon Corporation (Consolidated Goldfields), who referred the prospect to Cyprus Mines under the terms of an existing exploration agreement. The resulting joint venture between Cyprus and Congdon and Carey has been in effect since 1975 to develop the gold potential of Gilt Edge. Azcon participated in the joint venture during 1976 and 1977, but terminated their interest under the terms of another Cyprus-Azcon business arrangement.

Sixteen claims and fractions owned by the Northwestern Metals Company were acquired in 1976. Several claims owned by the estate of Magdalena Waggoner (Kosel Claims) were optioned in 1978, and have been wholly owned by the joint venture partners since 1981 (Figure 1). During mid-1982, nine lode claims or fractions were staked by Cyprus to help fill-in gaps within the claim block, and to extend control in areas believed favorable for additional mineralization.

Shallow Exploration - From 1975 to 1980, Cyprus, as operator for the joint venture, explored shallow gold mineralization at Gilt Edge, anticipating development of an open-pit, heap-leach operation. During this period, some 214 shallow rotary holes totalling 52,050 feet were drilled on a roughly 100- to 150-foot grid. Maximum hole depth is 350 feet. Sixteen core holes totalling 9,366 feet and varying in depth from 355 to 1,150 feet were drilled to aid delineation of the shallow gold deposit. This exploration program has defined a significant low-grade gold deposit. Actual reserve estimates based on various cut-off grade and pit design assumptions are summarized in the Ore Deposits section of this report.

Deep Exploration - The 1981 drilling program was designed to search for deeper, higher-grade gold mineralization that could be developed by underground mining. Results of 18 core holes totalling 19,740 feet indicate that substantial tonnages of previously undefined gold mineralization exist in at least one of three separate target areas.

The three target areas tested at Gilt Edge during 1981 are briefly as follows: 1) The deeper continuation of the Dakota Maid shear zone: Ten core holes totalling 10,855 feet encountered numerous narrow zones exceeding 0.1 ounce of gold per ton, but do not develop any significant tonnage. Potential of the deep Dakota Maid zone is probably less than a million tons. 2) The general vicinity of the old Sunday pit: Four diamond core holes totalling 4,170 feet encountered sporadic 10- to 20-foot intervals that exceed 0.1 ounce per ton, but no significant tonnage was developed. 3) The "cabin site" adjacent to a deep Congdon and Carey gold intercept from 1968: Four diamond core holes totalling 4,715 feet tested and partly delineated gold mineralization exceeding 0.1 ounce per ton over intervals of as much as 500 feet. Geologic reserves of roughly 2 million tons may be inferred in this area, with the extent of mineralization essentially undefined in all directions. Additional drilling is expected to increase this tonnage substantially.

Continuing exploration in 1982 involves deep drilling exploration in the cabin site area, continued mapping in the northern stock area, prior to shallow rotary drilling to test bedrock for anomalous gold, and magnetometer work in the Anchor Hill area. In May, 1982, JEN claims 1-7 were staked on open ground just to the north of Anchor Hill. Two other claims, JEN 9 and JEN 10 were staked in the southern stock area to cover open claim fractions (see claim map at the end of this report).

At writing of this report, two of three holes have been drilled in the southern stock-cabin area and land negotiations are underway to acquire the Blackhawk claim and 75 percent of the Crown Point claim (25 percent is already owned by Cyprus).

Metallurgical Testing - Cyprus has conducted relatively extensive metallurgical studies at the Gilt Edge property since its acquisition in 1974. This work is summarized in three broad stages. 1) Early Characterization of Ore: Studies of this period identified different types of mineralized rock and summarized their gold content and typical gold recovery. 2) Heap Leach Testing: Leaching experiments to demonstrate the viability of heap leaching Gilt Edge material were progressively scaled up to a 1,700 ton pilot heap facility. Recovery of gold from the pilot heap was within expected limits, and the safety and reclamation of cyanide heap-leach projects was demonstrated to state authorities. 3) Conventional Mill Testing: In line with the change of exploration to underground, various studies are now in progress to determine the behavior of Gilt Edge material in a conventional (agitation) milling circuit. Flotation studies have produced high-sulphide concentrates, which are now being tested further to determine leaching characteristics.

Three final reports are yet to be received from the metallurgical engineering consultant, Kappes, Cassiday & Associates. They include results of the heap leach testing, results of the 40-foot column leach testing and results of the bottle roll testing.

GEOLOGIC SETTING

GEOLOGIC SETTING

Regional Geology

The Gilt Edge project area is located in the northern part of the Black Hills, a Laramide domal uplift that exposes Precambrian metamorphic and plutonic rocks rimmed by tilted Paleozoic and Mesozoic sedimentary rocks. Superimposed as a northwest-trending belt upon the northern Black Hills is a Cenozoic igneous province 12 miles wide by 75 miles long. These silicic intrusives contain precious metals at Gilt Edge and in other areas along the belt. The Gilt Edge area is one of thirteen recognized intrusive centers within this province and all appear to have been emplaced at fairly shallow depths. Compositionally, they are alkalic with strongly undersaturated varieties in the western three-fourths of the belt and silicasaturated rocks in the eastern quarter, which includes Gilt Edge.

Geology of the Project Area

The project area is located in the western part of the Gilt Edge-Galena intrusive complex. Detailed geologic mapping by Cyprus shows Precambrian quartzites and schists unconformably overlain by Cambrian flat-lying to gently dipping Deadwood Formation sandstones and shales. Younger Paleozoic sedimentary rocks are largely eroded. This sequence has been intruded by a complex of Tertiary stocks, sills, and other bodies that include rhyolite, trachyte, diorite and more mafic rock types.

The earliest Tertiary intrusion is a 200-foot-thick sill of hornblende diorite porphyry. Later is a sequence of trachytic to rhyolitic porphyritic intrusions. The earliest of these occur as elongate masses and dikes that parallel a northeast structural grain. Detailed logging of drill holes indicates that a highly altered trachytic rock found only as breccia in deep drill holes seems to contain the plus-0.1 ounce gold values.

Three small stocks of sanidine rhyolite porphyry intrude the earlier porphyries. The central stock is roughly circular and 1,000 feet in diameter; the Dakota Maid and Sunday workings are located around its western and southeastern contacts, respectively. The southern stock is elongate east-west, measuring 800 feet by 1,400 feet. It is centered some 1,400 feet south of the Dakota Maid pit. Very little is known of the northern body because mapping was terminated in this area by snowfall in November, 1981. As of the writing of this report, mapping of the northern stock area is underway. Alteration and gold mineralization are associated with all three stocks.

The Gilt Edge area was remapped in 1981 in order to clarify the geology indicated by earlier workers. A regional thesis map (1"=1000') in use for many years did not show sufficient detail, and an earlier (1980) detail map (1"=1000') did not cover sufficient area and utilized a confusing subdivision of rock types. The re-mapping was assisted by P. E. Chapman, consulting geologist, and field samples were carefully

collected to provide type specimens to aid detailed core logging. The central Gilt Edge area, covering 1/4 x 3/4 mile, was re-mapped at 1 inch-to-100 feet, providing adequate control for deep diamond-drilling exploration of the central and southern sanidine rhyolite porphyry stocks. A 1 inch-to-200 feet geologic map, covering 1 mile by 1 1/4 miles, was also made, extending detailed geologic control in all directions from the larger scale map area. This second map partly defines a third sanidine rhyolite stock in the Anchor Hill area where further exploration efforts are now underway.

This recent mapping effort has greatly assisted the rock identification of the drill core and provided data to define possible gold controls at Gilt Edge. Deep drilling in 1981 extended Amoco's knowledge of the deeper portions of the complex Gilt Edge intrusive center. Detailed foot-by-foot drill logs for the 1981 drilling are a part of this report.

Breccias are abundant at Gilt Edge, and intrusion breccias predominate. Rhyolitic and trachytic intrusions have incorporated locally significant volumes of pre-existing rock types into their mass. Other breccias have formed as crackle breccias that have been filled by secondary minerals, commonly pyrite, purple fluorite and hematite. Crackle breccias probably were created by physical emplacement of one or more of the hypabyssal intrusive units that may or may not have been a source of gold mineralization. Surface sampling shows anomalous base- and precious-metal mineralization in some breccia zones.

One result of the detailed core-logging program started in 1982 is the recognition of a rock type comprised of highly altered and silicified quartz-free porphyry (trachyte porphyry?) embedded in matrix of fine-grained and altered igneous material. Clasts and matrix are not easily distinguished, and the entire rock has a jumbled, igneo-fragmental appearance. The rock is pyritized and locally intensely fluoritized in addition to the intense silicification and quartz veining. This rock unit has been identified only in the southern stock-cabin area, and seems to carry high gold values somewhat consistently. Partial assay results have identified 80 feet of 0.116 ounce per ton material in the first 1982 drill hole.

Basal Deadwood Formation rocks typically contain anomalous values in the Gilt Edge area and throughout the region. Precambrian metamorphic rocks typically are not anomalous at Gilt Edge except near the old Oro Fino Mine.

Ore Deposits

Shallow Gold Mineralization - Exploration efforts at Gilt Edge since 1975 have developed extensive data on the distribution of shallow gold mineralization within and adjacent to the central sanidine rhyolite porphyry stock. Geologic controls for this mineralization are vague, and perhaps masked by alteration and intense surface oxidation. Gold was emplaced with one or more of the several generations of pyrite and is most abundant in more intensely pyritized and silicified shear zones. Oxidation assisted the distribution of very fine grained gold

throughout a broad area where it is now contained in residual clay and silica-rich rock. Actual reserve estimates based on various cut-off grades and pit designed are summarized in the table in the section on Ore Deposits. At present it is not known if similar low-grade gold deposits are associated with the other two untested sanidine rhyolite stocks in the Gilt Edge area.

The central stock gold occurs in two distinct zones about 900 feet apart and is contained wholly in Tertiary porphyry rocks. In the western Dakota Maid area, gold occurs with varying amounts of copper and silver associated with silica and pyrite alteration. This silica-pyrite zone is 200 feet wide by 600 feet long and trends roughly north-south. Significant gold mineralization occurs to 900 feet depth but assays from deeper angle drilling showed little potential for much additional tonnage. This drilling also showed that the trachyte porphyry (Ttp) is the gold host and the sanidine rhyolite porphyry (primary Tsrp) unit is generally barren of any significant primary gold mineralization.

The second gold zone of the central stock area is the Rattlesnake-Sunday zone. Here, gold occurs in relatively unsilicified, pyrite-rich breccia zones and fractures adjacent to a system of steeply dipping rhyolite dikes trending N 40°-45°E. The dike and fracture system occurs in a fine to medium grained rhyolite porphyry (Trp) near the contact with the sanidine rhyolite porphyry (Tsrp). Much of this brecciated, pyrite-rich ore is oxidized and leaches readily, as defined by metallurgical studies. The dike itself is siliceous and leaches poorly. A zone of uneconomic copper enrichment is in mine workings only a couple hundred feet deep.

Some shallow rotary drilling has tested the northern part of the southern sanidine rhyolite stock for oxidized and partly oxidized, low grade gold mineralization. Results of the few completed holes are inconclusive. Additional shallow rotary drilling is necessary to fully test this area. Surface prospects and rock chip analysis show significant mineralization that remains untested by drilling.

Narrow, limited rock outcrops are all that is exposed of the northern sanidine rhyolite stock and there too, surface sampling and the occurence of prospect pits suggest low grade shallow gold mineralization may exist. Further evaluation of these areas are currently underway at this time.

Deep Gold Mineralization - Deeper, higher grade gold mineralization was confirmed at Gilt Edge by the 1981 core drilling effort. Numerous intercepts exceeding 0.1 ounce gold were encountered in three separate areas but only one area shows promise for sufficient tonnage to warrant further exploration. A tabulation of 1981 drilling for each of the three areas is shown in the following table. Drilling in 1982 is confirming and extending this type of mineralization.

A bulk gold deposit with grades in excess of 0.1 ounce gold appears to have been intersected by drilling in the "cabin site" near the northwestern margin of the southern sanidine rhyolite porphyry stock. Substantial intercepts in two drill holes, DDH 6 and DDH 36, suggest a geologic

Gilt Edge 1981 Drilling Assays using a 0.064 ounce gold per ton cut-off.

Hole No.	Interval	Footage	Aver. Au o.p.t. Fire Assay
Deeper Mineralization beneath Shallow Dakota Maid and Rattlesnake Zone.			
DDH-28	560-570	10	.491
	1060-1090	30	.155
	1240-1260	20	.145
	1730-1754	24	.450
	1790-1810	20	.151
DDH-29	170-200	30	.116
	490-500	10	.104
DDH-30	400-410	10	1.097
DDH-32	610-650	40	.124
DDH-33	470-480	10	.144
Contact Zone between Central and Southern Rhyolite Stocks.			
DDH-35	310-330	20	.113
	410-420	10	.103
DDH-36	930-1190	260	.106
	1250-1360	110	.103
	1630-1640	10	.165
DDH-37	1080-1090	10	.199
DDH-38	650-660	10	.281
Shallow Oxide Zone in East Part of the Central Rhyolite Stock.			
DDH-39	540-550	10	.125
	870-890	20	.117
DDH-40	80-90	10	.164
DDH-41	160-210	50	.108
DDH-42	550-560	10	.089
DDH-44	200-220	20	.107
	250-260	10	.430
	300-320	20	.125
DDH-45	290-333	43	.178

reserve of greater than 2 million tons with the deposit open in several directions. Gold mineralization occurs with pyrite and silica alteration possibly related to fine-grained, late trachyte porphyry dikes. Some high grade gold is also associated with open-void crackle breccia that has been filled with coarse pyrite, purple fluorite and hematite. At the writing of this report a 1982 core hole DDH 46 has intersected well silicified and pyritized zones. Partial assay results include an 80-foot zone averaging 0.116 ounce per ton. Assays for the rest of the hole and for subsequent holes are pending. If results are favorable a much larger tonnage than 2 million tons can be inferred to exist in this deeper zone; perhaps a deposit of 10 million tons is indicated.

Thin, high-grade intercepts generally 20 feet or less in thickness, were encountered in the deeper continuation of the Dakota Maid shear zone. Only a very steeply dipping, lensoid rock mass of less than 1 million tons grading 0.1 ounce gold or better was found. Over a dozen core holes have sufficiently tested this target area and very little potential exists for discovery of additional tonnages.

About eight core holes have been drilled in the deep portions of the Sunday Pit area. Again only a few narrow zones of about 0.1 ounce gold were discovered and potential for large tonnages is limited.

Target Concept and Exploration Potential

Two types of gold deposits warrant exploration at Gilt Edge. Shallow low-grade gold, believed to form by oxidation of auriferous pyrite has been defined in the central sanidine rhyolite stock area. Disseminated gold with grades in excess of 0.1 ounce per ton have been recently discovered in the deeper portions of the intrusive complex.

Shallow oxide ore may be found by rotary drilling both north and south of the known Dakota Maid-Sunday reserves, where surface geochemical gold anomalies occur near the margins of the northern and southern sanidine rhyolite stocks. These areas have little rock outcrop, and significant tonnages may exist beneath shallow soil cover. Mineable reserves of 5 to 10 million tons averaging 0.05 ounces per ton may be located in addition to the known mineable reserve of 8.5 million tons of 0.049 ounce gold in the Dakota Maid-Sunday deposit.

Deeper, bulk gold deposits with grades in excess of 0.1 ounce per ton are related to highly altered trachytic breccias. A deposit of this type was inferred from the results of the 1981 drilling program in the southern stock-cabin area, and the 1982 drilling thus far completed confirms the significance of the discovery. Surface mapping and detailed core logging suggests that the area near the southern stock known to contain similar rock units is capable of containing a gold deposit in excess of 10 million tons averaging better than 0.1 ounce per ton. Additional examination of the poorly known northern stock and reevaluation of data from the central stock area in terms of this new discovery may suggest similar geologic situations in these areas as well.

Ore Reserves

Several independent studies of the shallow ore reserves at Gilt Edge have been undertaken. In the following table, each study is identified with various tonnage, average and cut-off grades and stripping factors defined. The text of each study is contained in this section.

It must be noted that all of the ore reserve calculations done by Goddard and by Computing Associates in Spokane utilized only 1980 and earlier drilling data. Whitney and Whitney, Inc. of Reno have been retained to generate geologic reserves for the Gilt Edge project utilizing all data through 1981. Their work is currently underway and their final report will be attached to this summary report.

Whitney and Whitney have received from Computing Associates all of the drill hole data, variograms and interpolation parameters to assist their current studies. It is anticipated that they will develop similar numbers for open pit geologic reserves as Computing Associates, where drill hole density is sufficient to use krigging techniques. However, the deep drilling in 1981 probably will not permit the utilization of krigging for deeper gold reserves and some other technique may be used.

Gilt Edge Project Reserve Estimates
(based on drilling data through 1980 only)

Type of Estimate	Cut-off Grade (oz/ton)	Tons (MM)	Avg. Grade (oz/ton)	Waste to Ore Stripping Ratio
Geologic - In Place Computing Assoc. 12/81	0.020	19.71	0.047	-
	0.025	14.36	0.056	-
	0.030	10.83	0.065	-
	0.035	8.74	0.073	-
	0.040	6.37	0.086	-
	0.045	5.21	0.095	-
	0.050	4.04	0.109	-
	0.055	3.46	0.119	-
Geologic, cross section method, Goddard et al, 10/81	0.020	11.63	0.051	-
	0.030	8.66	0.061	-
	0.040	6.15	0.072	-
	0.050	4.08	0.086	-
	0.060	3.17	0.096	-
Geologic, polygon method Goddard et al, 10/81	0.020	11.62	0.051	-
	0.030	8.66	0.061	-
	0.040	6.15	0.072	-
	0.050	4.08	0.086	-
	0.060	3.17	0.096	-
Mineable, 0.020 pit design, Computing Assoc., 12/81. Using 0.020 oz. gold for pit base	0.020	11.32	0.042	1.97 to 1
	0.025	8.52	0.049	2.95 to 1
	0.030	6.38	0.055	4.27 to 1
	0.035	5.03	0.062	6.26 to 1
	0.040	3.55	0.072	-
	0.045	2.86	0.079	-
	0.050	2.15	0.089	-
	0.055	1.81	0.096	-
Mineable, cross section method, Goddard et al, 10/81	0.020	8.03	0.052	1.70 to 1
Mineable, polygon method Goddard et al, 10/81	0.020	8.80	0.046	1.70 to 1

MODELLING OF GILT EDGE PROJECT

by

COMPUTING ASSOCIATES
INTERNATIONAL

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AMOCO MINERALS

MODELLING OF GILT EDGE PROJECT

Feb. 1982

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MODELLING OF GILT EDGE PROJECT

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INTRODUCTION

To aid Amoco Minerals in their evaluation of the Gilt Edge deposit, Lawrence County, South Dakota, a computer model and mineral inventory analysis was carried out. Geological, analytical, drill hole survey, and topographic data were coded and stored in computer files. The analytical data consists of gold assays from one or more of six assay laboratories. These data then became the basis for the development of the computer model which simulated the chemical and physical characteristics of the Gilt Edge deposit and lead ultimately to geological reserves and mineable reserves. Three mineable reserves were calculated by expanding pit bottoms. The elevation of the pit bottoms were governed by the lowest level where the potentially leachable oxide and mixed oxide material existed in mineable amounts. The configuration of the pit bottoms were governed by the 0.015, 0.020, and 0.025 ounce gold contour intervals on the lowest level that a reasonable mining size could be contoured.

DATA BASE

A total of 7,082 assay records were loaded into a sequential data base. Each assay record consists of assay length, down-the-hole survey coordinates, the intensity of the oxidization, the rock type and the gold assay values from a number of laboratories.

Statistical comparisons were carried out on the assay values from the various laboratories. While variations can be seen visually between labs for a given assay interval, the average and standard deviation of the assay values from the various laboratories are very similar. Therefore, all assay

values for a given assay interval were averaged together to form an average gold value. This average gold value was then used in the model. Once a single gold value per assay interval was achieved, the drill holes were composited into 20 foot intervals which approximate a mineable bench height. A total of 3,615 composites were created.

Each assay interval was assigned an oxide code of either 1, 2, or 3 depending upon the degree of oxidization. If the interval has predominantly sulfide mineralization, a 1 was assigned. If the interval has mixed oxide and sulfide, a 2 was assigned or if the interval has predominantly oxide, a 3 was assigned. Along with the oxide code each different lithology from the drill logs was assigned a value. Interlayered trachyte and sediments were assigned a lithology code of 10, trachyte and rhyolite porphyry a 20, bimodal rhyolite a 30 and dikes a 40.

GEOSTATISTICAL ANALYSIS

Frequency Distribution Analysis

Histograms were computed for the four rock types. Initially the 7,082 assay file records were used and subsequently the 3,615 composited values were analyzed. The 20 foot composites are considered to be more representative of the Gilt Edge deposit because they "average out" erratic assay values inherent in the assay file. Rock type 40 was considered to be insignificant because there are only six composites of rock type 40.

Rock Type	N	$\overline{X}$ oz/ton	σ oz/ton[1]
10	1,062	0.0263	0.0651
20	1,602	0.0218	0.0419
30	848	0.0111	0.0255
	3,512		

[1] N = number of samples
$\overline{X}$ = mean value in ounces per ton
σ = standard deviation in ounces per ton

The discrepancy between the total composite records in the data base and those used in the frequency distribution can be attributed to composite intervals where no assays were available and the omission of composites from rock type 40.

Variograms

Horizontal variograms were calculated in four directions N-S, E-W, N45^{o}E and N45^{o}W for rock types 10 and 20 using the composite assay file. Vertical variograms were calculated also for rock types 10 and 20 using both the assay file and the composite file. Rock types 10 and 20 are statistically and geologically similar and are treated as a single ore bearing unit. Consequently they were the units used in the variogram analysis. The maximum gold value used in the variogram analysis was 0.10 ounces. Approximately 98 percent of all composited assay values are less than 0.10 ounces; therefore, by eliminating these erratic high values, the variograms more closely approximate the vast majority of the composited assay values at the Gilt Edge deposit.

A spherical model was fitted to the variograms. This model has a 270 foot horizontal range in all directions and a vertical range of approximately 135 feet.

Model Verification

The spherical model that was fitted to the deposit was verified by point kriging. Point kriging consists of estimating the grade of a particular composite by using the surrounding composites. The estimated grade is then compared to the actual assayed grade. This routine is done for every composite. As a result of the point kriging, it was determined that the variogram model for the deposit is suitable for kriging the gold values.

COMPUTER MODEL

The computer mine model was designed using blocks 25 feet on a side and a 20 foot bench height. The model boundary extends from 1,125,500 E. to 1,128,100 E. and from 744,000 N. to 746,500 N. The topography was first loaded into the model and checked with a plotted contour map.

A geological model was constructed using rock type 30 to represent the barren rock unit. A code of 60 was used in the model to represent the combination of rock types 10 and 20, the auriferous units. It was believed there was no benefit, in terms of the results, to digitize and load units 10 and 20 separately as they would ultimately be treated as equals during grade interpolation and assignment. The oxide codes were interpolated throughout the model using a "half-way to the next drill hole" type of interpolation. This type of integer interpolation was carried out on each of the 100 levels that comprise the model with those drill holes that penetrate a particular level assigning the integer value. Following the oxide interpolation, material code 60, the combination of rock types 10 and 20 was modified to reflect the degree of oxidization that exists in a particular block within the model. The result is block codes for rock types of 61, 62, and 63 respectively representing sulfides, mixed oxides and sulfide, and oxide. Through this method it was possible to report reserves by degree of oxidization while still considering the lithological characteristics of the Gilt Edge deposit (Figure 1).

Block Grade Assignments

Using kriging parameters developed from the point kriging, linear kriging was performed to estimate gold values for blocks having a material type code of 61, 62, or 63.

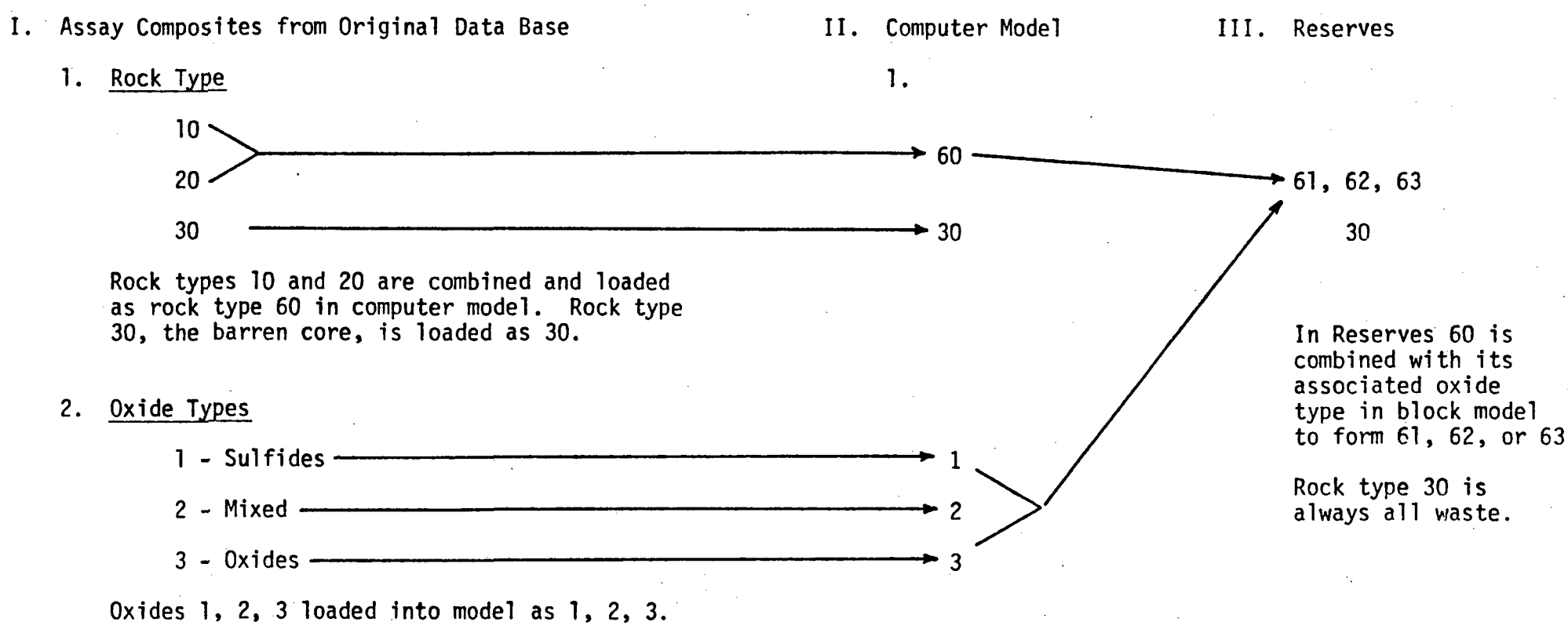

Figure 1. Development of material codes used in the computer model.

For a composited gold assay to be used in kriging it had to meet certain requirements. Only composites from the data base of rock types 10 and 20 could be used for grade assignments to model material type 60's. Furthermore, oxide type 1 blocks, as assigned by the oxide interpolation, could only receive estimated gold values from an assay composite whose oxide value was also 1. The same procedure was used for types 2 and 3. Grades assigned to oxide types 2 or 3 could only receive assay composite values from 2 or 3. Also for a block to receive an estimated value the block center must be within 400 feet of two drill holes. The interpolation distance used for assigning block values within the drilled area was 200 feet. An extrapolation distance for assigning block values around the periphery of the deposit was 50 feet.

GEOLOGICAL RESERVES

Geological reserves were calculated for material types 61, 62, 63 using 27 different cut-off grades. The amount of material type 30, which will always have a grade of zero, was also reported (Table 1). Also calculated was the kriging standard deviation for each block that was assigned an estimated grade. The standard deviations are displayed on the level maps along with their associated grade and material type.

MINEABLE RESERVES

Subsequent to delimiting the geological reserves, mineable reserves were calculated for three different pit bottom cut-off grades, 0.015, 0.020 and 0.025 ounces of gold respectively.

Initially it was intended to outline an economic pit limit by using a two dimensional Lerchs-Grossman approach. This method determines the positive or negative dollar value, based on mining and extraction costs,

of each block through a given cross section. The value of the block was predicated upon the following cost information supplied by Amoco Minerals:

Cost of mining ore or waste	$2.80 per ton
General and Administrative	1.00 per ton
Crushing, Leaching and Stripping	3.80 per ton

The recovery factors applied were 52 percent for oxide type 1, the sulfide material; 62 percent for oxide type 2, the mixed sulfide and oxide; and 72 percent for oxide type 3, the highly oxidized material. A gold price of $500 per ounce was used.

The shallow occurrence of the mineralization coupled with erratic higher grade zones within the deposit and the examination of the economics in only two dimensions resulted in an unrealistic pit outline because of the inability to project pit bottoms from section to section. The 2-D Lerchs-Grossman method would probably be more successful in outlining reasonable pit limits in a deposit where the grades are more evenly distributed thus permitting reasonable projections of pit bottoms from one cross-section to the next.

The positive aspect of the Lerchs-Grossman analysis is that it did point out approximately the lowest elevation that one could reasonably expect to mine. With this elevation in mind, cumulative geological ore reserves were determined from the surface down. The results show that all of the mixed oxide material, type 2, and the oxide material, type 3, using a grade cut-off of 0.015 ounces occurs above level 33, or 5040 feet in elevation. Following this the gold grades were contoured from the computer model. These level maps began at the 5100 elevation and extended to the 5500 elevation level in 100 foot increments. These maps provided the bases for hand drawing a pit outline for the lower levels of a pit using a 0.015, 0.020

and a 0.025 ounce contour cut-off for a pit outline. These pit outlines were then digitized, loaded into the computer and expanded to the surface at a 45 degree angle. Tonnage and grade were then calculated within the expanded area.

A tonnage and grade curve is shown on Figure 2 as well as the resulting reserves on Tables 2 through 4.

RECOMMENDATIONS

Prior to estimating a true economic leachable reserve for the Gilt Edge deposit, it is necessary to ascertain the actual cyanide recoverability of the gold with the various rock and oxide types found in the Gilt Edge deposit. With this information, the gold grades modelled could be adjusted to some recovery factor. Also in order to gain greater confidence in the mineable reserves, it is recommended some type of three-dimensional pit optimizer be used, for example, the floating cone.

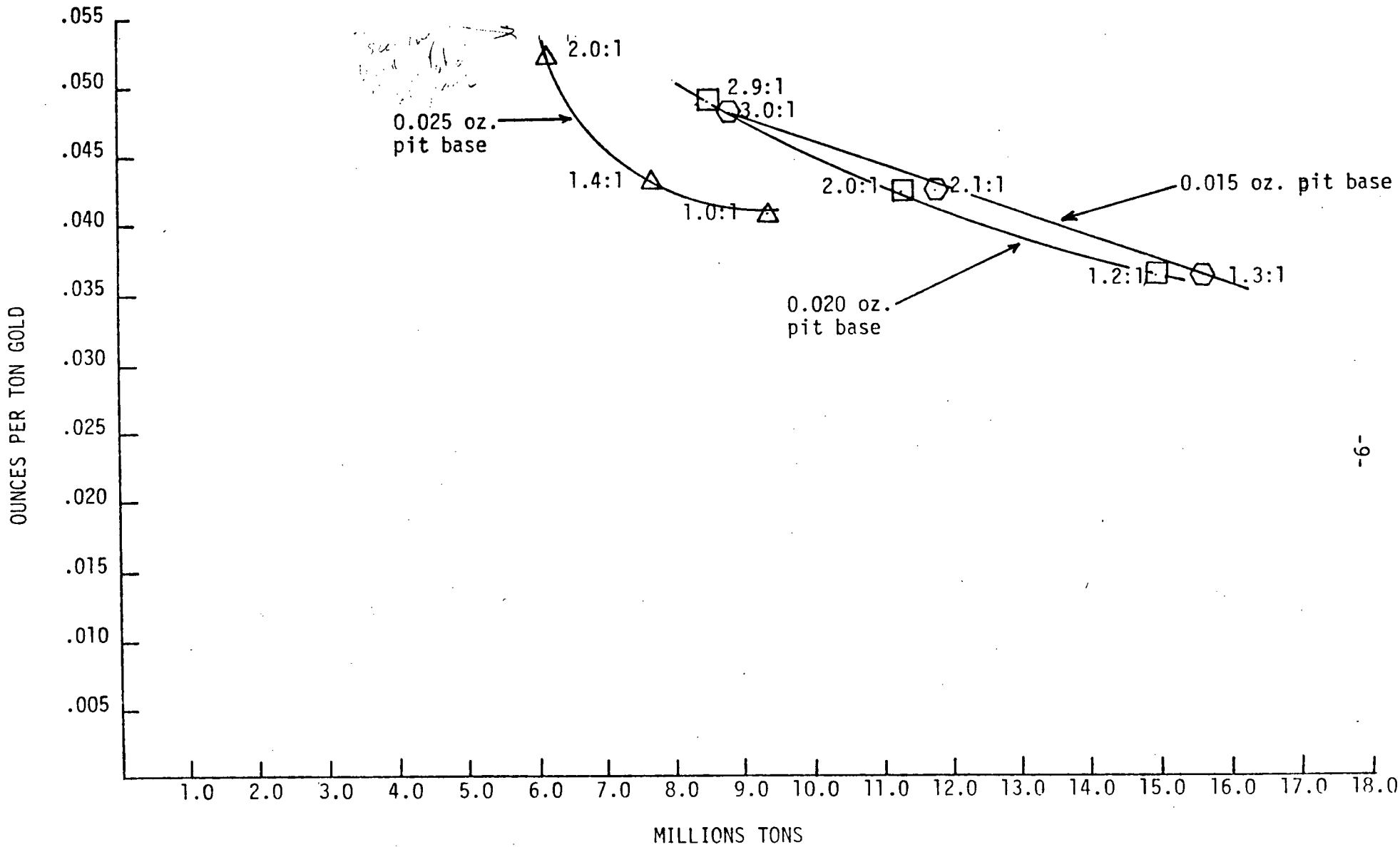

Figure 2. Grade-Tonnage curves from pit designs. Grades plotted at 0.025, 0.020, and 0.015 cut-offs, also showing waste to ore ratios.

AMOCO MINERALS-GILT EDGE-GEOLOGICAL RESERVES-DEC 1981

GEOLOGICAL TOTAL RESERVES

GEOLOGICAL RESERVES

KILO TONS DISTRIBUTED BY ROCK TYPE

GRADE DISTRIBUTION CATEGORIES		Sulfide MAT. TYPE 61	mixed MAT. TYPE 62	oxide MAT. TYPE 63	Barren MAT. TYPE 30	= TOTAL KILO TONS	1000s tons ACCUM KILORX TONS	Head Grade ACCUM TYPE RX GRADE
EQ 0.000		46186.	11170.	11446.	803388.	872190.	935286.	.001
GT 0.000	LT .005	3603.	454.	1732.	1087.	6877.	63096.	.022
GE .005	LT .010	9694.	2247.	2412.	971.	15325.	56220.	.024
.010	.015	7926.	1331.	2462.	957.	12676.	40895.	.030
.015	.020	4801.	925.	1904.	878.	8508.	28220.	.038
.020	.025	2941.	522.	1123.	771.	5356.	19712.	.047
.025	.030	2052.	354.	630.	487.	3523.	14355.	.056
.030	.035	1222.	205.	361.	303.	2091.	10833.	.065
.035	.040	1268.	228.	397.	484.	2376.	8742.	.073
.040	.045	573.	124.	181.	278.	1156.	6366.	.086
.045	.050	541.	140.	244.	244.	1169.	5210.	.095
.050	.055	237.	71.	130.	147.	585.	4042.	.109
.055	.060	236.	69.	156.	177.	638.	3456.	.119
.060	.065	115.	49.	76.	93.	333.	2818.	.133
.065	.070	105.	61.	78.	78.	322.	2485.	.142
.070	.075	101.	41.	81.	58.	281.	2163.	.153
.075	.080	45.	17.	64.	62.	188.	1882.	.166
.080	.085	35.	9.	46.	62.	152.	1693.	.175
.085	.090	32.	17.	37.	40.	126.	1542.	.184
.090	.095	21.	13.	40.	53.	127.	1415.	.193
.095	.100	21.	11.	40.	29.	101.	1288.	.203
.100	.105	24.	6.	31.	15.	76.	1188.	.212
.105	.110	21.	11.	38.	22.	92.	1112.	.219
.110	.115	19.	5.	28.	14.	66.	1020.	.229
.115	.120	16.	2.	26.	20.	64.	954.	.238
.120	.125	11.	3.	21.	11.	46.	890.	.246
.125	.130	9.	5.	21.	20.	55.	844.	.253
.130	.135	8.	5.	23.	13.	49.	790.	.262
GE .135		281.	49.	171.	240.	741.	741.	.270
TOTALS		82145.	18144.	23995.	811002.			

TONS MAY NOT BALANCE EXACTLY DUE TO NEAREST-KILOTON ROUNDING

Table 1. Geological Reserves

above .02 = 19.712 million tons

with 3.66 million tons in Barren Mat type 30

AMOCO MINERALS-GILT EDGE-MINING RESERVES-DEC 81 - .015 CUTOFF

MINEABLE TOTAL RESERVES

DESIGN NUMBER 1 INCREMENT NUMBER 1

MINING TOTAL RESERVES
KILO TONS DISTRIBUTED BY ROCK TYPE R

GRADE DISTRIBUTION CATEGORIES		MAT. TYPE 61	MAT. TYPE 62	MAT. TYPE 63	MAT. TYPE 30	TOTAL KILO TONS	ACCUM KILORX TONS	ACCUM TYPE R GRADE
EQ 0.000 (equal to zero)		595.	598.	533.	9079.	10856.	36508.	.018
GT 0.000	LT .005	517.	93.	639.	0.	1299.	25652.	.026
GE .005	LT .010	2035.	640.	1291.	0.	4016.	24353.	.027
.010	.015	2335.	712.	1652.	0.	4699.	20337.	.031
.015	.020	1884.	585.	1338.	0.	3807.	15638.	.036
.020	.025	1720.	366.	924.	0.	3011.	11831.	.042
.025	.030	1445.	252.	574.	0.	2271.	8820.	.048
.030	.035	903.	187.	338.	0.	1427.	6549.	.055
.035	.040	964.	159.	365.	0.	1519.	5122.	.061
.040	.045	421.	108.	159.	0.	688.	3603.	.072
.045	.050	372.	126.	215.	0.	714.	2915.	.078
.050	.055	171.	59.	111.	0.	342.	2201.	.088
.055	.060	153.	61.	141.	0.	355.	1860.	.095
.060	.065	59.	43.	65.	0.	177.	1505.	.104
.065	.070	52.	55.	71.	0.	188.	1328.	.110
.070	.075	61.	36.	69.	0.	166.	1140.	.117
.075	.080	32.	15.	61.	0.	108.	974.	.124
.080	.085	29.	9.	45.	0.	83.	866.	.130
.085	.090	27.	17.	35.	0.	79.	782.	.135
.090	.095	18.	13.	40.	0.	72.	703.	.140
.095	.100	17.	11.	36.	0.	64.	632.	.146
.100	.105	13.	6.	31.	0.	50.	568.	.151
.105	.110	13.	11.	36.	0.	60.	518.	.156
.110	.115	9.	5.	28.	0.	42.	459.	.162
.115	.120	12.	2.	24.	0.	38.	417.	.167
.120	.125	6.	3.	21.	0.	30.	379.	.172
.125	.130	5.	5.	21.	0.	31.	349.	.176
.130	.135	3.	5.	23.	0.	31.	318.	.181
GE .135		58.	47.	171.	0.	283.	288.	.186

Table 2. Mineable reserves using 0.015 ounce gold contour for pit base.

AMOCO MINERALS-GILT EDGE-MINING RESERVES-.020 CUTOFF-JAN 82

MINEABLE TOTAL RESERVES

DESIGN NUMBER 1 INCREMENT NUMBER 1

MINING TOTAL RESERVES
KILO TONS DISTRIBUTED BY ROCK TYPE R

GRADE DISTRIBUTION CATEGORIES		MAT. TYPE 61 +	MAT. TYPE 62 +	MAT. TYPE 63 +	MAT. TYPE 30 =	TOTAL KILO TONS	ACCUM KILORX TONS	ACCUM TYPE R GRADE
EQ 0.000		397.	542.	397.	8753.	10089.	33622.	.019
GT 0.000	LT .005	337.	79.	651.	0.	1067.	23534.	.027
GE .005	LT .010	1596.	537.	1183.	0.	3316.	22466.	.028
.010	.015	1928.	714.	1559.	0.	4200.	19150.	.031
.015	.020	1645.	600.	1384.	0.	3629.	14950.	.036
.020	.025	1515.	355.	936.	0.	2806.	11321.	.042
.025	.030	1317.	245.	571.	0.	2133.	8515.	.049
.030	.035	844.	181.	326.	0.	1351.	6382.	.055
.035	.040	935.	185.	363.	0.	1483.	5031.	.062
.040	.045	420.	108.	157.	0.	685.	3547.	.072
.045	.050	374.	125.	210.	0.	709.	2862.	.079
.050	.055	170.	59.	111.	0.	340.	2153.	.089
.055	.060	148.	61.	135.	0.	344.	1814.	.096
.060	.065	69.	43.	58.	0.	170.	1469.	.105
.065	.070	59.	55.	64.	0.	178.	1299.	.111
.070	.075	58.	35.	65.	0.	159.	1122.	.118
.075	.080	32.	15.	57.	0.	105.	963.	.125
.080	.085	29.	9.	45.	0.	83.	858.	.131
.085	.090	27.	17.	31.	0.	75.	775.	.136
.090	.095	19.	13.	38.	0.	70.	700.	.141
.095	.100	17.	11.	35.	0.	63.	630.	.147
.100	.105	13.	6.	28.	0.	47.	568.	.152
.105	.110	13.	11.	36.	0.	60.	521.	.157
.110	.115	9.	5.	27.	0.	41.	461.	.163
.115	.120	12.	2.	24.	0.	38.	420.	.168
.120	.125	6.	3.	19.	0.	28.	382.	.173
.125	.130	6.	5.	20.	0.	31.	354.	.177
.130	.135	3.	5.	22.	0.	30.	323.	.182
GE .135		73.	49.	170.	0.	293.	293.	.187
TOTALS		12071.	4076.	8723.	8753.			

Table 3. Mineable reserves using 0.020 ounce gold contour for pit base.

Based on $500 Au

AMOCO MINERALS-GILT EDGE-MINING RESERVES(.025 CUTOFF-)JAN 82

MINEABLE TOTAL RESERVES

DESIGN NUMBER 1 INCREMENT NUMBER 1

MINING TOTAL RESERVES
KILO TONS DISTRIBUTED BY ROCK TYPE R

GRADE DISTRIBUTION CATEGORIES		MAT. TYPE 61	MAT. TYPE 62	MAT. TYPE 63	MAT. TYPE 30	TOTAL KILO TONS	ACCUM KILORX TONS	ACCUM TYPE R GRADE
EQ 0.000		60.	105.	59.	6074.	6298.	18702.	.022
GT 0.000	LT .005	49.	44.	285.	0.	378.	12404.	.033
GE .005	LT .010	388.	219.	471.	0.	1078.	12026.	.034
.010	.015	459.	314.	785.	0.	1558.	10948.	.037
.015	.020	573.	339.	822.	0.	1734.	9390.	.041
.020	.025	598.	249.	655.	0.	1502.	7656.	.046
.025	.030	704.	195.	440.	0.	1338.	6154.	.052
.030	.035	476.	150.	269.	0.	895.	4816.	.058
.035	.040	639.	163.	290.	0.	1092.	3921.	.064
.040	.045	292.	88.	134.	0.	515.	2829.	.074
.045	.050	272.	87.	173.	0.	532.	2314.	.082
.050	.055	129.	41.	88.	0.	257.	1782.	.092
.055	.060	116.	37.	109.	0.	261.	1525.	.098
.060	.065	56.	31.	50.	0.	137.	1263.	.107
.065	.070	48.	29.	55.	0.	132.	1126.	.112
.070	.075	49.	23.	58.	0.	129.	995.	.118
.075	.080	25.	14.	48.	0.	87.	865.	.125
.080	.085	22.	8.	40.	0.	70.	778.	.130
.085	.090	24.	16.	29.	0.	69.	708.	.135
.090	.095	16.	13.	35.	0.	64.	639.	.140
.095	.100	13.	11.	31.	0.	55.	575.	.146
.100	.105	12.	6.	27.	0.	44.	520.	.151
.105	.110	12.	10.	33.	0.	55.	476.	.155
.110	.115	8.	5.	24.	0.	37.	421.	.161
.115	.120	11.	2.	21.	0.	34.	384.	.166
.120	.125	6.	3.	17.	0.	26.	350.	.171
.125	.130	4.	5.	19.	0.	28.	324.	.175
.130	.135	3.	5.	20.	0.	28.	296.	.179
GE .135		50.	49.	169.	0.	268.	268.	.184
TOTALS		5112.	2259.	5257.	6074.			

TONS MAY NOT BALANCE EXACTLY DUE TO NEAREST-KILOTON ROUNDING

Table 4. Mineable reserves using 0.025 ounce gold contour for pit base.

MINERAL RESERVE STUDY

by

BRUCE B. GODDARD
&
VICTORIA C. SCHURER

October 1981

MINERAL RESERVE STUDY
GILT EDGE PROJECT
LAWRENCE COUNTY, SOUTH DAKOTA

AMOCO MINERALS COMPANY
DENVER, COLORADO

BY

BRUCE. B. GODDARD
&
VICTORIA C. SCHURER

OCTOBER 1981

GILT EDGE PROJECT

LAWRENCE COUNTY, SOUTH DAKOTA

MINERAL RESERVE INVENTORY
OPEN PIT STUDY

INTRODUCTION

A trial pit was designed involving the two major zones or trends of mineralization, the Dakota Maid and the Sunday. It had been previously thought that mining of these deposits would require two medium sized pits. It now appears there is the possibility that the two pits might be merged into one large pit. However, more information must be gained through additional drilling before this can be positively stated.

The figures quoted below are the result of studying the assay data from some 215 rotary and 20 diamond drill holes. Cross and longitudinal sections were constructed from engineering data provided by the surveying firm, Banner Associates, Inc., and the topography provided by the engineering firm of Horizons, Inc. Both firms are located in Rapid City, South Dakota. Some discrepancies were noted in surveying and survey control. However, for the purposes of this study, the errors did not appear to be major, and therefore the work was used. The assay and other data which has been developed over a period of years has been done mostly by Cyprus Exploration Company. It should be noted that the assay results used in these reserve calculations were the average of each sample being assayed in three laboratories. Where major differences still existed, check assaying was done to arrive at firm assay figures.

Two major zones or trends of mineralization are indicated, the Dakota Maid and the Sunday. There is some tendency for the better mineralized zones within these trends to change direction, dip and rake, or to be cut off by faulting, making drilling for continuation of these zones difficult. Also, in some areas the gold mineralization is very erratic or randomly distributed,

and values are not projectable for long distances. Finally, the depth of the blocks vary widely, which would create some difficulties in an open pit mining operation.

VERTICAL SECTIONS

A number of vertical sections looking toward the northeast and labeled AA' through VV' were drawn to obtain the maximum cross-sectional area across the mineralized trends. This NE pattern of sections gives an adequate view of the plotted assay information for preliminary pit design and ultimate slope. Due to the arcuate configuration of the Dakota Maid and the Sunday mineralized zones, a few radial cross sections WW' - ZZ' were developed. Sections AA-AA' and BB-BB' were drawn through the area where the two mineralized zones appear to merge. Sections CC-CC' through FF-FF' were constructed to provide additional longitudinal views.

All gold assays that are greater than the .020 ounce per ton cut off are blocked out on the sections. Therefore, all areas shown in solid blocks represent mineralization that might be considered ore within this shallow pit, with the provision that stripping ratios and treatment costs are not too high.

The sections were drawn on 100-foot centers with a 50-foot zone of influence on either side of the section. Holes were projected to the section (not more than 50 feet) and areas of ore and waste were determined. The areas were then assigned a proper length value, 50 feet on each side of the section, or as determined by other factors, especially toward the ends of the pit.

DEFINITIONS

For clarification of terms used in this report, the following definitions are given:

1. Positive Ore - ore for which tonnage is computed from dimensions revealed in surface cuts, trenches and drill holes, and for which the grade is computed from results of detailed sampling and assaying.

The geologic character is so well defined that the general size, shape and mineral content are well established. (Sections are 100 feet apart, drill holes projected to section less than 50 feet, maximum space between drill holes is less than 150 feet.)

2. Probable Ore - ore for which tonnage and grade are computed partly from specific measurement and partly from projection for a reasonable distance based on geologic evidence.
3. Potential Ore - quantitative estimates are based largely on knowledge of the geologic character of the deposit and for which there are few and widely spaced sampled intervals. This mineralization must be regarded as speculative, or prospective ore.
4. Waste - barren rock within the pit, or rock material too low in grade to be of economic value under present economic conditions.
5. Dilution - ten per cent dilution is applied to the calculated ore reserve to provide for overbreak/change in slope.
6. Mining Ore Reserve - the sum of calculated ore reserve tonnages and the dilution tonnage, as a result of actual mining.
7. Pit Slope - assumed at 45° ultimate slope, though, in actuality, it may be possible to steepen walls.

MINERAL RESERVE INVENTORY

The mineral inventories shown below in Table A, represent the total reserves that drilling has developed to date. These total reserves are analyzed from a mineability standpoint by means of a relatively shallow open pit, and the results of this study are quoted in Table B below. These figures should not be confused with the total mineral reserve figures. A factor of 12 cubic feet per ton of rock in place was used to convert cubic volumes into tons.

TABLE A: MINERAL INVENTORY: COMBINED ZONES

Category Oz/Gold/Ton	Inventory Tons	Aver. Grade Oz/Ton	% of Total Inventory
.020 - .029	2,971,270	.024	25.5%
.030 - .039	2,512,245	.034	21.6%
.040 - .049	2,065,749	.043	17.8%
.050 - .059	909,293	.055	7.8%
.060 - .069	774,428	.063	6.7%
.070 - .079	1,195,687	.074	10.3%
.080 - .089	228,793	.082	2.0%
.090 - .099	27,583	.093	0.2%
.100 - .199	697,716	.130	6.0%
.200 & Above	249,334	0.214	2.1%
Grand Total	11,632,096	.052	100.0%

A mineral reserve of positive and conservatively stated probable blocks of 11,632,096 short tons grading 0.052 ounce gold per ton has thus far been developed by the drilling effort. No figure can realistically be stated for the potential ore blocks within this erratically mineralized orebody due to the lack of information at depth. However, the potential ore tonnage possibly would equal the inventory thus far developed.

PRELIMINARY TRIAL PIT

To delineate a mineable ore reserve, sample assays were first inspected and grouped for each hole. Generally, sufficient drilling has been done along both mineralized trends to establish continuity of mineralization along strike and to give approximate width. Further interior rotary drilling and deeper diamond drilling is indicated, both for pit planning purposes and for ore development.

C. A. Mark of Amoco Minerals Company established the cut off grade of 0.020 ounce gold per ton for the trial pit. A number of open pit gold mines located in Nevada and other western states are currently using a similar cut-off grade, although this cut-off would vary with the price of gold. Ton-

nage and grade for this preliminary trial pit were derived by specifying a pit limit that essentially includes all material with grades in excess of 0.020 ounce gold/ton. An ultimate pit slope of 45 degrees was used. This pit slope should be quite conservative, and it may be that a steeper pit slope could be utilized which would reduce the amount of internal waste and overburden that would have to be removed.

This volume of rock or mineralized mass was analyzed from a standpoint of mineability within a relatively shallow pit. Obviously, a much more precise and detailed mining plan taking into consideration the bench requirements, the haulage roads which spiral down on a continuous grade, switchbacks, etc., should be prepared.

In Table B shown below, the "mineable" ore reserve cut-off grade rises in increments of .01 oz. Au/ton. If Amoco Minerals should contemplate an open pit mining operation at Gilt Edge, these figures could be used for a comparison of cash flows generated. However, a number of pit configurations and options would have to be studied to determine stripping ratios for various grades involved. The following table (Table B) lists the inventory of calculated "mineable" ore reserves available within the trial pit.

TABLE B: MINEABLE ORE RESERVE

Section	Tons Ore	Grade	Tons Waste
G-G'	83,396	.043	223,488
H-H'	451,844	.048	651,083
I-I'	985,667	.074	989,572
J-J'	1,291,736	.038	1,395,247
K-K'	815,135	.058	1,531,639
L-L'	1,129,063	.054	1,682,021
M-M'	1,025,012	.061	1,558,938
N-N'	825,648	.041	1,689,177
O-O'	520,656	.048	1,267,527
P-P'	404,246	.060	1,192,758

(TABLE B CONTINUED ON NEXT PAGE)

TABLE B: MINEABLE ORE RESERVE (CONTINUED)

Section	Tons Ore	Grade	Tons Waste
Q-Q'	229,630	.050	934,404
R-R'	231,332	.046	327,168
S-S'	33,333	.052	188,525
T-T'	---	---	3,852
Totals	8,026,698 tons *		13,635,399 tons

Average Grade for Mineable Ore Reserve: 0.052 oz/ton

Stripping Ratio of Waste to Ore: 1.70:1

Conversion Factor: 12 cubic feet/ton ore in place.

Total gold contained in mineable ore reserve amounts to 422,648 ounces.

Within the two zones of mineralization, our study has indicated mineable ore reserves of 8,026,698 tons grading 0.052 ounce gold per ton. This ore would become available by excavating 13,635,399 tons of barren overburden and internal waste, reflecting an overall waste to ore ratio of 1.70:1. It should be pointed out that some of the tonnage of internal waste was given a waste designation simply for lack of assay data. There is the possibility that internal drilling would provide data which could reduce the overall waste tonnage. A computerized study of reserves would probably show a greater mineralized reserve than that quoted above. If constraints are not put on the computer, much internal mass would show as ore.

There are several areas outside the confines of the trial open pit which have ore blocks near the surface. These could possibly be mined as separate small open pits and cuts which would add to the total ore reserve tonnage. Additional mineral reserve blocks exist below the bottom of the trial pit, but were considered too deep to be economically open-pitted and underground mining may be warranted.

* No allowance was made for the subtraction of tonnage previously mined underground, since little is known about the configuration of the stopes, or if these stopes would be within the pit limits.

RECOMMENDATIONS

Until recently, the property had been considered as a potential open pit operation and most exploration drilling activity focused on proving shallow ore reserves. An angled core drill hole drilled in 1980 intersected 280 feet of continuous gold mineralization avaraging 0.11 ounce of gold per ton, with one interval as high as 0.227 ounce per ton. Since this intersection lies 600 to 900 feet below surface, underground mining is indicated. A drilling program is now underway to evaluate and delineate underground reserves. Should this drilling block out enough underground reserves of sufficient grade to warrant an underground operation, the shallow pitting operation should also be considered. Mining operations at the Gilt Edge would then consist of an underground and a surface mining combination which would feed a large cyanidation plant utilizing agitation leaching.

After completion of the program to delineate the potential deeper ore reserves, a complete feasibility study should be conducted. The feasibility study should include the combination open pit and underground mining operation, as well as the cost of building a cyanidation agitation leach plant.

A formal open pit plan and a design for bulk underground mining production could be simultaneously developed. Design and construction of the cyanidation plant could immediately follow. Construction of the agitation leach plant should require about two years. During the construction period both mines could be under development.

MINERAL RESERVE STUDY
HORIZONTAL (LEVEL) PLANS

by

BRUCE B. GODDARD
&
VICTORIA C. SCHURER

January 1982

MINERAL RESERVE STUDY

HORIZONTAL (LEVEL) PLANS

GILT EDGE PROJECT

LAWRENCE COUNTY, SOUTH DAKOTA

TEXT TO ACCOMPANY SET
OF HORIZONTAL PLANS

For
AMOCO MINERALS COMPANY
DENVER, COLORADO

By
BRUCE B. GODDARD
&
VICTORIA C. SCHURER
Consulting Geologists and Mining Engineers

JANUARY 1982

GILT EDGE PROJECT
LAWRENCE COUNTY, SOUTH DAKOTA

Mineral Reserve Inventory
Horizontal (Level) Plans

INTRODUCTION

The figures quoted below are the results of studying and averaging assay data from approximately 215 rotary and 20 diamond drill holes. These tonnage figures are believed to be quite conservative because of the restricted horizontal and vertical influence exerted by the assay intercepts and because of the fact that the drilling has not been completed. Level plans utilizing polygons were constructed from engineering data provided by Banner Associates, Inc. and Horizons, Inc. Both firms are located in Rapid City, South Dakota. Some discrepancies were noted in surveying and survey control work. Some of these differences were resolved. For the purposes of this study, the remaining discrepancies or errors are not large or significant, and therefore the work was used. The assay data and other data used in the study was developed over a period of years by Cyprus Exploration Company, which has now been merged into the Amoco Minerals Company. It should be noted that the assay results used in these reserve calculations were the average of each sample assayed in three independent laboratories. Where major differences still existed, check assaying was done to arrive at firm assay figures.

It had been previously thought that there were two zones of mineralization, the Dakota Maid and the Sunday, and that mining these two zones would require separate open pits. It now appears that the mineralization merges, at least in part, and that there is the possibility of mining the deposit with one large pit, which would be arcuate in configuration. However, more information must be gained through additional drilling and engineering studies before this can be positively known.

The figures quoted below were developed from a series of fourteen level (horizontal) diagrams developed to view the deposit

in plan. The intermediate levels (5500 to 5300) show numerous holes and constructed polygons. With increase in depth (5000 level) very few holes attained this elevation, and therefore there are only a few polygons. The same can be said of the uppermost levels.

This study of the deposit by means of a series of plans is in part a continuation of an earlier study of Gilt Edge mineral reserves done by the vertical cross section method (see report, Mineral Reserve Study, October 1981) and has provided a check on results obtained in the earlier study. Some variations were noted, with the mineral reserves grading over 0.02 ounces per ton decreasing about 24 per cent or by 2,828,171 tons. Overall, the mineral reserves above 0.02 ounces Au per ton decreased from 11,632,096 tons (Combined - Positive & Probable) to 8,803,925 tons using the horizontal polygon study. The reason for this reduction is the absolute 50 feet maximum lateral influence allowed around each hole.

HORIZONTAL SECTIONS OR LEVEL PLAN CONSTRUCTION

Fourteen plan maps were constructed to view the mineral deposit at 50 foot intervals. The uppermost plan is at an elevation of 5650 feet. Subsequent plans were constructed at 50 foot intervals down to and including the 5000 level. Each level map shows the rotary and diamond drill holes that intersect or penetrate that particular level interval. In addition, the plans show the complete trace of angled diamond drill holes. Portions of the diamond drill holes not intersecting a given level are depicted by broken lines. Circles were constructed around each hole using a 50 foot radius. This 50 foot radial distance from the hole represents the maximum horizontal influence that a sampled intercept can exert. Polygons were constructed and obtained their shape through the intersections of the 50 foot radii scribed from adjoining holes. Where there are no adjoining holes, the 50 foot radius scribes a complete circle having a planar area of 7,854 square feet.

The maximum vertical influence of each hole is established by a 25-foot vertical distance above the plan elevation and by a 25 feet vertical distance below the plan elevation. The polygons thus have a 50-foot maximum total vertical influence, and a 50-foot lateral or horizontal influence from each hole.

ASSAY RESULTS

To incorporate the existing assay data from the various holes into the polygonal analysis, it was necessary to average the assays from each hole into the 50-foot increments corresponding to the designated level intervals. Where a hole only partly penetrates a level interval (i.e., the top or bottom segment) the vertical influence of the associated polygon is equal to the vertical footage of the hole within the given level interval. Where less than 50 per cent of a hole had been sampled within a given level interval, the associated polygon is labeled "Not sampled" on the corresponding level map.

CALCULATIONS

The areas of the individual polygons were determined using a compensating polar planimeter. Each measured area was multiplied by the vertical influence of the associated polygon (usually 50 feet) to obtain the volume, which was in turn divided by a tonnage factor of 12 cubic feet per ton, the appropriate factor for this rock type.

For purposes of evaluation, the polygons were divided into 15 average-grade categories. (See Appendix.) This grade classification scheme serves to illustrate that the bulk of the mineralization is low grade. A cut-off grade of 0.016 ounces Au/ton yields a total of 11,727,747 tons averaging .039 ounces Au/ton. However, this is not equivalent to the total mineable reserves, and it is possible that some of the very low grade material would have to be mined in the process. Also, there are areas that are undoubtedly mineralized that would increase total mineral reserves but are, as yet, insufficiently drilled out.

It must be noted here that there remains a significant lack of assay data, as is illustrated by the somewhat irregular clustering of drill holes in the Gilt Edge area. Further drilling needs to be done to more firmly establish mineable reserves and continuity of mineralization.

LEVEL (POLYGON) MAPS

Fourteen level maps were constructed, each with a maximum vertical influence of 50 feet. The maps contain average grades and total estimated tonnages for each polygon. Furthermore, the polygons have been color-coded based on the 14 grade categories listed in the legend of each level map.

Each level map contains the locations of only those drill holes actually intersecting that particular level interval; the exceptions are the angled diamond drill holes, the locations and lateral traces of which are depicted on all level maps.

CONCLUSIONS

It is believed that this study of Gilt Edge mineral reserves represents a conservative method of estimating total mineral reserves in the deposit. The study has shown a reserve of 11,727,747 tons present with an average grade of .039 ounce gold per ton. The cut-off grade for the reserve is 0.016 ounce gold per ton. This mineral reserve contains 453,151 ounces of gold, which is an impressive figure. The tons and average grade utilizing different grade categories are shown in Table A below. The cumulative tonnage and the average calculated grade using a number of different cut-off grades are shown in Table B.

The focus of this study has been on relatively shallow mineralization delineated to a large extent by rotary drilling. At the time of this study, a diamond drilling program was underway to evaluate the deeper potential of the deposit. The results of this drilling are unknown to the writers, and hence, any tonnages developed by the 1981 drilling program are not included in this report.

Our previous study of the two zones of mineralization by vertical sections has indicated that most of the mineral reserve quoted is available by open pit mining at a relatively low waste-

to-ore ratio. It should be pointed out that some additional mineral reserves should be found with further drilling or mining, because some internal areas were given a waste designation simply for lack of assay data. There is the possibility that more internal drilling would provide data that would increase mineral reserves.

TABLE A: MINERAL INVENTORY: COMBINED ZONES

Category oz/Gold/ton	Inventory Tons	Average Grade oz. Au/ton	Contained oz. Au
0.300 & above	10,030	0.391	3,922
0.200 - 0.299	161,135	0.253	40,778
0.151 - 0.199	134,928	0.163	22,056
0.100 - 0.150	294,648	0.124	36,659
0.090 - 0.099	1,824	0.097	177
0.080 - 0.089	200,907	0.085	17,147
0.070 - 0.079	293,744	0.072	21,158
0.060 - 0.069	440,641	0.064	28,070
0.050 - 0.059	747,881	0.054	40,283
0.040 - 0.049	1,119,856	0.044	49,338
0.030 - 0.039	1,415,722	0.034	47,888
0.020 - 0.029	3,982,609	0.024	95,382
0.016 - 0.019	2,923,822	0.017	50,303
0.010 - 0.015	6,485,439	0.012	79,841
0.001 - 0.009	14,347,694	0.005	69,875

TABLE B. CUMULATIVE TONNAGES & AVERAGE GRADES

Cut-Off Grade	Cumulative Tonnage	Average Grade	Contained ozs. Au
0.200 & above	171,165	0.261	44,690
0.151	306,093	0.218	66,746
0.100	600,741	0.172	103,405
0.090	602,565	0.172	103,582
0.080	803,472	0.150	120,729
0.070	1,097,216	0.129	141,887
0.060	1,537,857	0.111	169,957

TABLE B. CUMULATIVE TONNAGES & AVERAGE GRADES

(Continued)

Cut-Off Grade	Cumulative Tonnage	Average Grade	Contained ozs. Au
0.050	2,285,738	0.092	210,240
0.040	3,405,594	0.076	259,578
0.030	4,821,316	0.064	307,466
0.020	8,803,925	0.046	402,848
0.016	11,727,747	0.039	453,151
0.015	12,696,247	0.037	467,678
0.010	18,213,186	0.029	532,992
0.001	32,560,880	0.019	602,867

METALLURGY

Metallurgy - Extensive data has been acquired relative to the metallurgical character and processing of Gilt Edge gold mineralization. Early studies identified three principal categories of mineralized rock-oxide, sulphide, and mixed-oxide sulphide. Cyanide leaching experiments indicated that gold recoveries ranged from 90 percent for oxide material to 65 percent for sulphide material. A series of experiments designed to evaluate the heap leach potential of Gilt Edge ores progressed in scale from bottle tests to bucket-leach tests to laboratory column tests. All results suggested oxide zone gold recoveries on the order of 70 percent.

At this point, a 1,700-ton pilot heap-leach facility was constructed to demonstrate the safety and reclamation of such projects to state authorities and to provide preliminary data on large-scale gold recovery. Demonstration of heap-leach operation to the State of South Dakota was an unqualified success. Gold recovery during the test was approximately 48 percent, even though the rock had not been crushed prior to heap construction. Additional experimentation with large-scale leaching of 25-ton bulk samples in 40-foot-high column tests proved both that the percolation characteristics of Gilt Edge ore are acceptable for the construction of tall heaps and that gold recoveries of 75 percent can be routinely achieved outside the laboratory.

Following the heap-leach experiments, a series of studies is now underway to determine the behavior of Gilt Edge material under conventional (agitation) milling conditions. Results of bottle-roll tests indicate 1) gold recovery is up to 82 percent and averages 76 percent; 2) silver recovery is 3.5 times that expected under heap-leach conditions; 3) interference with gold recovery by cyanide-soluble copper is significant in parts of the shallow deposit, but will not effect overall processing of ore by either conventional milling or heap leaching. Flotation studies have produced high-grade sulphide concentrates, and tests are continuing to determine best recovery procedures for these concentrates. Reports from the various consulting groups performing the metallurgical studies are included in this section.

METALLURGICAL EVALUATION 1978-1981

by

KAPPES, CASSIDAY & ASSOCIATES

July 6, 1982

The Gilt Edge deposit is both metallurgically and geologically complex. The problems of this evaluation are compounded by the presence of coarse gold, which makes interpretation of individual test results very difficult. This report is an attempt to summarize a massive amount of data, which is presented in the various other reports and laboratory studies, referred to throughout this report.

Submitted by,

Daniel W. Kappes
KAPPES, CASSIDAY & ASSOCIATES

DWK/df

REPORT 1982 C

SUMMARY REPORT

METALLURGICAL EVALUATION 1978-1981

GILT EDGE, SOUTH DAKOTA

6 July, 1982

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REPORT 1982 C

SUMMARY REPORT

METALLURGICAL EVALUATION 1978-1981

GILT EDGE, SOUTH DAKOTA

CONCLUSIONS

The metallurgical test work which has been performed on the Gilt Edge ores since 1978 has largely concentrated on examining the ores for their amenability for heap leaching, but preliminary data has also been generated on mill recoveries. The results of this work can be summarized as follows:

Oxidized Ores: Heap Leach Potential. Both the Sunday Zone and Dakota Maid Zone oxidized ores are heap leachable, with recoveries averaging 70 percent of contained gold when the ore is crushed to 2-inches or less. The Sunday Zone appears to show slightly higher recoveries than the Dakota Maid Zone (73 percent versus 67 percent). The oxidized ore represents approximately 1,500,000 tons of material at an average grade of 0.06 oz gold per ton. It can be mined at a stripping ratio on 3.0:1. By itself, it could be the basis for a small-sized mine/ heap leach operation.

Unoxidized Ores: Heap Leach Potential. The heap leach behavior of the unoxidized ore is highly variable. Average recovery from highly pyritic ores appears to be approximately 30 percent, and from all ores classed as "unoxidized", 45 - 50 percent, in a heap leach system when crushed to 2-inches or finer.

Even more so than with the oxidized ores, the Sunday Zone unoxidized material shows better recoveries than that from the Dakota Maid: general recovery from all unoxidized ores in the Dakota Maid Zone will be in the range of 30 - 45 percent, whereas Sunday Zone material will be in the range 45 - 60 percent.

Recovery Potential in Conventional Cyanide Mill. Average recovery in agitated tests, which reflect the recoveries in a conventional mill at medium grind sizes, is 76 percent of contained gold.

The laboratory test results indicate that mill recoveries from the remaining ore will be similar to recovery in the historic mills, which operated on the property in the late 1930's. A March 1968 report by Dolf Fieldman gives recoveries in the 1937-1940 period as "approximately 75 percent".

Improved recovery may be possible with very fine grinding, or with a combined flotation/cyanidation circuit. These studies were beyond the scope of the present work.

Recovery Potential of Existing Tailings. A complete plane table survey of the existing tailings piles was made in 1979, and the tailings were sampled by auger drilling. Recovery in agitated leach tests on pulverized portions of the auger drill samples showed 58 percent gold recovery from an average fire assayable gold content of 0.03 oz per ton. These tests essentially confirm work done by Battelle Memorial Institute in the 1950's.

The tailings piles contain a relatively small tonnage of material (150,000 tons), but they significantly improve the economics of heap leaching, since they can be used as the protective sand layer which is needed below the heaps.

SCOPE OF WORK PERFORMED

The conclusions summarized here are based on a total of 90 laboratory bucket leach tests on 40 different samples; four 40-foot column leach tests on 25-ton samples; one 1700 ton field leach test; 500 cyanide bottle roll tests; and 160 bottle roll and centrifuge tube leach tests on tailings samples. In procuring the samples, a total of five col-

lapsed mine portals were re-opened, including the King Tunnel, which required excavation of a fifty foot deep cut and re-timbering of 300 feet of new entry. One-hundred-fifty feet of new drift was driven. A surface pit was excavated, totalling about 3000 tons (1700 tons were actually moved from the pit for testing), and a heap leach test was constructed on the ore from the pits and operated through two seasons.

Ore blasted and moved during the various underground sampling programs totalled about 800 tons, at 37 different locations.

RECOMMENDED FOR FURTHER WORK

Laboratory testing of ores for heap leach evaluation is essentially complete. Further work is not likely to change, or improve the accuracy of, the conclusions presented here. Recoveries from the only large-scale field heap leach test were below expectations (46 percent versus the projected 57 percent); this has been attributed to the coarse size of the uncrushed ore placed on the heaps, and non-ideal stacking procedures. At least one further large-scale heap leach test (2500 tons) is recommended to verify that modified procedures will result in predicted recoveries (70 percent from crushed ore).

SUMMARY OF METALLURGICAL RESULTS

Figures 1 and 2 present bar charts showing the various metallurgical test programs which have been carried out on the Gilt Edge ores. Figure 1 summarizes the results for the oxidized ores, and Figure 2 summarizes the results of the unoxidized ores and tailings.

Footnotes for the bar charts are presented below.

(1) Cyprus Research Laboratories conducted two column leach tests in 1978 on rotary drillhole cuttings.

(2) In October of 1978, a preliminary series of seven 50 to 200 pound samples were taken by hand methods; three of these were from the walls of the Dakota Maid Pit, four from walls of underground workings in the Sunday Zone. These were subject to bucket leach tests, and the results were highly variable. The results were reported in Appendix C to the report titled "Gilt Edge Field Sampling and Laborotary Tests, 1979 through 1980", dated 10 August, 1981.

(3) In the 1979 bulk sampling program, a total of 21 one-ton bulk samples were taken at various underground locations. Thirteen of these were Sunday Zone oxidized samples (three in the One John), five were Sunday Zone unoxidized samples, and three were Dakota Maid unoxidized samples taken from the Dakota Maid decline. Forty-two bucket leach tests were run. The sampling procedures and test results were presented in the same report mentioned in footnote (2). In a subsequent program, ten bucket leach tests were run on selected large rocks, 2 to 8-inches diameter, from the same bulk samples. These were reported in a dual report issued 10/23 November, 1981, titled "1979 Mini-Bulk Samples/1979 Bulk Samples - Selected Large Rocks".

(4) During 1979, samples were also taken from sites which were inaccessible to the mechanized mining equipment used for the one-ton samples. A total of twelve mini-bulk (200 pound) samples were taken at these sites using a hand-held hammer and moil. These consisted of five oxidized, and two unoxidized, samples taken from the R-3 level of the internal Rattlesnake shaft (approximately 60 feet above the present water table, at a depth of 250 feet below the present surface); three samples of oxidized ore from the extreme north end of the Dakota Maid Zone in adits "B" and "C", and one oxidized and one unoxidized sample from the Dakota Maid Pit. Results of these tests are presented in the report named in footnote (3).

(5) In 1980, the long-collapsed King Tunnel was re-opened by excavating a 50-foot deep cut through overburden, re-mining through the collapsed timbers until solid rock and open tunnel was reached, then re-timbering and backfilling the new portal. The old tunnel was still open and accessible for sampling beneath most of the Dakota Maid Pit area. Ten one-ton and one 25-ton bulk samples were taken at various locations in the tunnel. Of the one-ton samples, two of these were too low grade to warrant further testing. Four oxidized, and four unoxidized, samples were tested in a laboratory bucket leach test program. Two of the oxidized samples tested contained below 0.007 ounces gold per ton. The apparent results were similar to the higher grade samples, but they are not reported in Figure 1 because the grade is too low to permit meaningful recovery figures to be derived from the tests.

The 25-ton sample was a sample of highly pyritic unoxidized ore. This sample was the subject of a 40-foot field column test on a 25-ton sample, and two laboratory bucket leach tests. Figure 4 shows a plot of recovery versus time for the bucket and tall-column leach tests on this sample (identified as column 4).

(6) In 1980, five 15-ton samples were taken in the Sunday Zone, Rattlesnake Tunnel, at five of the sites which had been sampled the previous year with one-ton samples. The 15-ton samples were combined by color (two yellow, three red) to make two bulk samples, which were leached at run-of-mine size in 40-foot, 25 ton test columns. Additional bucket leach tests were also performed on the samples. The additional bucket leach tests, and the 40-foot columns, essentially duplicated the bucket test results of the previous year. Figure 3 shows a plot of recovery versus time for the bucket and tall-column leach tests on these ores.

(7) As part of the same 1980 bulk sampling program, a new tunnel, the Laron Adit, was driven into an oxidized portion of the Dakota Maid Zone, approximately 80 feet north of the north end of the Dakota Maid Pit. A 25-ton bulk sample was mined from the face of the tunnel and used for a 25-ton, 40-foot column test and two laboratory bucket leach tests. Figure 4 shows a plot of recovery versus time for the bucket and tall-column leach tests on this sample (identified as column 3).

Figure 5 presents a plot showing recovery versus time for several of the different types of tests conducted on a typical sample of Sunday Zone oxidized ore.

(8) In 1980, the south pit wall of the Sunday Zone Pit was drilled and blasted to break approximately 3000 tons of rock. Seventeen-hundred tons of this material was excavated from the pit, and stacked 12 feet high on a double-layer hypalon pad for a heap leach test. Six laboratory bucket leach tests, and an extensive sampling and assaying program, were run on the ore to determine laboratory recoveries and heap ore grade.

Field heap construction methods were presented in a pictorial report titled "Pictorial Summary, 1980 Gilt Edge Heap Leach Project", dated 7 March, 1981. Heap leach recovery in 130 days leaching, was 46 percent of contained gold, which was less than the projected 57 percent (based on expected recovery from uncrushed rocks). The lower recovery is attributed primarily to non-ideal stacking procedures. Head assays and recoveries from the test are discussed in a letter to Ron Graichen, dated 19 April, 1982.

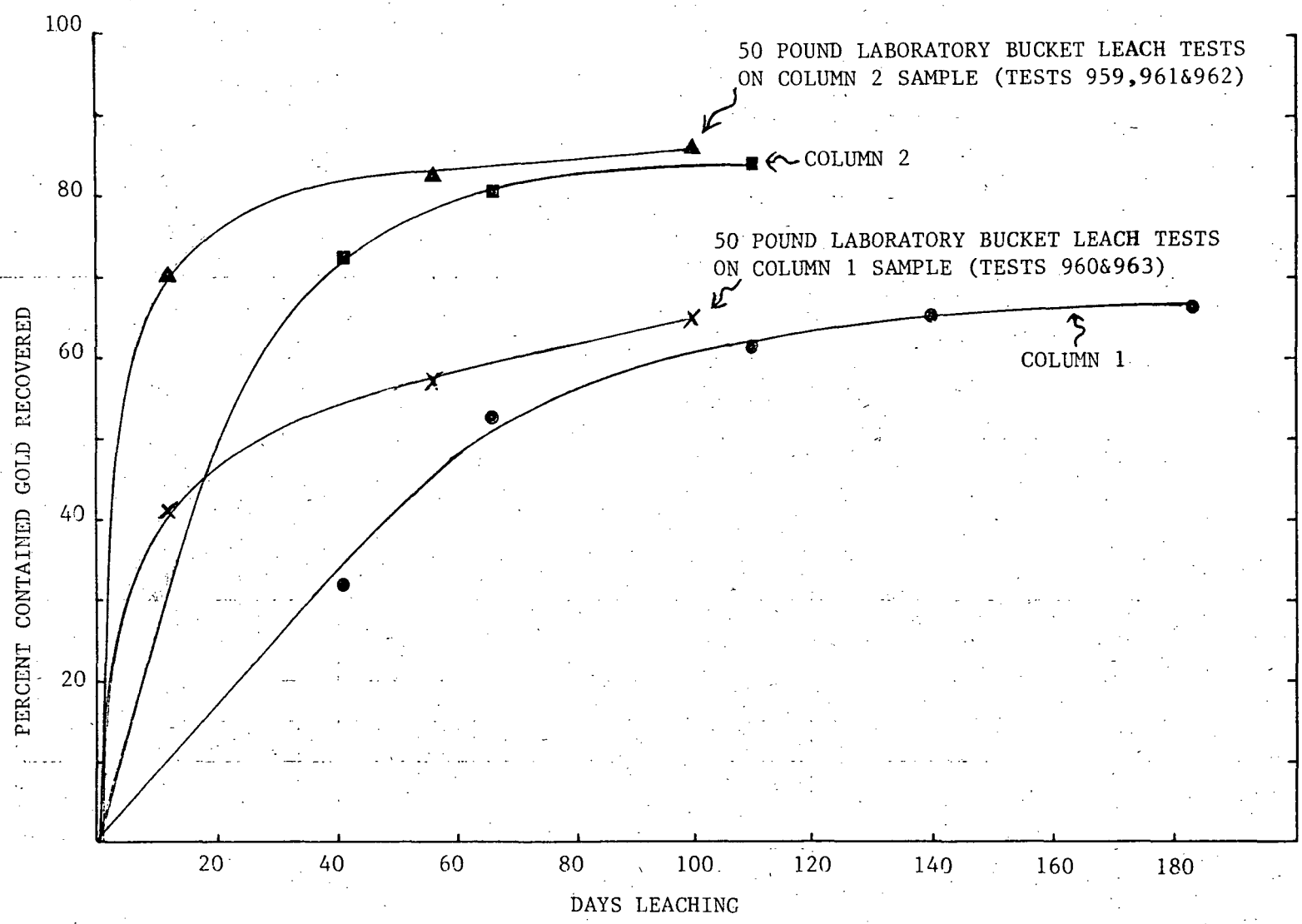

FIGURE 3. GOLD RECOVERY:GILT EDGE 40 FOOT COLUMN TESTS ON UNCRUSHED ORES. RATTLESNAKE TUNNEL SAMPLES:COLUMNS 1&2

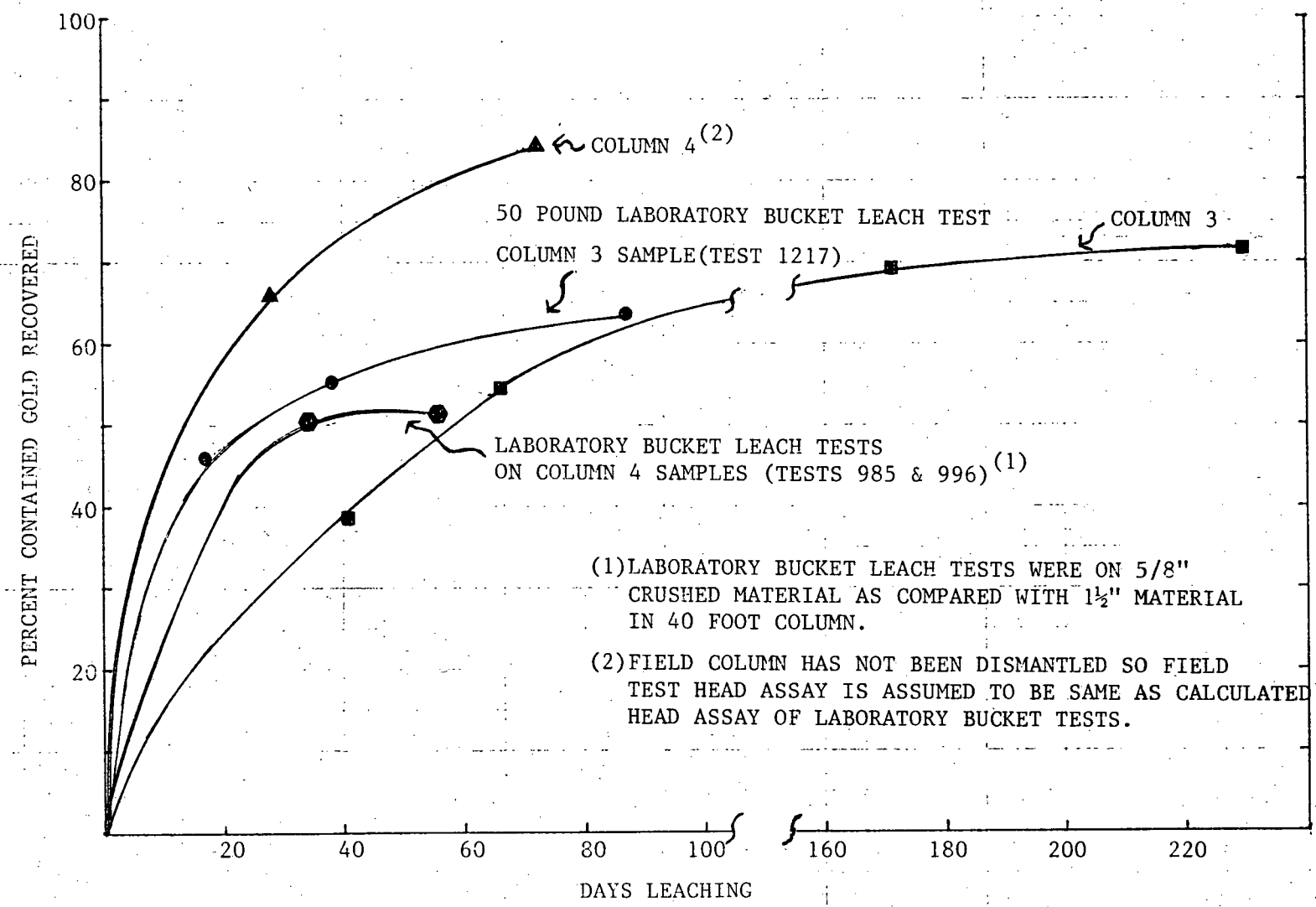

FIGURE 4. GOLD RECOVERY: GILT EDGE 40 COLUMN TESTS
UNCRUSHED DAKOTA MAID SAMPLE: COLUMN 3
CRUSHED KING TUNNEL SAMPLE: COLUMN 4

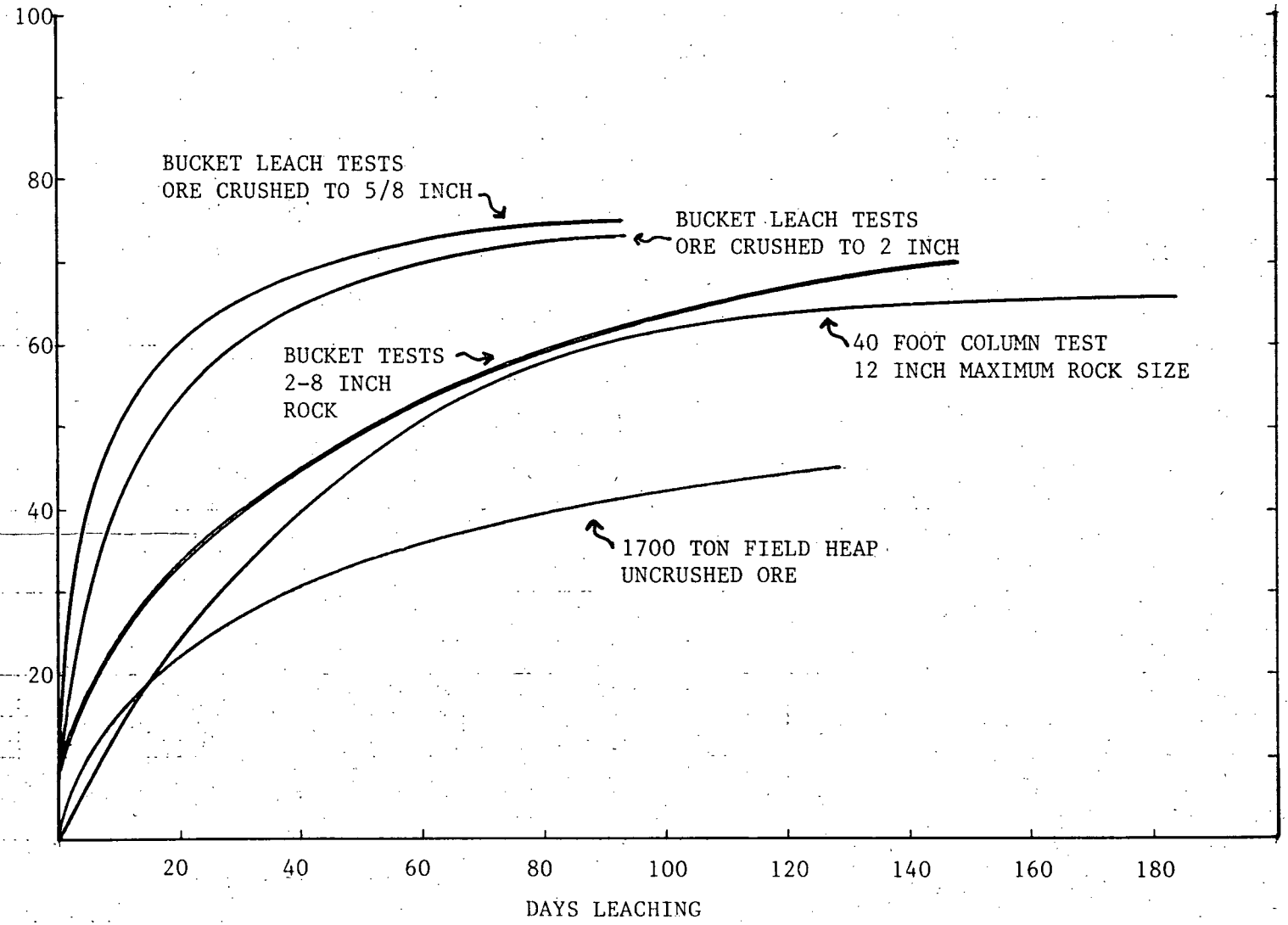

FIGURE 5. GILT EDGE OXIDIZED ORES
RATE OF GOLD RECOVERY
COMPARISON BETWEEN TEST TYPES & ORE SIZES

At the end of the 1980 leach program, an attempt was made to destroy residual cyanide in the heap using sodium hypochlorite. The attempt was only partially successful. During 1981, the heap was re-leached to yield an additional 7 percent recovery of contained gold in 30 days leaching, then another attempt was made to destroy residual cyanide, this time using hydrogen peroxide. The results of this attempt were largely successful. They were outlined in a letter to George Trabits dated 21 June, 1982, which was a proposal for further neutralization (if this was deemed necessary to meet conditions for abandonment of the heap).

The map pocket of this report contains two drawings showing the significant operating statistics for the heap leach test. One drawing graphically presents operating statistics such as cyanide usage, gold recovery, and chemical conditions for the heap operating days in 1980. The second presents data showing chemical consumption and solution assay data for the 1981 neutralization period.

(9) In late 1979, a plane table survey was made of the two existing tailings piles from the old mills. A soils engineering firm was contracted with to drill the tailings, using a large rotary auger drill of the type used for soils sampling. Where the drill rig could not gain access, the tailings were manually sampled. A total of 175 feet of drilling was completed, and 100 samples were taken. The samples were subjected to bottle roll and centrifuge tube tests on pulverized portions. Also, a composite portion of the samples from each auger hole was sent out to determine residual free and total cyanide content.

(10) Nearly all rotary drillhole intervals assaying more than 0.005 ounces gold per ton have been subject to bottle roll tests using pulverized samples. Approximately 40 percent of those same samples were tested for cyanide solubility from unpulverized material (approximately 100 percent minus 1/4-inch, which is the normal size that is created by the drill). The samples were also assayed for cyanide-soluble silver and copper. Results are presented in a report titled "Report 1982 B, Cyanide Solubility", dated 19 March, 1982.

NEUTRALIZATION of CYANIDE in 1981 HEAP SOLUTION

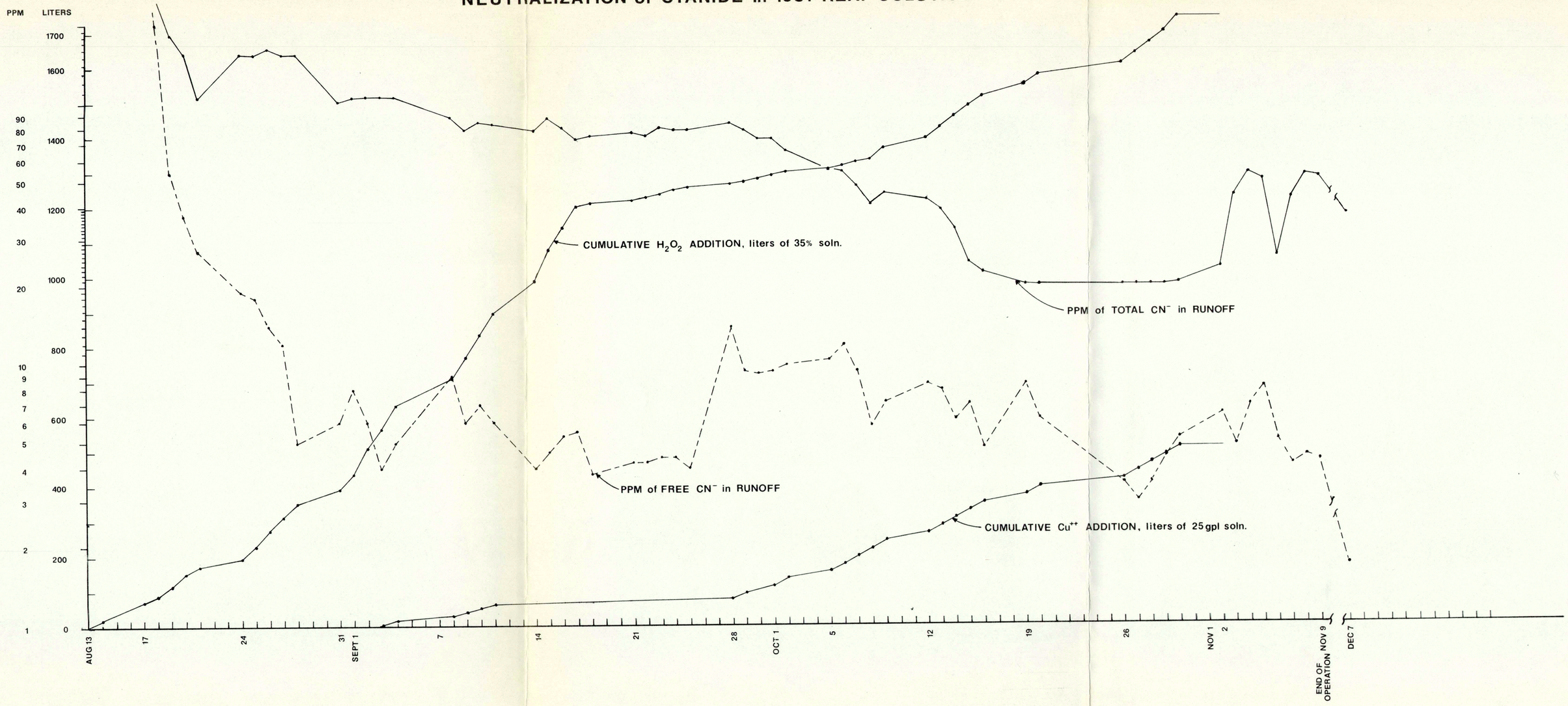

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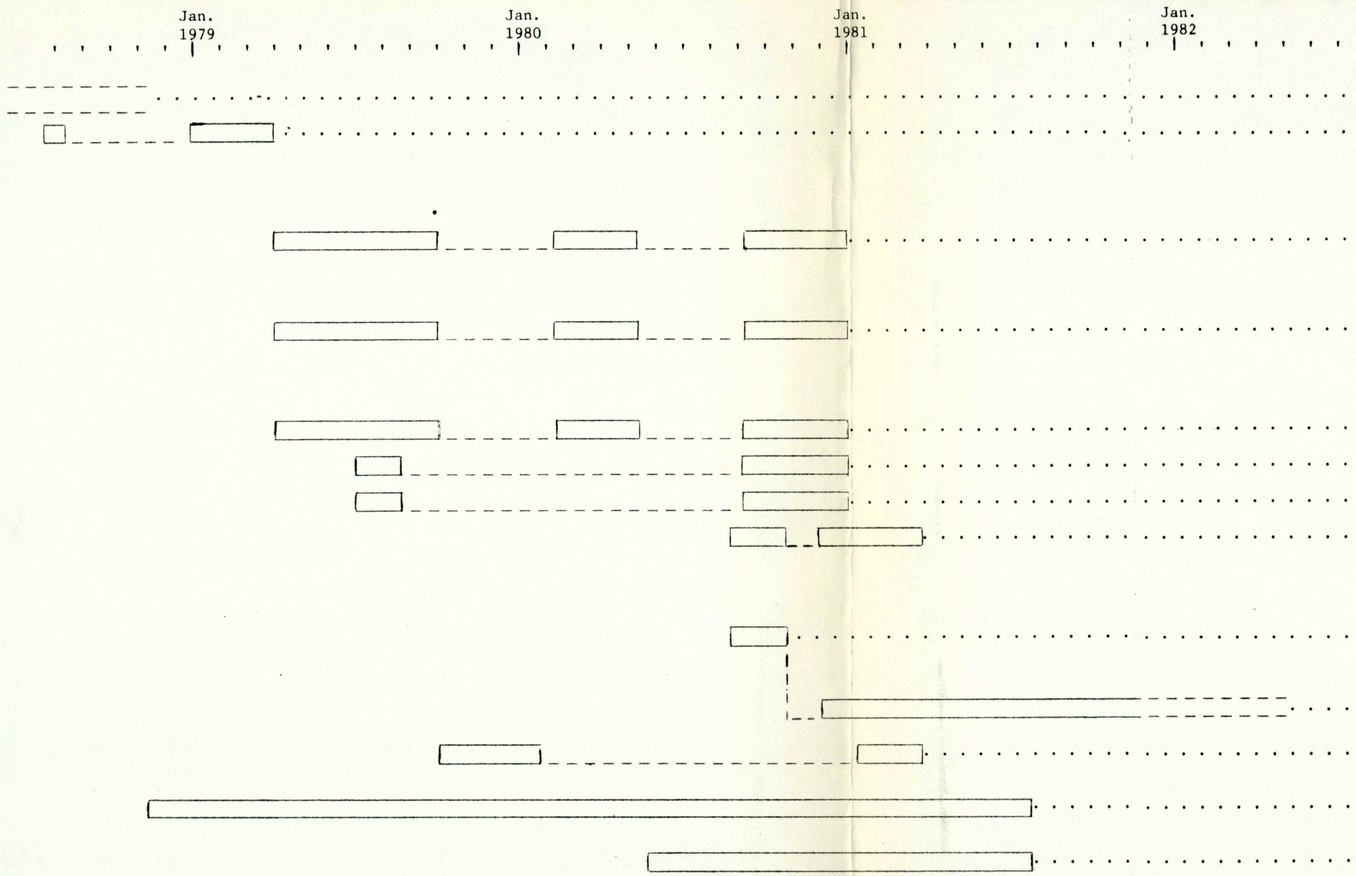
FIGURE 2. GILT EDGE, SOUTH DAKOTA

METALLURGICAL TESTING 1978-1981

HISTORY OF TESTS ON UNOXIDIZED ORES & TAILINGS & RDH CUTTINGS



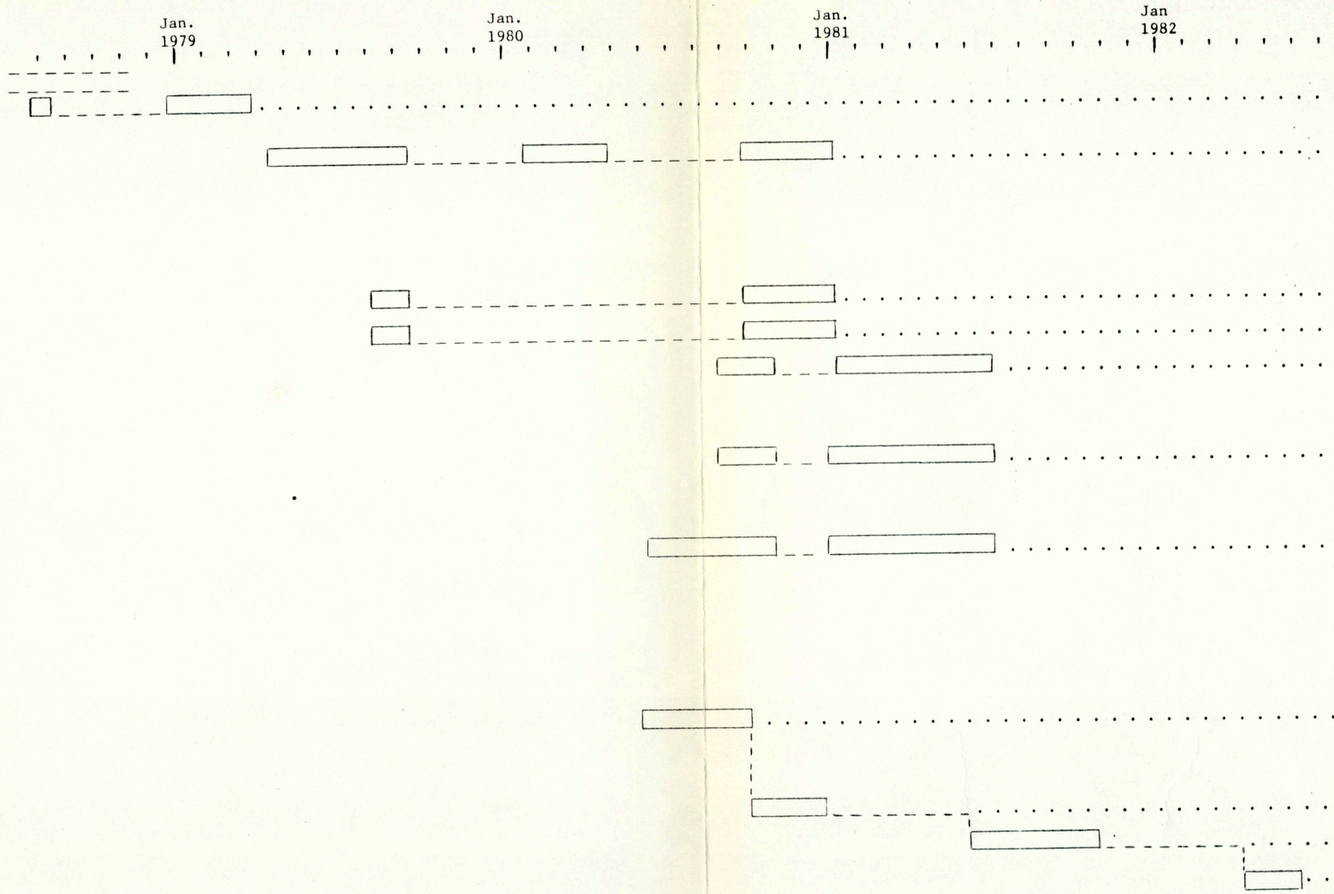
	Percent Gold Recovery in Bucket Tests
Column Leach Tests on RDH cuttings, Cyprus Research Laboratory (1)	
Three 50 - lb Samples from exposed walls, Dakota Maid Pit (2)	one - 60 % *
	two - 30 % *
One 50 - lb Sample from Hoisthouse Level, highly pyritic ore, (2) 100 foot level Sunday Zone	20 % *
Five one-ton Samples from 100 feet below the surface, Sunday Zone (3)	
5 Bucket Leach Tests, crushed to 2-inches	63 %
5 Bucket Leach Tests, crushed to 5/8-inches	63 %
Two one-ton, High Pyrite Samples from 80 feet below surface, Dakota Maid Decline (3)	
2 Bucket Leach Tests, crushed to 2-inches	58 %
2 Bucket Leach Tests, crushed to 5/8-inches	58 %
One 1-ton Low-Pyrite Sample from 80 feet below surface, Dakota Maid Decline (3)	88 %
Two 200-pound Samples from 250 feet below surface, Sunday Zone (4)	62 %
Two 200-pound High Pyrite Samples from Stope Walls, Dakota Maid Pit (4)	47 % *
Five one-ton bulk samples from 80 feet below surface, King Tunnel, (5) Dakota Maid Zone	
5 Bucket Leach Tests, crushed to 2-inches	28 % *
10 Bucket Leach Tests, crushed to 5/8-inches	32 %
One 25-ton High Pyrite Sample from 80 feet below surface, King Tunnel, (6) Dakota Maid Zone	
2 Bucket Leach Tests, crushed to 5/8-inches	52 % *
1 40-foot Column Leach Test on 1-1/2 inch rocks (Recovery based on head assay of the bucket leach tests)	80 %
Agitated Leach Tests on Samples taken from 175 ft of auger drilling and (7) channel samplings in old mill tailings	
500 Bottle Roll Tests on Pulverized Portions, oxidized and unoxidized (10) rotary drillhole cuttings - average recovery 77%	
200 Bottle Roll Tests on Unpulverized (minus 1/4-inch) Portions, rotary (10) drillhole cuttings - average recovery 67%	
*Overall most probable recovery from unoxidized ores, based on average recovery from tests marked with asterik.	43%

(1) All footnotes to this chart are presented in the text of the report.

6 July, 1982
Report 1982 C
Summary Metallurgical Evaluation 1978-1981

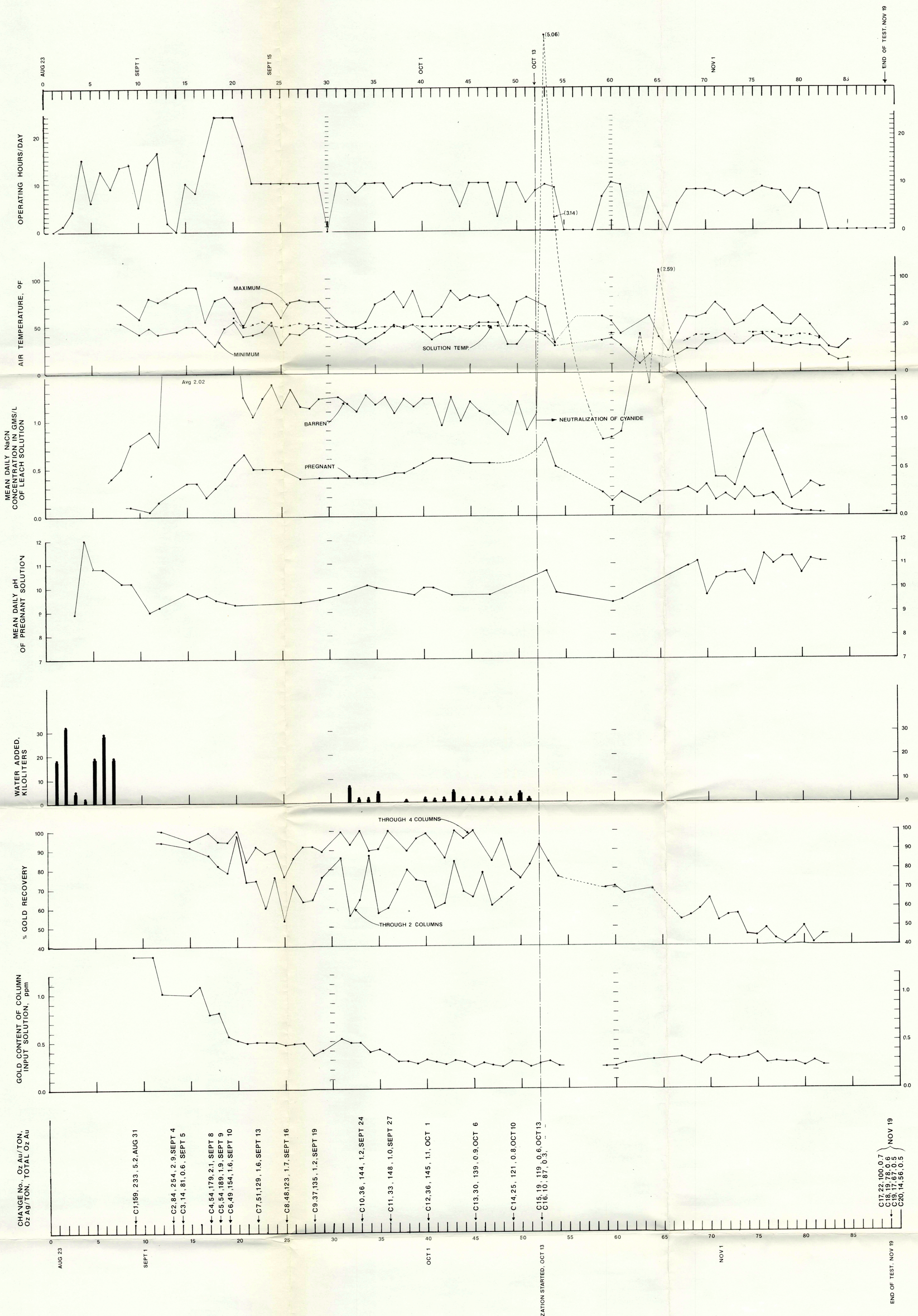
FIGURE 1. GILT EDGE, SOUTH DAKOTA

METALLURGICAL TESTING 1978 - 1981
HISTORY OF TESTS ON OXIDIZED ORES



		Percent Gold Recovery in Bucket Tests
Column Leach Tests on RDH cuttings, Cyprus Research Laboratory (1)		
Four Samples from old exposed walls, 5 Bucket Tests (2)		14 - 70 %
Thirteen one-ton Samples from 100 ft below surface, Sunday Zone (3)		
13 Bucket Leach Tests, crushed to 2-inches		72 % *
13 Bucket Leach Tests, crushed to 5/8-inches		72 %
10 Bucket Leach Tests, @ 2 - 8-inches	67 Day Recovery	55 %
	150 Day Recovery	72 % *
Five 200 Pound Samples from 250 ft below surface, Sunday Zone (4)		69 % *
Three 200 Pound Samples from Dakota Maid Zone and North End of Property (4)		65 % *
Two one-ton Samples from 80 ft below surface, King Tunnel, Dakota Maid Zone (5)		
2 Bucket Leach Tests, crushed to 2-inches		76 % *
2 Bucket Leach Tests, crushed to 5/8-inches		76 %
Two 25 -ton Samples taken for 40-ft Column Leach Tests, Sunday Zone (6)		
5 Bucket Leach Tests, crushed to 1-1/2 inches		76 %
2 40 ft Column Leach Tests on run-of-mine (12-inch rocks)		75 % *
One 25-ton Sample taken for 40 ft Column Leach Test, north Dakota Maid Zone (7)		
1 Bucket Leach Test, crushed to 2-inches	87 Day Recovery	63 %
	87 Day Recovery	67 %
1 Forty-foot Column Leach Test on run of mine (12-inch rocks)		
	87 Day Recovery	58 %
	230 Day Recovery	71 % *
1700 ton Sample taken for Heap Leach Test, Sunday Zone surface (8)		
4 Bucket Leach Tests, crushed to 1-1/2 inches		72 % *
1 1700 ton Field Leach Test on run-of-mine (24-inch rocks)		47 %
130 Day Recovery		
Attempted Neutralization of heap with Sodium Hypochlorite		
Neutralization of heap with Hydrogen Peroxide, decrease of free NACN to 4.0 ppm		
Continuing monitoring of heap, further decrease of free NACN to 2.1 ppm		
*Overall Average Gold Recovery from oxidized ores, all tests marked with asterisk		72 %

(1) All footnotes to this chart are presented in the text of the report.



AMOCO MINERALS CORPORATION

GRAPHICAL PRESENTATION
OF DATA RELATING TO THE

1980 HEAP LEACH TEST
OF THE
GILT EDGE PROJECT

LAWRENCE COUNTY, SOUTH DAKOTA

MILLER-KAPPES Co.
W. F. PLYLEY

NOVEMBER, 1981

CYANIDATION OF GILT EDGE ORE

by

CYPRUS METALLURGICAL PROCESSES CORPORATION

May 12, 1982

P-1000

This X-Copy for Ron Graichen

CYPRUS METALLURGICAL PROCESSES CORPORATION

TUCSON, ARIZONA

FILE NUMBER: 823-42-5001

SUBJECT: CYANIDATION OF GILT EDGE ORE

AUTHOR: Jerry E. Dobson

REPORT DATE: May 12, 1982

May 12, 1982

CYANIDATION OF GILT EDGE ORE

INTRODUCTION

Samples of gold ore from several diamond drill hole cores and composites of the Gilt Edge prospect were received for cyanidation testwork. The furnished samples ranged in gold content from 0.7 to 7.7 ppm gold and from 1.5 to 20 ppm silver content. The leaching tests were directed toward the treatment of agitated ore pulps although some flotation concentrates as well as roasted concentrates were leached. The latter effort resulted from a spate of erroneous assays which led us to conclude mistakenly that the gold value was quite refractory.

EXPERIMENTAL

Sample Preparations

The various ore samples were reduced from the as received condition to about -14 mesh using jaw and roller crushing. Samples of the crushed core specimens were split out for head assays, test samples and a reserve supply.

Further size reduction was carried out in a laboratory steel ball mill or in the instance of concentrates, which were small samples, by hand in a mortar and pestle.

Samples which were roasted were treated in an oven operating at between 600 and 625°C.

CYANIDATION OF GILT EDGE ORE

Analytical

The metal values in both ore residue and solution was monitored by atomic absorption spectroscopy. In the case of gold some difficulties arose which resulted in poor accountability and delayed production of believable extraction data. Some liquor samples, perhaps related to the sulfide content of the ores, seem to undergo a reductive loss of part of their gold content with time. Delays in assaying as short as one day may be serious in the matter of gold accountability under these circumstances.

LEACHING PROCEDURES

Cyanide leachings of ores and concentrates were carried out using the rolled bottle method of agitation. Using untreated ores, 200g samples were employed per test whereas flotation concentrates or roasted concentrate samples were leached on a 20g scale. All leachings were performed on 45% solids in aqueous NaCN slurry.

The concentrations of the metal values developed in the leaching solution were monitored as a function of time. Similarily the consumption of lime and sodium cyanide during the dissolution was measured. The test samples were leached a minimum of 24 hours and occasionally longer.

Records of quantities were kept entirely by weight necessitating only that a thorough washing of solid be achieved to have accuracies within the limits of the assay precision.

CYANIDATION OF GILT EDGE ORE

Flotation

Each ore sample was subjected to a rough flotation expected to recover its pyritic fractions. Samples were ground to an intermediate size in seven minutes of grinding, the pH adjusted to the range from 7.5 to 8.5 using Na_2CO_3 and brought to ca. 30% solids. A total dose of 0.1 lb/ton i-amylxanthate was added over a ten minute flotation time and MIBC was used as needed for froth. Head, concentrate, and tails assays indicated typical recoveries of about >80% of the gold content and 60% of silver.

RESULTS

Table I presents the referencing identification for the Amoco Minerals Company's sample designation and the letter identification assigned for convenience by Cymet. For quick reference the overall performance of the leaching of gold from each of the various samples under the several conditions employed in this study is also reported. The degree of grinding is designated by the series A thru E for each sample in order of increasing grind time. The split at 200 mesh was measured and is keyed at the bottom of the table. The quantity of gold developed in the cyanide leachate for these various conditions is reported in the fourth column in ounces per ton of ore leached and was based upon the quantity of gold detected in solutions after the cyanidation reaction. As to leaching efficiency, the calculated head derived from product assays was used to determine the percentage reported in the fifth column. Finally, the sixth column of Table 1 records the

consumption of NaCN per ton of ore.

The remaining tables detail the individual leachings including not only the gold results but silver and copper extractions as well.

Table II sets out the cyanidation efficiencies which were found for several samples of Gilt Edge ore when cyanidation was tested on coarsely ground materials. Because of the sometimes difficulty in accountability, perhaps because of coarse gold, we report two extraction values in this numerical tabulation. The first is the extraction based upon the average gold content of the head samples; the second extraction column is based upon the level of gold found in that particular sample's leached products, i.e. a calculated head basis. The third column lists the mass balance across the leaching process from average head composition to leached tailings and liquor levels. Similarily, Columns 4, 5, and 6 report the corresponding results calculated for the silver content of the ore which, though generally low, were also monitored. Strong cyanide extraction, 0.2%, of the gold from these rather coarsely crushed samples established a base with which to compare other conditions. This was the most coarsely crushed of the samples measuring about 22% -200 mesh fraction. The recovery was inadequate, averaging only 51%.

A somewhat different format is employed in Tables III through VI and VIII to take advantage of a computer printout. It is self explanatory in large, but contains more information. The columns under the heading Assays give the nead, leach liquor,

tails and calculated head values. Extractions are reported based both upon calculated head values, which are preferred, as well as head values which are included for the sake of confidence as well as a measure by which to gauge the balances.

Table III summarizes the leaching results for an intermediate grind of the ores. A typical screen analysis in this sample set yielded 45% - 200 mesh fraction. This resulted in a substantially better degree of leaching than was given by the coarsely ground samples in Table II. The extractions averaged 74% based upon calculated head values and 74% as a gold weighted average as well.

Table IV and V are the result of yet finer grinding at 65% and 70% -200 mesh respectively. This spacing is closer than planned but the data of both are included to increase the data base. The only difference, other than the marginal size distribution change, was that the NaCN level of Table V (70% -200 mesh) was reduced to 0.05% to verify the usual lack of effect of CN^- concentration upon leaching kinetics in the ranges being employed. As may be seen from the individual tests and the weighed averages presented in Table I the lowered cyanide level may have had some effect, but this is primarily due to depletion between samplings rather than a bona fide kinetic rate effect, i.e. the reaction time was truncated by reagent consumption. The extractions in Table IV and V calculated as a straight average were 79 and 71% respectively. Calculated weighted average based upon contained gold values were 83 and 74%.

CYANIDATION OF GILT EDGE ORE

Table VI furnishes the data of the cyanidation behavior of the most finely ground set of samples, corresponding to test E of the summary Table I. These leachings attempted to remove particle size from consideration as a limiting factor in dissolution. All were subjected to twenty minutes grinding in the steel mill and reported >98% as a -200 mesh fraction, in fact they were >95% -325 mesh. The background cyanide level was restored to 0.2% NaCN in order to handle any increase in copper and acid activity resulting from enhanced oxidation at this very fine state of subdivision. The average extraction was 76% as a straight average and 80% as a weighted average. This seems to be biased by two very poor performances by samples D and K in this experiment.

Table VII assembles the data concerning the small investigation of concentrating the ore. This is included for completeness, however, the reason for its existence was based upon some assay difficulties which, when resolved, faded along with the need of concentration. As mentioned earlier, no optimization of flotation recovery was attempted, merely a rougher concentration in order to attain sufficient concentrate for testing. Any assessment of concentration or concentration and roasting as possible processing steps would require additional evaluation. In this table we report the overview of the results obtained in concentrating and concentrate leaching the gold from each ore sample. A composite ore sample was also processed through each operation. The final entry in the table gives the weighed average gold extraction <u>from the concentrates</u> D thru L.

CYANIDATION OF GILT EDGE ORE

Table VIII provides the detailed test by test data from the leaching of the concentrates and the composite concentrate. Also included here is the leaching behavior of the composite concentrate after four hours roasting at 600°C which reduced the sulfur content from >30 to >2%. The format of this table parallels those given as III through VI.

The figure presented gives the extraction curve for gold as a function of the fineness of grind. The general feature is obvious and expected in the indication of higher extraction resulting from increasing particle subdivision. A principle feature would appear to be the rapid increase in leachability as the quantity of -200 mesh material increased from about 20% to 45%; further grinding did not dramatically affect recovery (see weighted average extraction for A thru E grinds in Table I). The latter two data points for grinds D and E may, however, as mentioned before, somewhat underestimate extraction. If so, the flattening of the curve should not be as promounced as portrayed in the Figure. The recoverable upper limit of gold from this ore may thus approach 90% under the conditions employed here.

May 12, 1982

SUMMARY

The cyanidation of Gilt Edge ore in an agitated leaching operation may be expected to yield about 75 to 80% of the gold content or 0.046 oz/ton on a weighted average basis. This assumes a grinding to about 70% -200 mesh; marginal increase in yield to the 85% range might be expected with the samples ground to 100% -200 mesh. The consumption of sodium cyanide, of course, increases with the fineness of grind reaching an average of 72 lb/oz Au in our most finely ground samples. Lime demand, on the other hand, showed only modest increases during the same experiments; about 100-110 lb/oz Au is required.

The brief examination of concentrates revealed that flotation may easily recover ca. 85% of the gold value and that upon cyanidation approximately 84% of this is recoverable. The net yield then is about 72% with the advantage of about 90% less bulk to be treated. The average grade of concentrate treated was 14.9 ppm Au yielding 0.37 oz Au/Ton of concentrate. Cyanide consumption was approximately the same as the unconcentrated ore at 30-40 lb/oz Au; lime usage decreased sharply to about 13 lb/oz Au.

The leaching of the roasted concentrate gave significantly greater recovery of 97% of the gold as expected since the occlusion of particles in the pyrite matrix is probably responsible for their inactivity.

Roasting in conjunction with flotation will recover about 82% of the gold value with reduced cyanide and grinding costs. These factors in addition to the size reduction of concentrate handling facilities may justify more thorough evaluation of the flotation recovery limits.

CYANIDATION OF GILT EDGE ORE

TABLE I

Sample Referencing and Overview of Results

Sample					Leaching Behavior
Cymet Letter	AMOCO Designation		Gold Recovery OPT	Gold Recovery %	NaCN Consumed #T
D	GLE Composite DDH#21 50'-444"	A			
		B	0.028	71	2.6
		C	0.028	71	2.8
		D	0.025	63	3.4
		E	0.018	43	4.5
E	GLE Composite DDH#22 450'-740'	A	0.059	61	2.0
		B	0.053	79	0.4
		C	0.064	85	0.4
		D	0.068	77	0.6
		E	0.060	91	3.3
F	GLE Composite DDH#22 80'-180' & 320'-450'	A	0.014	41	0.5
		B	0.021	55	0.8
		C	0.025	68	1.2
		D	0.021	55	3.0
		E	0.028	71	4.6
G	81 DDH-6 610'-680' Sample "A"	A	0.122	57	0.6
		B	0.132	76	0
		C	0.160	90	0.3
		D	0.146	82	0.4
		C	0.160	96	2.5
H	81 DDH-6 680'-775' Sample "B"	A	0.029	46	0.3
		B	0.046	76	2.4
		C	0.043	83	0.3
		D	0.046	73	1.6
		E	0.053	95	1.8
I	81 DDH-16 100'-200' Sample "A"	A			
		B	0.021	88	1.7
		C	0.021	88	1.9
		D	0.021	88	1.0
		E	0.021	88	1.8

CYANIDATION OF GILT EDGE ORE

TABLE I (con't)

Sample Referencing and Overview of Results

Sample					Leaching Behavior
Cymet Letter	AMOCO Designation		Gold Recovery OPT	Gold Recovery %	NaCN Consumed #T
J	81 DDH-16 240'-295' Sample "A"	A			
		B	0.032	85	1.5
		C	0.046	89	0.3
		D	0.036	86	1.8
		E	0.036	92	2.6
K	81 DDH-17 9' - 196'	A			
		B	0.021	71	3.3
		C	0.021	71	2.6
		D	0.014	49	3.6
		E	0.007	26	4.6
L	81 DDH-17 196'391' Sample "A"	A			
		B	0.025	68	1.1
		C	0.025	68	1.2
		D	0.036	71	1.1
		E	0.028	83	3.4
D thru L - Weighted Average of all Samples		A	0.054	51%*	Only D,E,F&G
		B	0.042	74%	
		C	0.048	83%	
		D	0.046	74%	
		E	0.044	80%	

Grind A 22% -200 mesh
B 45% -200 mesh
C 65% -200 mesh
D 70% -200 mesh
E 98% -200 mesh

TABLE II

(Gilt Edge Ore Cyanidation)

Sample	% Au Ext H	-CH	% Bal	% Ag Ext H	-CH	% Bal	# NaCN / T	# CaO / T
E	60	61	98	58	21	282	2	8
F	32	41	78	54	56	96	0.5	8
G	54	56	96	35	35	100	0.6	6.7
H	45	46	98	16	16	101	0.3	6.7

Fire Assay Au

E	100	71	141
F	55	39	72
G	73	70	104
H	56	72	77

Typical screen analysis: 36.1%-65; 22.2%-200

NaCN: 0.2%
Time: 24 hours
H: Based on head assay
CH: Based on calculated head assay.

40 TO 50%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

TABLE III

Test ID	*	Assays PPM Ag	Assays PPM Au	Assays PPM Cu	% Extraction Ag	% Extraction Au	% Extraction Cu	% Balance Ag	% Balance Au	% Balance Cu	Reagent consumptio #/T CN	Reagent consumptio #/T CaO
D	H	3.6	1.6	295.	58.	61.	64.	99.	86.	113.		
	L	1.7	0.8	154.								
	T	1.5	0.4	146.								
	C	3.6	1.4	334.	58.	71.	56.				2.6	4.9
	OPT				0.060	0.028	0.38@					
E	H	2.1	3.4	114.	46.	54.	30.	118.	69.	98.		
	L	0.8	1.5	28.								
	T	1.5	0.5	78.								
	C	2.5	2.3	112.	39.	79.	30.				0.4	4.7
	OPT				0.028	0.053	0.07@					
F	H	6.0	1.5	233.	53.	49.	57.	498.	89.	102.		
	L	2.6	0.6	109.								
	T	26.7	0.6	104.								
	C	29.9	1.3	237.	11.	55.	56.				0.8	4.5
	OPT				0.093	0.021	0.27@					
G	H	6.1	7.7	183.	34.	59.	11.	509.	77.	112.		
	L	1.7	3.7	16.								
	T	29.0	1.4	186.								
	C	31.1	5.9	206.	7.	76.	9.				-0.0	4.6
	OPT				0.060	0.132	0.04@					
H	H	5.9	2.1	298.	17.	76.	9.	95.	99.	116.		
	L	0.8	1.3	22.								
	T	4.6	0.5	318.								
	C	5.6	2.1	345.	18.	76.	8.				2.4	4.8
	OPT				0.028	0.046	0.05@					
I	H	1.5	1.0	49.	49.	73.	25.	155.	83.	147.		
	L	0.6	0.6	10.								
	T	1.6	0.1	60.								
	C	2.3	0.8	72.	31.	88.	17.				1.7	4.9
	OPT				0.021	0.021	0.02@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

40 TO 50%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumption #/T CN	Reagent consumption #/T CaO
J	H	2.7	1.6	152.	63.	69.	9.	137.	81.	81.		
	L	1.4	0.9	11.								
	T	2.0	0.2	109.								
	C	3.7	1.3	122.	46.	85.	11.				1.5	4.9
	OPT				0.050	0.032	0.03@					
K	H	19.9	1.4	736.	61.	52.	63.	98.	81.	108.		
	L	9.9	0.6	383.								
	T	7.4	0.4	329.								
	C	19.5	1.1	796.	62.	65.	59.				3.3	4.9
	OPT				0.352	0.021	0.93@					
L	H	4.6	1.6	255.	48.	53.	68.	2863.	78.	105.		
	L	1.8	0.7	142.								
	T	129.5	0.4	94.								
	C	131.7	1.3	267.	2.	68.	65.				1.1	4.7
	OPT				0.064	0.025	0.35@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

approx.65%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

TABLE IV

Test ID	*	Assays PPM Ag	Assays PPM Au	Assays PPM Cu	% Extraction Ag	% Extraction Au	% Extraction Cu	% Balance Ag	% Balance Au	% Balance Cu	Reagent consumption #/T CN	Reagent consumption #/T CaO
D	H	3.6	1.6	295.	61.	61.	67.	83.	86.	101.		
	L	1.8	0.8	163.								
	T	0.8	0.4	99.								
	C	3.0	1.4	298.	73.	71.	67.				2.8	5.0
	OPT				0.064	0.028	0.40@					
E	H	2.1	3.4	114.	46.	65.	45.	85.	76.	127.		
	L	0.8	1.8	42.								
	T	0.8	0.4	93.								
	C	1.8	2.6	144.	55.	85.	36.				0.4	4.7
	OPT				0.028	0.064	0.10@					
F	H	6.0	1.5	233.	53.	57.	64.	81.	84.	108.		
	L	2.6	0.7	123.								
	T	1.7	0.4	102.								
	C	4.9	1.3	252.	65.	68.	60.				1.2	4.7
	OPT				0.093	0.025	0.30@					
G	H	6.1	7.7	183.	36.	71.	23.	88.	79.	113.		
	L	1.8	4.5	35.								
	T	3.2	0.6	165.								
	C	5.4	6.1	208.	41.	90.	21.				0.3	4.7
	OPT				0.064	0.160	0.09@					
H	H	5.9	2.1	298.	21.	70.	8.	129.	84.	107.		
	L	1.0	1.2	20.								
	T	6.4	0.3	295.								
	C	7.6	1.8	319.	16.	83.	8.				0.3	4.5
	OPT				0.036	0.043	0.05@					
I	H	1.5	1.0	49.	73.	73.	32.	153.	83.	171.		
	L	0.9	0.6	13.								
	T	1.2	0.1	68.								
	C	2.3	0.8	84.	48.	88.	19.				1.9	4.9
	OPT				0.032	0.021	0.03@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

approx.65%-200 mesh,0.2% NaCN,5#/T CaO,24 hrs.

Test ID	*	Assays PPM As	Assays PPM Au	Assays PPM Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumptio #/T CN	Reagent consumptio #/T CaO
J	H	2.7	1.6	152.	68.	99.	10.	90.	112.	82.		
	L	1.5	1.3	12.								
	T	0.6	0.2	110.								
	C	2.4	1.8	125.	75.	89.	12.				0.3	4.9
	OPT				0.053	0.046	0.03@					
K	H	19.9	1.4	736.	68.	52.	69.	115.	74.	111.		
	L	11.1	0.6	418.								
	T	9.3	0.3	309.								
	C	22.8	1.0	819.	59.	71.	62.				2.6	4.8
	OPT				0.395	0.021	1.02@					
L	H	4.6	1.6	255.	56.	53.	70.	110.	78.	101.		
	L	2.1	0.7	147.								
	T	2.5	0.4	78.								
	C	5.1	1.3	257.	51.	68.	70.				1.2	4.7
	OPT				0.075	0.025	0.36@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Approx. 70% -200 mesh, 0.05% NaCN, 24 hrs.

TABLE V

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumption #/T NaCN	#/T Cal
D	H	3.6	1.6	295.	64.	54.	60.	128.	86.	97.		
	L	1.9	0.7	146.								
	T	2.3	0.5	109.								
	C	4.6	1.4	287.	50.	63.	62.				3.4	5.
	OPT				0.068	0.025	0.36@					
E	H	2.1	3.4	114.	64.	68.	41.	187.	89.	113.		
	L	1.1	1.9	38.								
	T	2.6	0.7	83.								
	C	3.9	3.0	129.	34.	77.	36.				0.6	5.
	OPT				0.039	0.068	0.09@					
F	H	6.0	1.5	233.	59.	48.	69.	98.	87.	97.		
	L	2.9	0.6	131.								
	T	2.3	0.6	66.								
	C	5.8	1.3	226.	61.	55.	71.				3.0	5.
	OPT				0.103	0.021	0.32@					
G	H	6.1	7.7	183.	42.	65.	13.	102.	79.	102.		
	L	2.1	4.1	19.								
	T	3.7	1.1	163.								
	C	6.3	6.1	186.	41.	82.	12.				0.4	5.
	OPT				0.075	0.146	0.05@					
H	H	5.9	2.1	298.	23.	76.	16.	96.	104.	102.		
	L	1.1	1.3	40.								
	T	4.3	0.6	254.								
	C	5.6	2.2	303.	24.	73.	16.				1.6	6.
	OPT				0.039	0.046	0.10@					
I	H	1.5	1.0	49.	75.	76.	17.	137.	87.	124.		
	L	0.9	0.6	7.								
	T	0.9	0.1	52.								
	C	2.0	0.8	61.	55.	88.	14.				1.0	6.
	OPT				0.032	0.021	0.02@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Approx. 70% -200 mesh, 0.05% NaCN, 24 hrs.

Test ID	*	Assays ppm Ag	Assays ppm Au	Assays ppm Cu	% Extraction Ag	% Extraction Au	% Extraction Cu	% Balance Ag	% Balance Au	% Balance Cu	Reagent consumptio #/T CN	Reagent consumptio #/T CaO
J	H	2.7	1.6	152.	72.	76.	14.	112.	88.	84.		
	L	1.6	1.0	18.								
	T	1.1	0.2	105.								
	C	3.1	1.4	127.	64.	86.	17.				1.8	6.0
	OPT				0.057	0.036	0.04@					
K	H	19.9	1.4	736.	75.	36.	70.	108.	73.	104.		
	L	12.3	0.4	422.								
	T	6.4	0.5	254.								
	C	21.4	1.0	769.	70.	49.	67.				3.6	5.9
	OPT				0.438	0.014	1.03@					
L	H	4.6	1.6	255.	67.	75.	69.	111.	106.	104.		
	L	2.5	1.0	144.								
	T	2.0	0.5	89.								
	C	5.1	1.7	265.	60.	71.	66.				1.1	5.8
	OPT				0.069	0.036	0.35@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Approx. 96% -325 mesh, .2% NaCN, 24 hrs.

TABLE VI

Test ID	*	Assays ppm As	Assays ppm Au	Assays ppm Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumptio $/T CN	Reagent consumptio $/T CaO
D	H	3.6	1.6	295.	47.	39.	71.	100.	90.	117.		
	L	1.4	0.5	172.								
	T	1.9	0.8	136.								
	C	3.6	1.4	346.	47.	43.	61.				4.5	4.9
	OPT				0.050	0.018	0.42@					
E	H	2.1	3.4	114.	64.	61.	19.	116.	67.	127.		
	L	1.1	1.7	18.								
	T	1.1	0.2	123.								
	C	2.4	2.3	145.	55.	91.	15.				3.3	5.4
	OPT				0.039	0.060	0.04@					
F	H	6.0	1.5	233.	39.	64.	87.	88.	90.	115.		
	L	1.9	0.8	167.								
	T	2.9	0.4	64.								
	C	5.2	1.4	268.	44.	71.	76.				4.6	5.5
	OPT				0.068	0.028	0.41@					
G	H	6.1	7.7	183.	42.	71.	26.	101.	74.	110.		
	L	2.1	4.5	39.								
	T	3.6	0.2	153.								
	C	6.2	5.7	201.	42.	96.	24.				2.5	5.9
	OPT				0.075	0.160	0.10@					
H	H	5.9	2.1	298.	21.	87.	16.	100.	92.	104.		
	L	1.0	1.5	40.								
	T	4.7	0.1	261.								
	C	5.9	1.9	310.	21.	95.	16.				1.8	5.9
	OPT				0.036	0.053	0.10@					
I	H	1.5	1.0	49.	92.	76.	52.	112.	87.	205.		
	L	1.1	0.6	21.								
	T	0.3	0.1	75.								
	C	1.6	0.8	101.	82.	88.	25.				1.8	4.8
	OPT				0.039	0.021	0.05@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

Approx. 96% -325 mesh, .2% NaCN, 24 hrs.

Test ID	*	Assays PPM As	Assays PPM Au	Assays PPM Cu	% Extraction As	% Extraction Au	% Extraction Cu	% Balance As	% Balance Au	% Balance Cu	Reagent consumption $/T CN	Reagent consumption #/T CaO
J	H	2.7	1.6	152.	94.	76.	31.	131.	82.	109.		
	L	2.1	1.0	38.								
	T	1.0	0.1	120.								
	C	3.6	1.3	166.	72.	92.	28.				2.6	4.8
	OPT				0.075	0.036	0.09@					
K	H	19.9	1.4	736.	10.	18.	55.	98.	70.	100.		
	L	1.6	0.2	331.								
	T	17.6	0.7	330.								
	C	19.6	0.9	734.	10.	26.	55.				4.6	4.9
	OPT				0.057	0.007	0.81@					
L	H	4.6	1.6	255.	72.	60.	80.	107.	72.	107.		
	L	2.7	0.8	168.								
	T	1.6	0.2	67.								
	C	4.9	1.2	272.	67.	83.	75.				3.4	4.9
	OPT				0.096	0.028	0.41@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

CYANIDATION OF GILT EDGE ORE

TABLE VII

Flotation

Sample	Float Recovery (%)		Grade (ppm)		Leachability (%)		
	Au	Ag	Au	Ag	Au	Ag	#/TNaCN
D	77	79	9.0	22.1	73	55	11.1
E	82	87	18.8	22.5	84	52	10.8
F	88	71	7.5	20.6	68	59	11.0
G	79	86	55.3	43.8	88	39	10.5
H	95	72	21.8	51.9	81	69	10.7
I	45	-	8.0	18.3	95	79	12.3
J	51	39	13.3	17.9	97	84	11.8
K	75	57	5.5	61.8	58	48	11.6
L	79	36	7.5	18.6	73	63	11.2
Gold Weighted Average D thru L					84	59	11.2
Composite Conc.	85	70	15.8	30.3	83	59	11.0
Roasted Compo Conc			19.9	35.4	97	23	4.7

GLE concs., 0.45% NaCN, 5#/T CaO, 24 hrs.

TABLE VIII

Test ID	*	Assays PPM Ag	Assays PPM Au	Assays PPM Cu	% Extraction Ag	% Extraction Au	% Extraction Cu	% Balance Ag	% Balance Au	% Balance Cu	Reagent consumptio #/T CN	Reagent consumptio #/T CaO
D	H	22.1	9.0	1334.	44.	68.	74.	79.	93.	93.		
	L	7.9	5.0	809.								
	T	7.8	2.3	250.								
	C	17.4	8.4	1237.	55.	73.	80.				11.1	4.9
	OPT				0.281	0.178	1.97@					
E	H	22.5	18.8	735.	31.	83.	42.	61.	98.	109.		
	L	5.8	12.8	251.								
	T	6.6	2.9	496.								
	C	13.7	18.5	802.	52.	84.	38.				10.8	4.9
	OPT				0.206	0.455	0.61@					
F	H	20.6	7.1	904.	64.	67.	70.	108.	98.	104.		
	L	10.8	3.9	521.								
	T	9.1	2.2	309.								
	C	22.3	7.0	945.	59.	68.	67.				11.0	4.9
	OPT				0.384	0.139	1.27@					
G	H	43.8	49.1	1862.	38.	78.	9.	99.	88.	106.		
	L	13.7	31.5	134.								
	T	26.5	5.0	1816.								
	C	43.2	43.4	1979.	39.	88.	8.				10.5	4.9
	OPT				0.487	1.121	0.33@					
H	H	51.9	21.8	3914.	194.	86.	5.	282.	106.	98.		
	L	82.5	15.3	148.								
	T	45.9	4.4	3661.								
	C	146.6	23.1	3842.	69.	81.	5.				10.7	4.9
	OPT				2.936	0.544	0.36@					
I	H	18.3	8.0	356.	23.	89.	44.	29.	94.	339.		
	L	3.4	5.8	129.								
	T	1.1	0.4	1049.								
	C	5.3	7.5	1207.	79.	95.	13.				12.3	6.1
	OPT				0.121	0.207	0.32@					

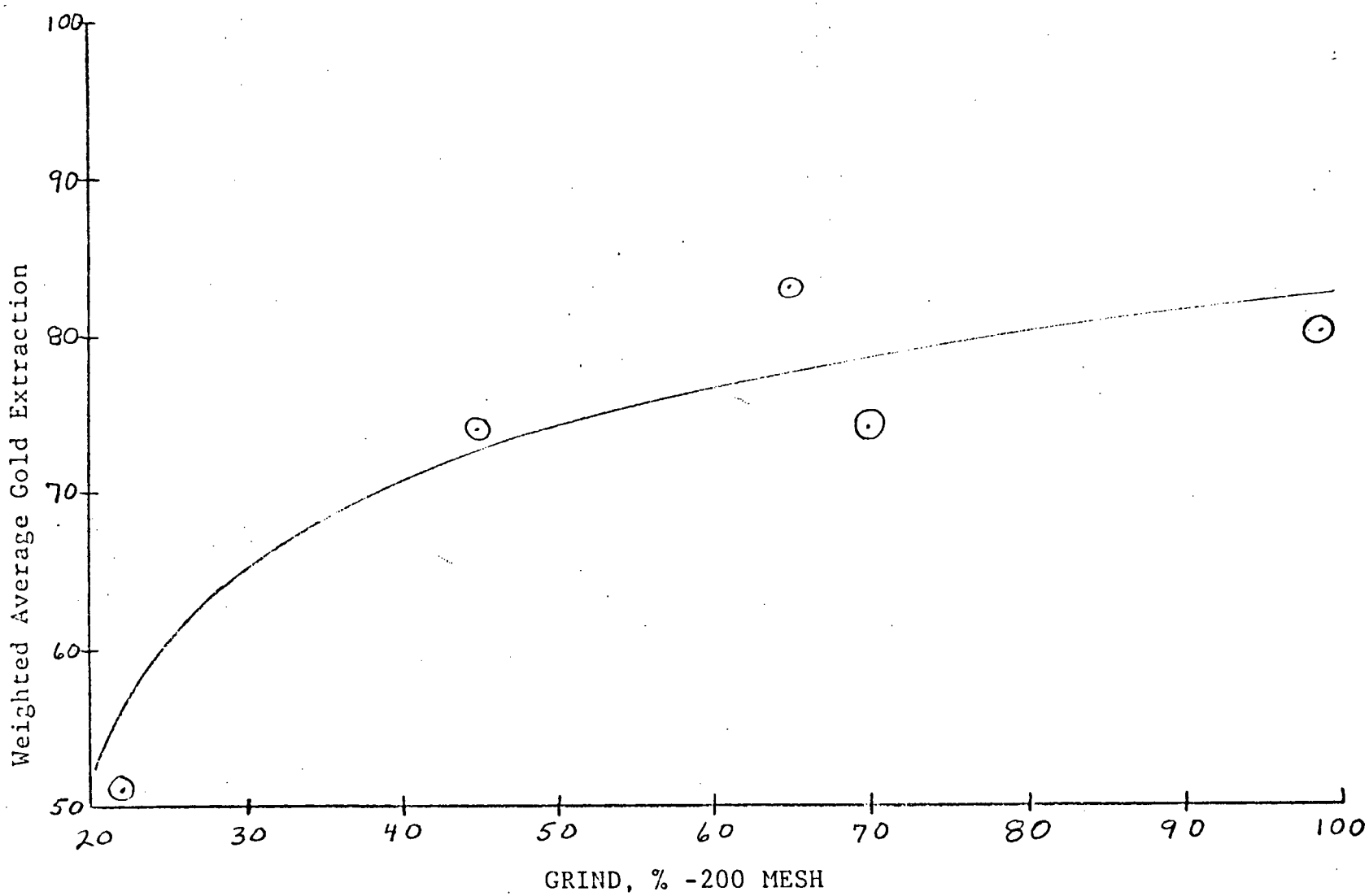
* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

GLE concs., 0.45% NaCN, 5#/T CaO, 24 hrs.

Test ID	*	Assays ppm Ag	Assays ppm Au	Assays ppm Cu	% Extraction Ag	% Extraction Au	% Extraction Cu	% Balance Ag	% Balance Au	% Balance Cu	Reagent consumption #/T CN	Reagent consumption #/T CaO
J	H	17.9	13.3	407.	51.	103.	35.	60.	106.	227.		
	L	7.4	11.2	115.								
	T	1.7	0.4	784.								
	C	10.7	14.1	924.	84.	97.	15.				11.8	5.9
	OPT				0.264	0.399	0.28@					
K	H	61.8	5.5	2423.	54.	51.	80.	113.	87.	97.		
	L	27.3	2.3	1584.								
	T	36.6	2.0	419.								
	C	69.9	4.8	2351.	48.	58.	82.				11.6	4.9
	OPT				0.971	0.082	3.86@					
L	H	18.6	7.5	1362.	65.	73.	83.	103.	100.	98.		
	L	9.9	4.5	930.								
	T	7.0	2.0	203.								
	C	19.1	7.5	1338.	63.	73.	85.				11.2	4.9
	OPT				0.352	0.160	2.27@					
Cmp	H	30.3	15.1	1646.	60.	69.	51.	102.	83.	89.		
	L	14.9	8.6	686.								
	T	12.7	2.1	626.								
	C	30.9	12.6	1463.	59.	83.	57.				11.0	4.9
	OPT				0.530	0.306	1.67@					

* H=head L=liquor T=tail C=calculated head OPT=tr.oz./Ton
@=lbs./Ton

FIGURE: GOLD EXTRACTION VS GRIND



CYANIDE SOLUBILITY
GOLD FROM CRUSHED AND PULVERIZED ORES IN AGITATED TESTS AND SILVER AND COPPER IN AGITATED AND BUCKET LEACH TESTS

by

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GILT EDGE, SOUTH DAKOTA

REPORT 1982 B

CYANIDE SOLUBILITY

GOLD FROM CRUSHED AND PULVERIZED ORES IN AGITATED TESTS
AND
SILVER AND COPPER IN AGITATED AND BUCKET LEACH TESTS

INTRODUCTION

During the period 1979-1981, approximately 500 rotary drillhole cuttings from Gilt Edge were tested in Reno for cyanide soluble gold, silver and copper. The samples tested included nearly all intervals above 0.02 ounces gold per ton, and several intervals below this level.

For approximately 300 of the samples only pulverized portions were tested. Approximately 200 of the samples were tested both pulverized and unpulverized.

The goals of these tests were as follows:

1. To measure the ability to use gold solubility from agitated tests on small laboratory samples, as a method of guiding mine operations for production heap leaches.

2. To identify areas of the orebody in which the leach behavior of the ore varies significantly from the norm, and to measure cyanide consumption in agitated leach tests.

3. To determine the relationship between silver as reported in fire assays, cyanide soluble silver in agitated tests, and cyanide soluble silver in bucket leach tests, as a guide to designing the recovery processes.

4. To compare the solubility of copper in agitated tests on pulverized samples, and in bucket leach tests on coarse-crushed samples. This data also provides a guide for design of conventional milling operations, and a comparison of recoveries and solution chemistry between the mill and the heap leach.

5. To determine distribution of cyanide soluble copper, as a geologic and metallurgical guide to zones of secondary copper enrichment.

This report is primarily a presentation of the laboratory results, which are listed in Appendices A and B. The report itself, briefly presents some conclusions which can be drawn from the data as they apply to each of the goals listed above. A final section outlines the test procedures used.

1. AGITATED TESTS AS A PRODUCTION GUIDE

A previous report, titled "Gilt Edge Field Sampling and Laboratory Tests - 1979 - May 1980", and dated 10 August, 1981, included a list of the gold recoveries in agitated tests on 479 samples. The list is not reproduced here, although the gold solubilities from that list are presented in Appendix A. Figure 1 of this report summarizes the results.

Level of Gold Recovery. The tests indicate that the gold recovery from pulverized samples averages 76 percent of the fire assayable gold. There is a distinct difference between the Dakota Maid Zone (70 percent of the fire assay values) and the Sunday Zone (82 percent).

Number of Samples Required. Approximately 200 of the rotary drillhole cuttings, selected from the sample population exceeding 0.02 ounces gold per ton, were tested twice, once using 100 grams pulverized material and once using 100 grams unpulverized drillhole cuttings (mostly minus 1/4-inch). Gold recoveries from these tests are listed in Appendix B of this report. Figure 2 presents a summary of the results, differentiated between the Dakota Maid and Sunday Zone orebodies. Tailings of all tests were fire assayed so recovery results could be based on actual gold content. The average gold content based on calculated head assays, was essentially the same for both pulverized and unpulverized samples.

FIGURE 1. GILT EDGE: GOLD RECOVERY FROM AGITATED LEACH TESTS

24 HOUR TESTS ON PULVERIZED ROTARY DRILLHOLE CUTTINGS

INTERVAL FEET	DAKOTA MAID ZONE NO. SAMPLES TESTED	DAKOTA MAID ZONE CYANIDE SOLUBLE Au oz/ton	DAKOTA MAID ZONE CALC. HEAD Au oz/ton	SUNDAY ZONE NO. SAMPLES TESTED	SUNDAY ZONE CYANIDE SOLUBLE Au oz/ton	SUNDAY ZONE CALC. HEAD Au oz/ton
0 - 50	65	.051	.0565	53	.0286	.0350
50 - 100	57	.038	.0589	54	.0586	.0669
100 - 150	43	.030	.0480	43	.0509	.0599
Below 150	86	.026	.0439	78	.0491	.0631
TOTAL/AVERAGE	251	.0359	.513	228	.0469	.0569

OVERALL AVERAGES

Cyanide Soluble Au oz/ton	=	.0411
Calculated Head Au oz/ton	=	.0539
Average Recovery	≐	76.25%

FIGURE 2. GILT EDGE: GOLD RECOVERY FROM AGITATED CYANIDE LEACH TESTS
COMPARISON OF PULVERIZED VERSUS UNPULVERIZED SAMPLES
ROTARY DRILLHOLE CUTTINGS

	DAKOTA MAID ZONE		SUNDAY ZONE	
	PULVERIZED	UNPULVERIZED	PULVERIZED	UNPULVERIZED
NUMBER SAMPLES TESTED	105	105	102	102
AVERAGE Au RECOVERED, oz/ton	.0349	.0308	.0539	.0441
AVERAGE FIRE ASSAY HEAD, Au oz/ton	.0472	.0472	.0622	.0622
AVERAGE CALC. HEAD, Au oz/ton	.0528	.0539	.0630	.0581
PERCENT CYANIDE SOLUBLE, Au. Based on Calc. Head Assay	66.10	57.14	85.56	75.90

OVERALL AVERAGES (Both Zones)	PULVERIZED	UNPULVERIZED
Average oz/ton Au Recovered	.0443	.0374
Average Fire Assay Head	.0546	.0546
Average Calc. Head	.0578	.0560
Percent Au Recovered, Based on Calc. Head	76.64	66.79

Gold recovery from pulverized samples averaged 10 percent higher than gold recovery from unpulverized (below 1/4-inch) portions of the same samples.

Statistical analysis has been run on this data using a "t" test, which indicates that the calculated head grade of the samples can be predicted 90 percent of the time within 5 percent of actual, using about 22 samples in the Dakota Maid Zone, and about 32 samples in the Sunday Zone. Surprisingly, there did not seem to be much statistical difference in required sample size when using coarse-crushed, as compared with pulverized material (the pulverized material was usually made by pulverizing 400 grams of coarse-crushed material).

The use of 30 samples to predict gold recoveries in a specific ore block does not seem to be too prohibitive a number. The statistical analysis does not predict anything about the number of samples required to get a valid sample of the ore block, however. It merely shows that if 30 samples are analyzed, a good prediction of the overall sample gold content is obtained. This eliminates the need to fire assay each sample.

Typically, during production, a standard ore block of 3000 tons (perhaps one days production) would be drilled on 9 foot centers for blasting. Sampling and assaying all blastholes on 5 foot vertical intervals would yield 90 samples for the block. This level of sampling is higher than normal practice, but would not be prohibitively costly. We cannot, at present, predict whether or not this level of sampling is sufficient, but we feel a system for bottle roll cyanide soluble gold analyses can be developed for use in mine grade control.

Direct Use of Unpulverized Samples. The data in Figure 2 also presents a measure of overall gold recovery in the agitated tests on unpulverized, versus pulverized, samples. The unpulverized samples yield about 10 percent lower recovery than the pulverized samples. In the Sunday Zone, recovery from the unpulverized samples was about 76 percent, (as compared with 86 percent for this set of samples from pulverized material, and 81 percent for all Sunday Zone drillholes from pulverized material, as presented in Figure 1). The data indicates that bottle roll tests on unpulverized samples will yield an overall cyanide soluble gold content for the Sunday Zone, approximately 71 percent of the fire assayable gold content. Since this is the same percent recovery expected from the heap leach, such testing procedures should make fairly good operational guides for ore control purposes.

The use of unpulverized drillhole cuttings in agitated bottle roll tests presents several operational advantages for the field production laboratory:

1. Much larger sample sizes can be used in the bottles.

2. The samples do not have to be thoroughly dried (a few percent moisture prevents pulverizing, but can be factored into test results).

3. Because of the larger sample size, and direct testing of cuttings as they arrive at the lab from the field drills, the agitated bottle roll leach tests can be set up quickly and very productively. One operator can easily run 100 determinations for cyanide soluble gold per day.

2. VARIATION IN LEACH BEHAVIOR THROUGHOUT THE OREBODY

A set of orebody cross-sections, showing percent gold recoveries in agitated tests, was included in the appendices to the 10 August, 1981 report mentioned earlier. As those maps indicate, there do not appear to be any zones defineable laterally in cross-sections, where gold recoveries are significantly higher or lower than the norm.

Figure 3 of this report breaks down gold, silver and copper recoveries by ore depth and within each ore zone. There appears to be a recognizable pattern of behavioral differences within the Dakota Maid Zone, on this basis. (Note that the data is organized with depth in the hole, rather than by fixed elevation; organizing data on the latter basis has not yet been tried.)

Dakota Maid Zone. The Dakota Maid Zone contains approximately three times as much soluble silver and copper, on an ounce per ounce basis with soluble gold, as does the Sunday Zone. Data was also generated on cyanide consumption which averaged 6.42 pounds NaCN per ton in the Dakota Maid tests, and 4.35 pounds NaCN per ton in the Sunday tests.

Gold solubility with depth decreases uniformly and significantly in the Dakota Maid Zone, while cyanide consumption steadily increases. This, and the silver and copper results, are consistent with the geologic evidence that the Dakota Maid Zone contains an oxidized, copper-depleted cap above a "base" of high-sulfide ore.

Sunday Zone. There does not appear to be any significant variation with depth, in ore gold recovery within the Sunday Zone. In underground drifts in the Sunday Zone, there is evidence of secondary chalcocite and copper salts starting 100 - 200 feet below the surface. This "enrichment zone" is reflected by increasing soluble copper and cyanide consumption in the tests. These findings are consistent with the geologic evidence, which indicates a central vertical "core", or partially brecciated mass of oxidized, low sulfur, volcanics.

The general conclusion is, that there is not any area of the deposit which can be singled out for special treatment as a "refractory" ore block. For milling purposes, and probably even more markedly for heap leach purposes, the upper portions of the Dakota Maid Zone appear to show much better cyanide leach response than the lower portions.

FIGURE 3. GILT EDGE 24 HOUR AGITATED

CYANIDE LEACH TESTS
ON PULVERIZED RDH SAMPLES

Results Summarized by Vertical Depth in Ore Zones

DAKOTA MAID ZONE

INTERVAL Feet	NO. SAMPLES TESTED	CYANIDE SOLUBLE Au oz/ton	CYANIDE SOLUBLE Ag oz/ton	CYANIDE SOLUBLE Cu,ppm	NaCN CONSUMPTION lbs NaCN/ ton of ore	CALC.HEAD Au oz/ton	Au, % RECOVER Based on CALC.HEAD Assay
0 - 50	65	.051	.213	36	5.62	.0565	90.3
50 - 100	57	.038	.154	145	6.67	.0589	64.5
100 - 150	43	.030	.149	91	6.77	.0480	62.5
Below 150	86	.026	.137	78	6.69	.0439	59.2
AVERAGE	251	.0359	.163	85	6.42	.0513	70.0

SUNDAY ZONE

INTERVAL Feet	NO. SAMPLES TESTED	CYANIDE SOLUBLE Au oz/ton	CYANIDE SOLUBLE Ag oz/ton	CYANIDE SOLUBLE Cu, ppm	NaCN CONSUMPTION lbs NaCN/ ton of ore	CALC.HEAD Au oz/ton	Au,% RECOVERY Based on CALC.HEAD Assay
0 - 50	53	.0286	.051	15	3.45	.0350	81.7
50 - 100	54	.0586	.063	32	3.75	.0669	87.6
100 - 150	43	.0509	.078	25	4.52	.0599	85.0
Below 150	78	.0491	.091	58	5.29	.0631	86.3
AVERAGE	228	.0469	.073	36	4.35	.0569	82.4

For economic evaluation purposes, it would be useful to have data on the ore tonnages and projected grades, within each of the same vertical intervals shown in Figure 1.

It is also important to note that the recovery data presented in Figure 3 shows 24 hour cyanide soluble gold, from pulverized samples in a solution containing an excess of cyanide. Those recoveries are a good indication of potential recoveries in a conventional agitated cyanide mill. Mill recoveries might be improved somewhat by extremely fine grinding, and by additional oxidation or pre-oxidation, but the recoveries projected from Figure 3 are not far from the mid-70 percent recoveries actually achieved in the Dakota Maid mill of the 1930's. Heap leach recoveries are expected to be lower, 70 percent of fire assayable gold in the Sunday Zone and perhaps 50 percent in the Dakota Maid Zone.

3. SILVER CONTENT AND RECOVERY

Average silver content as reported by fire assay for the 479 samples used for pulverized tests was 0.14 ounces Ag per ton, and cyanide soluble silver in the same samples was 0.12 ounces Ag per ton. Gold recovery from the same tests was 0.041 ounces Au per ton, so the fineness of recovered gold bullion (ratio of gold to gold plus silver, times 1000) averaged 255 from the agitated leach tests. This should be a fairly good indication of the silver to gold recovery ratio that could be expected from conventional milling.

Figure 4 presents data on 18 one-ton bulk samples taken in 1979. The fineness of cyanide soluble metal in agitated tests on pulverized head samples is shown, as well as the fineness results from bucket leach tests on crushed (2-inches and 5/8-inches) rocks.

In the bucket leach test, 0.27 ounces silver were recovered for each ounce of gold recovered. In the bottle roll tests, 0.87 ounces of silver were recovered for each ounce of gold. After applying an adjustment for the difference in gold recovery (76 percent agitated, 71 percent bucket leach), the amount of silver recovered (i.e., ounces per ton of ore) in the bottle roll tests is 3.5 times the amount of silver recovered from the heap leach.

FIGURE 4. 1979 GILT EDGE BULK SAMPLES
SILVER AND COPPER SOLUBILITY
IN AGITATED TESTS AND HEAP LEACH TESTS

BULK SAMPLE NUMBER	FINENESS HEAD FIRE ASSAY (Au/Au+Agx1000)	AGITATED CYANIDE LEACH ON PULVERIZED SAMPLES		BUCKET TEST NOS.	BUCKET LEACH TESTS ON COARSE ROCKS		
		FINENESS RECOVERED METAL	CYANIDE SOLUBLE COPPER, ppm		FINENESS RECOVERED METAL	oz Au/ton RECOVERED	CYANIDE SOLUBLE COPPER, ppm
1	105	621	66	500-501	945	.049	8
2	434	634	57	502-503	944	.015	1
3	95	784	53	504-505	976	.008	6
4	978	810	80	506-507	678	.052	12
5	145	366	56	508-509	697	.009	2
6	37	711	60	510-511	864	.004	8
7	31	421	25	512-513	799	.007	5
8	186	297	28	514-515	746	.017	7
9	183	632	28	516-517	899	.031	8
10	99	543	40	518-519	863	.011	6
11	107	575	35	520-521	899	.016	6
12	64	462	38	522-523	764	.029	11
13	84	368	48	524-525	191	.002	29
14	193	586	39	526-527	895	.031	8
16	104	207	33	530-531	608	.001	6
17	91	425	59	532-533	790	.011	3
18	41	574	36	534-535	765	.020	5
20	738	602	92	538-539	901	.076	66
AVERAGE	206	534	48		790	.022	11

As already mentioned, for the orebody as a whole, the gold fineness in 479 tests on pulverized agitated samples, was 255. Assuming that the orebody averages 0.05 ounces gold per ton, the amount of silver recoverable from the orebody will be 0.095 ounces per ton in an agitated leach system (conventional mill). Applying the ratio of 3.5:1 for the amount of silver recoverable from the heap leach as compared with the mill, the amount of silver recoverable from the heap leach will be 0.027 ounces per ton. This represents a "charge" against heap leaching of 0.068 ounces Ag per ton, which is within the normal "noise" levels of economic projections.

As shown in Figure 3, there appears to be no major areas of the orebody which contain silver exceeding the average levels sufficient to require special treatment.

4. COPPER CONTENT AND RECOVERY

Cyanide soluble copper content of the 479 rotary drillhole samples, in agitated cyanide leach tests on pulverized material, is shown in Figure 3, and averages 85 ppm in the Dakota Maid Zone and 36 ppm in the Sunday Zone. The average for the two ore zones is equivalent to a copper solubility of 43 mg of copper per mg of gold. This is the ratio that could be expected in a conventional agitated cyanide mill.

Figure 4 presents cyanide soluble copper levels for 19 samples, which were tested in agitated tests and in 12 week long bucket leach tests. The data indicates that the copper that is leached in a heap will be 23 percent of the copper that is leached in a conventional mill. For the tests, this amounted to 14 mg per mg of gold, and for the orebody as a whole, it should amount to 10 mg per mg of gold.

Each milligram of copper will require use of an additional 2 to 5 mg of NaCN (depending on copper mineralogy). Assuming an average of 3:1, a gold recovery level of 0.035 ounces per ton, and a cyanide cost of $0.70 per pound, the cyanide cost due to dissolved copper in a convnetional milling situation will be $0.23 per ton.

Some specific areas of both ore zones contain visible chalcocite (cyanide soluble copper sulfide), and oxidized copper salts, and the cyanide soluble copper in some agitated tests exceeded 1000 ppm in solution. These local areas will require special consideration in either a mill or a heap leach, but they are not expected to affect overall economics or operations design.

5. ZONES OF SECONDARY COPPER ENRICHMENT

Visual evidence on both the Dakota Maid and the Sunday Zones, in underground workings, indicates that there could be significant areas of secondary copper enrichment. The location of these zones is important, both as a guide to expected behavior during processing, and as another bit of information to use in geologic evaluation of the orebody.

Data on cyanide soluble copper is presented in Figure 3, by depth within each ore zone. Appendix C (attached to the ten "primary" copies of this report), contains vertical cross-section maps through the orebody, and presents soluble copper in the rotary drillhole intervals, color-coded as to solubility levels (the solubility data is included in Appendix A). The maps do not present a clear picture of any identifiable pattern in copper solubility, either vertically or laterally.

The data in Figure 3 indicates a strong area of secondary enrichment in the Dakota Maid Zone at a depth of 50 to 100 feet, which is apparently also the contact zone between the oxidized cap and the pyrite-rich "base". Corresponding to the copper enrichment, gold recovery decreases and cyanide consumption increases. There does not appear to be an increase in overall gold or silver content.

In the Sunday Zone, copper solubility is fairly uniform, with a significant increase only at the deepest interval tabulated (below 150 feet). This corresponds with evidence underground. Copper staining is uncommon in the Rattlesnake Tunnel level (depth 0 to 130 feet), but chalcocite and copper staining can be seen heavily on the R 3 level (150 feet below the Rattlesnake, and 60 feet above the present water table).

TEST PROCEDURES FOR AGITATED TESTS

The bottle roll test procedure used for the Gilt Edge rotary drillhole cuttings was considerably more involved than a "conventional" cold cyanide leach test used to survey leach behavior on such ores as those from Northumberland, Nevada. In the conventional test, a small portion of ore (10 grams) is leached, the solution centrifuged and assayed, and the recoveries are calculated based on a head assay of the same ore pulp. Because of the presence of coarse gold in Gilt Edge ores, classical bottle roll test procedure was employed. A large portion of the ore (100 grams) was used, and at the end of the test, the tailings were filtered, washed, dried and fire assayed. The tailings fire assay results, rather than head fire assay values, were used in calculating per cent recoveries.

The test procedure was as follows:

1. A 100 gram portion of ore was placed in a 250 ml polybottle, 150 mls of water added and the pH adjusted to 10, if necessary, with lime.

2. NaCN was then added to the solution to give the equivalent of 5 grams per liter NaCN and the sample was placed on a set of rolls and rotated slowly for 24 hours.

3. After 24 hours, the solutions were filtered and checked by AA methods, for gold, silver and copper. Cyanide was measured by titration, and pH was measured using a pH meter.

4. The tailings were dried, pulverized if necessary, and submitted for fire assay.

Submitted by,

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DWK/df

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
75-GLE-1	2 - 10	75-GLE-2201	825 A	0.056	0.48	45
-1	25 - 30	-2204	825 B	0.102	0.24	84
-1	40 - 50	-2206	826 B	0.032	0.00	96
-1	50 - 60	-2207	824 B	0.030	0.15	72
-1	60 - 70	-2208	957 A	0.071	0.32	863
-1	70 - 80	-2209	844 A	0.020	0.02	1734
-1	80 - 90	-2210				
-1	90 -100	-2211	824 C	0.033	0.10	72
-1	100 -110	-2212	826 C	0.018	0.00	99
-1	110 -120	-2213				
-1	120 -130	-2214	841 A	0.121	0.07	223
-1	130 -140	-2215	825 C	0.014	0.00	81
-1	140 -150	-2216	494 F	0.023	0.06	51
75-GLE-2	2 - 10	-2222	839 A	0.239	0.19	14
-2	10 - 20	-2223				
-2	20 - 30	-2224	815 G	0.042	0.23	84
-2	30 - 40	-2225	844 B	0.033	0.03	357
-2	50 - 60	-2227				
-2	60 - 70	-2228	838 A	0.076	0.07	715
-2	70 - 80	-2229	824 D	0.057	0.12	66
75-GLE-3	2 - 10	-2242	551 A	0.031	0.11	5
-3	10 - 20	-2243	839 B	0.047	0.08	10
-3	20 - 30	-2244	551 B	0.030	0.08	9
-3	40 - 50	-2246	494 G	0.008	0.04	8
-3	50 - 60	-2247	836 A	0.039	0.08	7
-3	60 - 70	-2248	837 A	0.011	0.06	5

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
75-GLE-3	90 - 100	75-GLE-2251	824 E	0.033	0.19	72
-3	110 - 120	-2253	827 B	0.040	0.08	114
-3	140 - 150	-2256				
-3	150 - 160	-2257				
-3	160 - 170	-2258	836 B	0.006	0.00	283
-3	170 - 180	-2259	838 B	0.028	0.24	548
-3	180 - 190	-2260	841 B	0.121	0.23	433
-3	190 - 200	-2261	836 C	0.017	0.24	586
75-GLE-4	2 - 10	-2267	838 C	0.250	0.60	14
-4	10 - 20	-2268	836 D	0.070	0.54	7
-4	20 - 30	-2269	820 B	0.047	0.51	9
-4	40 - 50	-2271	841 C	0.289	1.18	398
-4	50 - 60	-2272	838 E	0.025	1.37	300
-4	60 - 70	-2273	481 D	0.011	0.31	78
-4	70 - 80	-2274	475 E	0.014	0.12	39
-4	80 - 90	-2275	844 C	0.026	0.11	840
-4	30 - 40	-2270	838 D	0.030	1.39	17
75-GLE-5	50 - 60	-2298	839 C	0.011	0.43	15
75-GLE-6	2 - 10	-2318	815 H	0.024	0.31	15
-6	10 - 20	-2319	836 E	0.164	0.48	11
-6	20 - 30	-2320	819 I	0.077	0.31	9
-6	30 - 40	-2321	819 F	0.059	0.48	16
-6	40 - 50	-2322	551 C	0.069	1.30	24
-6	50 - 60	-2323	838 F	0.032	0.04	2507

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
75-GLE-7	120 - 130	75-GLE-2351				
-7	130 - 140	-2352				
-7	140 - 150	-2353	836 F	0.035	0.18	17
-7	150 - 160	-2354				
-7	160 - 170	-2355	839 D	0.044	0.29	86
-7	170 - 180	-2356	550 A	0.015	0.18	54
75-GLE-8	270 - 280	-2391	475 F	0.017	1.13	39
75-GLE-10	20 - 30	-2420				
-10	140 - 150	-2432	822 C	0.015	0.08	57
-10	150 - 160	-2433	837 B	0.007	0.10	79
-10	160 - 170	-2434	810 A	0.028	0.00	72
-10	170 - 180	-2435	809 A	0.018	0.14	54
75-GLE-11	60 - 70	-2443	550 B	0.020	0.11	51
-11	70 - 80	-2444	551 D	0.018	0.13	24
-11	90 - 100	-2446	552 A	0.047	0.31	60
-11	100 - 110	-2447	550 D	0.007	0.00	22
-11	110 - 120	-2448	551 E	0.019	0.16	39
-11	150 - 160	-2452	475 G	0.016	1.13	39
75-GLE-12	140 - 150	-2471	463 B	0.035	0.28	39
-12	180 - 190	-2475	481 F	0.014	0.00	75
-12	190 - 200	-2476	475 H	0.014	1.09	36

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, p
75-GLE-13	25 - 30	75-GLE-2477	825 D	0.053	0.07	11
-13	30 - 40	-2478	462 A	0.021	0.24	37
-13	40 - 50	-2479	481 G	0.004	0.06	4
-13	140 - 150	-2489	547 B	0.007	0.04	51
-13	150 - 160	-2490	815 I	0.051	0.08	72
-13	160 - 170	-2491	476 A	0.026	0.07	37
-13	170 - 180	-2492	550 E	0.019	0.04	55
-13	180 - 190	-2493	815 J	0.022	0.04	72
-13	190 - 200	-2494	481 H	0.008	0.07	75
75-GLE-14	2 - 10	-2495	844 D	0.018	0.07	4
-14	20 - 30	-2497	467 A	0.028	0.23	9
-14	30 - 40	-2498	822 D	0.073	0.11	15
-14	50 - 60	-2500	825 E	0.026	0.05	39
-14	60 - 70	-2501				
-14	190 - 200	-2514	826 D	0.008	0.03	39
75-GLE-15	60 - 70	-2520	826 E	0.018	0.05	29
-15	80 - 90	-2522	824 F	0.065	0.05	39
-15	90 - 100	-2523	791 B	0.060	0.04	31
-15	100 - 110	-2524	494 H	0.028	0.03	15
-15	110 - 120	-2525	826 F	0.017	0.03	20
-15	130 - 140	-2527	821 C	0.054	0.05	72
-15	140 - 150	-2528	837 C	0.015	0.02	45
-15	150 - 160	-2529	820 C	0.041	0.02	23
-15	160 - 170	-2530	820 D	0.037	0.02	33

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu,
75-GLE-16	140 - 150	75-GLE-2551	827 C	0.075	0.03	9
-16	160 - 170	-2553	547 C	0.008	0.00	14
-16	170 - 180	-2554	820 E	0.074	0.02	13
-16	180 - 190	-2555	547 D	0.038	0.03	23
-16	200 - 210	-2557	844 E	0.088	0.04	277
-16	210 - 220	-2558	484 G	0.051	0.06	81
-16	220 - 230	-2559	480 B	0.010	0.01	51
75-GLE-18	80 - 90	-2599	462 B	0.069	0.09	25
75-GLE-20	100 - 110	-2633	820 F	0.044	0.04	51
-20	110 - 120	-2634	820 G	0.090	0.04	20
-20	130 - 140	-2636				
75-GLE-21	5 - 10	-2643	482 A	0.031	0.05	2
-21	10 - 20	-2644	476 B	0.088	0.08	4
-21	20 - 30	-2645	482 B	0.063	0.04	2
-21	30 - 40	-2646	484 H	0.018	0.01	22
-21	40 - 50	-2647	841 D	0.197	0.02	4
-21	60 - 70	-2649	482 C	0.068	0.02	22
-21	70 - 80	-2650	820 H	0.304	0.07	10
-21	90 - 100	-2652	825 F	0.062	0.03	19
75-GLE-22	110 - 120	-2674	476 C	0.028	0.25	39
75-GLE-23	130 - 140	-2696	485 A	0.034	0.04	1
-23	140 - 150	-2697	841 E	0.157	0.04	10
-23	150 - 160	-2698	482 D	0.018	0.01	
-23	170 - 180	-2700	482 E	0.037	0.01	
-23	180 - 190	-2701	482 F	0.031	0.01	
-23	190 - 195	-2702	476 D	0.023	0.08	

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, pp
75-GLE-24	210 - 220	75-GLE-2724	844 F	0.061	0.04	27
-24	230 - 240	-2726	841 F	0.132	0.04	42
-24	240 - 250	-2727	482 G	0.007	0.01	30
-24	250 - 260	-2728	482 H	0.011	0.01	21
-24	260 - 270	-2729	462 C	0.041	0.03	30
-24	270 - 280	-2730	462 D	0.027	0.02	19
-24	280 - 290	-2731	483 A	0.039	0.01	12
-24	290 - 300	-2732	476 E	0.121	0.03	17
75-GLE-25	60 - 70	-2739	836 G	0.043	0.02	6
-25	70 - 80	-2740	839 E	0.103	0.02	7
-25	80 - 90	-2741	841 G	0.248	0.03	5
-25	100 - 110	-2743	836 H	0.008	0.01	6
-25	110 - 120	-2744	826 G	0.021	0.02	5
-25	120 - 130	-2745	483 B	0.037	0.01	2
-25	140 - 150	-2747	820 I	0.100	0.02	8
-25	160 - 170	-2749	820 J	0.035	0.03	16
-25	170 - 180	-2750				
-25	190 - 200	-2752	476 F	0.046	0.05	23
75-GLE-27	2 - 10	-2783	547 E	0.009	0.01	9
-27	10 - 20	-2784	819 A	0.022	0.01	28
-27	40 - 50	-2787	547 F	0.032	0.01	9
-27	60 - 70	-2789	825 G	0.031	0.01	30
-27	70 - 80	-2790	485 B	0.005	0.01	11
-27	80 - 90	-2791	476 G	0.127	0.02	11
-27	100 - 110	-2793	485 C	0.051	0.03	15
-27	110 - 120	-2794				

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Cyanide Soluble Ag oz/ton	Cyanide Soluble Cu, ppm
75-GLE-29	5 - 10	75-GLE-2833	551 F	0.023	0.02	5
-29	10 - 20	-2834	485 D	0.007	0.02	6
-29	20 - 30	-2835	550 F	0.007	0.01	5
-29	30 - 40	-2836	485 E	0.034	0.02	16
-29	40 - 50	-2837	550 G	0.039	0.01	6
-29	50 - 60	-2838	548 G	0.017	0.03	8
75-GLE-33	80 - 90	-2936				
-33	90 - 100	-2937	476 H	0.031	0.02	29
75-GLE-34	50 - 60	-2944	838 G	0.021	0.07	8
-34	70 - 80	-2946				
-34	80 - 90	-2947	836 I	0.007	0.08	119
-34	100 - 110	-2949	480 C	0.012	0.08	75
-34	110 - 120	-2950	841 H	0.011	0.06	16
-34	120 - 130	-2951	837 D	0.005	0.07	17
-34	130 - 140	-2952	841 I	0.009	0.14	18
75-GLE-36	20 - 30	-2981	483 C	0.003	0.03	17
-36	30 - 40	-2982	548 H	0.018	0.04	18
-36	40 - 50	-2983	485 F	0.024	0.05	126
-36	50 - 60	-2984	485 G	0.021	0.04	40
-36	60 - 70	-2985	837 E	0.010	0.03	76
-36	70 - 80	-2986	820 K	0.006	0.02	48
-36	80 - 90	-2987	485 H	0.004	0.02	89
75-GLE-38	2 - 10	-3000	838 H	0.028	0.05	17
-38	20 - 30	-3002				

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
75-GLE-39	2 - 10	75-GLE-3026				
-39	10 - 20	-3027	841 J	0.017	0.07	7
-39	20 - 30	-3028	824 H	0.026	0.05	9
-39	30 - 40	-3029	547 A	0.047	0.11	11
-39	40 - 50	-3030	844 G	0.043	0.10	18
-39	50 - 60	-3031	826 H	0.032	0.05	17
-39	60 - 70	-3032	844 H	0.021	0.07	18
75-GLE-40	10 - 20	-3053	820 L	0.034	0.08	33
-40	30 - 40	-3055	480 D	0.016	0.08	48
-40	40 - 50	-3056	480 E	0.007	0.06	66
-40	50 - 60	-3057				
75-GLE-41	140 - 150	-3087	792 A	0.016	0.03	24
-41	150 - 160	-3088	795 A	0.044	0.02	13
-41	160 - 170	-3089	836 J	0.014	0.03	13
-41	170 - 180	-3090	792 B	0.026	0.04	14
-41	180 - 190	-3091	837 F	0.025	0.03	10
-41	190 - 200	-3092	821 E	0.029	0.03	15
75-GLE-42	50 - 60	-3120				
-42	60 - 70	-3121	826 I	0.024	0.07	93
-42	70 - 80	-3122	838 I	0.022	0.06	196
-42	110 - 120	-3126				
-42	120 - 130	-3127	838 J	0.015	0.00	26
-42	130 - 140	-3128	827 D	0.027	0.04	17
-42	140 - 150	-3129	825 H	0.043	0.04	18
-42	150 - 160	-3130	825 I	0.037	0.04	51
-42	160 - 170	-3131	826 J	0.024	0.10	24

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Cyanide Soluble Ag oz/ton	Cyanide Soluble Cu, p
75-GLE-42	280 - 290	75-GLE-3143				
-42	290 - 300	-3144	551 G	0.299	0.62	18
-42	300 - 310	-3145	547 H	0.026	0.04	24
-42	310 - 320	-3146	550 H	0.022	0.02	25
-42	320 - 330	-3147	822 E	0.029	0.04	60
-42	330 - 340	-3148	547 G	0.010	0.03	52
76-GLE-44	20 - 30	76-65-329				
-44	40 - 50	-331	486 A	0.001	0.00	16
-44	50 - 60	-332	822 A	0.006	0.01	26
-44	60 - 70	-333				
-44	80 - 90	-335	486 B	0.000	0.01	17
-44	170 - 180	-344	823 F	0.005	0.29	66
-44	180 - 190	-345				
-44	250 - 260	-352	552 B	0.121	0.04	39
76-GLE-45	130 - 140	-366				
-45	140 - 150	-367	492 A	0.071	0.03	10
-45	160 - 170	-369	486 C	0.019	0.01	12
-45	180 - 190	-371	492 B	0.013	0.01	12
76-GLE-46	0 - 10	-377	552 C	0.027	0.05	6
-46	10 - 20	-378	492 C	0.022	0.03	6
-46	20 - 30	-379	492 D	0.014	0.02	11
-46	30 - 40	-380	552 D	0.016	0.01	6
-46	40 - 50	-381	486 D	0.025	0.02	8
-46	50 - 60	-382	549 A	0.033	0.03	4
-46	60 - 70	-383	815 A	0.133	0.19	12
-46	70 - 80	-384	549 B	0.076	0.05	4
-46	90 - 100	-386	492 E	0.182	0.10	8

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, pp
76-GLE-46	100 - 110	76-65-387	552 E	0.118	0.08	8
-46	110 - 120	-388	819 B	0.499	0.13	36
-46	150 - 160	-392	823 I	0.102	0.06	21
-46	170 - 180	-394	552 F	0.123	0.08	8
-46	180 - 190	-395	552 G	0.032	0.03	16
-46	190 - 200	-396	486 E	0.451	0.37	20
-46	200 - 210	-397	486 F	0.113	0.14	94
-46	210 - 220	-398	486 G	0.027	0.38	208
-46	230 - 240	-400	549 C	0.040	0.20	60
76-GLE-47	80 - 90	-409	492 F	0.027	0.07	7
-47	90 - 100	-410	492 G	0.048	0.06	4
-47	100 - 110	-411	492 H	0.028	0.09	5
-47	140 - 150	-415	486 H	0.016	0.03	18
-47	210 - 220	-422	487 A	0.005	0.05	7
-47	250 - 260	-426	747 A	0.015	0.02	8
-47	270 - 280	-428	493 A	0.127	0.04	26
-47	280 - 290	-429	487 F	0.166	0.05	46
-47	290 - 300	-430	549 D	0.094	0.10	23
-47	310 - 320	-432	493 B	0.120	0.11	23
-47	320 - 330	-433	819 H	0.102	0.12	51
-47	330 - 340	-434	487 B	0.056	0.12	525
-47	340 - 350	-435	826 A	0.073	0.20	96
76-GLE-48	40 - 50	-440	487 C	0.018	0.04	15
-48	50 - 60	-441	487 D	0.008	0.04	14
-48	60 - 70	-442	487 E	0.018	0.03	22

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
76-GLE-52	130 - 140	76-65-540				
-52	160 - 170	-543	549 E	0.028	0.19	9
-52	170 - 180	-544	827 A	0.053	0.23	9
-52	180 - 190	-545	493 C	0.083	0.31	14
-52	190 - 200	-546	493 D	0.070	0.09	23
-52	200 - 205	-547	819 B	0.070	0.13	36
76-GLE-55	180 - 190	-624	493 E	0.028	0.30	52
-55	190 - 200	-625	814 A	0.027	0.36	66
-55	200 - 210	-626	493 F	0.029	0.14	52
-55	210 - 220	-627	548 A	0.050	0.20	66
-55	220 - 230	-628	493 G	0.053	0.35	49
-55	230 - 240	-629	747 B	0.022	0.17	60
-55	240 - 250	-630	821 A	0.018	0.13	72
-55	250 - 260	-631	493 H	0.015	0.06	49
76-GLE-56	20 - 30	-634	487 G	0.023	0.25	3
-56	30 - 40	-635	815 C	0.038	0.28	7
-56	40 - 50	-636	837 G	0.054	0.37	7
-56	100 - 110	-642	494 A	0.081	0.39	51
-56	110 - 120	-643	487 H	0.009	0.11	626
-56	120 - 130	-644	494 B	0.011	0.00	52
-56	140 - 150	-646	494 C	0.024	0.15	52
-56	200 - 210	-652	815 D	0.009	0.00	84
-56	210 - 220	-653	747 C	0.063	0.21	60
-56	220 - 230	-654				
-56	230 - 240	-655	494 D	0.012	0.15	52
-56	240 - 250	-656	837 H	0.020	0.13	440

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, pp
76-GLE-57	0 - 10	76-65-657	833 E	0.023	0.14	296
-57	100 - 110	-667	839 F	0.035	0.18	898
-57	120 - 130	-669	814 B	0.049	0.34	72
-57	160 - 170	-673	549 F	0.017	0.17	57
-57	170 - 180	-674	820 A	0.027	0.19	84
-57	180 - 190	-675	747 D	0.036	0.14	60
-57	190 - 200	-676	494 E	0.041	0.16	52
76-GLE-58	0 - 10	-677	462 E	0.020	0.08	13
-58	10 - 20	-678	457 D	0.024	0.05	11
-58	20 - 30	-679	459 D	0.014	0.01	28
-58	30 - 40	-680	467 B	0.020	0.08	11
-58	40 - 50	-681	463 A	0.014	0.11	24
-58	50 - 60	-682	467 C	0.019	0.08	13
-58	60 - 70	-683	467 D	0.017	0.03	27
-58	70 - 80	-684	462 F	0.010	0.03	22
-58	80 - 90	-685	467 E	0.026	0.06	22
-58	90 - 100	-686	814 C	0.026	0.04	60
-58	100 - 110	-687	810 B	0.077	0.06	67
-58	110 - 120	-688	823 D	0.030	0.08	72
-58	120 - 130	-689	810 C	0.028	0.00	108
-58	130 - 140	-690	833 F	0.016	0.07	356
-58	140 - 150	-691	791 C	0.014	0.07	72
-58	150 - 160	-692	821 G	0.012	0.12	72
-58	160 - 170	-693	809 B	0.018	0.12	96
-58	180 - 190	-695	456 A	0.004	0.19	33
-58	190 - 200	-696	467 F	0.003	0.15	45
-58	200 - 210	-697	821 B	0.009	0.44	45
-58	210 - 220	-698	819 J	0.007	0.00	72

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, pp
76-GLE-59	0 - 10	76-65-703	456 B	0.049	0.20	19
-59	10 - 20	-704	456 C	0.031	0.09	24
-59	20 - 30	-705	456 D	0.048	0.15	14
-59	30 - 40	-706	457 E	0.017	0.11	13
-59	40 - 50	-707	456 E	0.013	0.07	17
-59	60 - 70	-709	459 E	0.019	0.02	14
-59	70 - 80	-710	814 D	0.033	0.08	60
-59	110 - 120	-714	548 B	0.058	0.02	24
-59	170 - 180	-720	839 G	0.015	0.06	45
-59	180 - 190	-721	815 E	0.015	0.06	60
-59	190 - 200	-722	747 E	0.024	0.09	36
76-GLE-60	0 - 10	-723	461 A	0.022	0.02	14
-60	10 - 20	-724	465 A	0.030	0.02	6
-60	20 - 30	-725	465 B	0.036	0.02	4
-60	30 - 40	-726	468 A	0.011	0.04	4
-60	40 - 50	-727	468 B	0.029	0.03	4
-60	50 - 60	-728	468 C	0.038	0.02	4
-60	60 - 70	-729	468 D	0.018	0.02	6
-60	70 - 80	-730	461 B	0.011	0.01	9
-60	80 - 90	-731	795 B	0.012	0.02	6
-60	90 - 100	-732	822 G	0.052	0.03	8
76-GLE-62	20 - 30	-777	823 J	0.032	0.10	57
-62	30 - 40	-778	548 C	0.126	0.08	11
-62	40 - 50	-779	549 G	0.051	0.05	8
-62	50 - 60	-780	483 E	0.370	0.13	19
-62	60 - 70	-781	747 F	0.059	0.10	8
-62	70 - 80	-782	747 G	0.031	0.10	10
-62	80 - 90	-783	475 A	0.030	0.11	9

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu,
76-GLE-62	120 - 130	76-65-787	457 F	0.138	0.09	13
-62	150 - 160	-790	795 C	0.063	0.08	18
-62	160 - 170	-791	455 A	0.085	0.06	12
-62	170 - 180	-792	792 D	0.019	0.00	27
-62	190 - 200	-794	455 B	0.017	0.02	5
-62	200 - 210	-795	455 C	0.044	0.02	7
-62	210 - 220	-796	455 D	0.023	0.02	6
-62	220 - 230	-797	459 F	0.017	0.01	18
-62	230 - 240	-798	457 G	0.025	0.01	4
-62	240 - 250	-799	455 E	0.013	0.01	6
-62	250 - 260	-800	459 G	0.004	0.01	9
-62	260 - 270	-801	455 F	0.003	0.01	17
-62	270 - 280	-802	457 H	0.006	0.02	14
76-GLE-64	70 - 80	-830	791 D	0.079	0.08	60
-64	80 - 90	-831	795 D	0.035	0.11	66
-64	90 - 100	-832	455 G	0.030	0.08	33
-64	100 - 110	-833	792 E	0.004	0.11	78
-64	110 - 120	-834	455 H	0.039	0.13	33
-64	120 - 130	-835	455 I	0.031	0.09	33
-64	130 - 140	-836	455 J	0.017	0.07	33
-64	140 - 150	-837	455 K	0.027	0.07	33
-64	150 - 160	-838	460 A	0.018	0.06	31
-64	160 - 170	-839	456 F	0.016	0.09	28
-64	170 - 180	-840	809 C	0.023	0.05	93
-64	180 - 190	-841	810 D	0.048	0.04	75
-64	190 - 200	-842	814 E	0.043	0.12	72
-64	200 - 210	-843	795 E	0.008	0.00	66
-64	210 - 220	-844	810 E	0.027	0.00	60
-64	220 - 230	-845	810 F	0.026	0.00	60

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu,
76-GLE-64	230 - 240	76-65-846	809 D	0.022	0.00	9
-64	240 - 250	-847	809 E	0.019	0.00	9
-64	250 - 260	-848	809 F	0.021	0.00	9
-64	260 - 270	-849	814 F	0.004	0.00	6
-64	270 - 280	-850	792 F	0.039	0.00	7
-64	280 - 290	-851	792 G	0.012	0.00	7
76-GLE-66	0 - 10	-878	462 G	0.037	0.07	12
-66	10 - 20	-879	479 H	0.036	0.02	6
-66	20 - 30	-880	479 E	0.006	0.01	4
-66	30 - 40	-881	479 F	0.007	0.01	3
-66	40 - 50	-882	479 G	0.020	0.01	9
-66	50 - 60	-883	824 I	0.022	0.03	11
-66	140 - 150	-892	791 E	0.034	0.21	72
-66	150 - 160	-893	821 F	0.024	0.19	84
-66	160 - 170	-894	822 H	0.025	0.28	84
-66	170 - 180	-895	839 H	0.033	0.26	172
76-GLE-67	90 - 100	-912	747 H	0.063	0.02	2
-67	100 - 110	-913	481 A	0.019	0.01	1
-67	110 - 120	-914	483 F	0.021	0.58	1
-67	140 - 150	-917	483 G	0.015	0.06	1
-67	150 - 160	-918	821 I	0.020	0.02	1
-67	160 - 170	-919	839 I	0.022	0.03	8
-67	170 - 180	-920	475 B	0.014	0.04	3
-67	180 - 190	-921	810 G	0.056	0.03	9
-67	190 - 200	-922	822 I	0.004	0.00	8

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
76-GLE-68	230 - 240	76-65-948				
-68	240 - 250	-949	481 B	0.004	0.05	45
76-GLE-69	220 - 230	-973	844 I	0.001	0.01	43
-69	230 - 240	-974				
-69	240 - 250	-975				
-69	260 - 270	-977	483 H	0.010	0.04	75
-69	270 - 280	-978	748 A	0.017	0.04	60
-69	320 - 330	-983	484 A	0.014	0.04	81
-69	330 - 340	-984	475 C	0.041	0.12	27
76-GLE-70	90 - 100	-994	791 F	0.021	0.01	22
-70	100 - 110	-995	792 H	0.015	0.00	29
-70	110 - 120	-996	823 H	0.001	0.00	39
-70	120 - 130	-997	792 I	0.037	0.08	9
76-GLE-71	40 - 50	-1014	823 C	0.018	0.05	25
-71	50 - 60	-1015	822 B	0.015	0.00	54
-71	60 - 70	-1016	814 G	0.019	0.09	12
-71	70 - 80	-1017	815 F	0.018	0.08	108
-71	80 - 90	-1018	814 H	0.104	0.11	66
-71	90 - 100	-1019	463 C	0.024	0.06	39
-71	100 - 110	-1020	463 D	0.019	0.07	28
-71	110 - 120	-1021	467 G	0.016	0.07	8
-71	120 - 130	-1022	461 C	0.010	0.04	12
-71	130 - 140	-1023	460 B	0.024	0.07	33
-71	140 - 150	-1024	461 D	0.011	0.12	36
-71	150 - 160	-1025	461 E	0.005	0.00	31
-71	160 - 170	-1026	461 F	0.006	0.00	36

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, ppm
76-GLE-71	170 - 180	76-65-1027	461 G	0.004	0.00	36
-71	180 - 190	-1028	461 H	0.005	0.00	34
-71	190 - 200	-1029	791 G	0.011	0.00	42
-71	200 - 210	-1030	795 F	0.006	0.00	45
-71	210 - 220	-1031	822 J	0.032	0.15	84
-71	220 - 230	-1032	810 H	0.032	0.00	108
-71	230 - 240	-1033				
-71	240 - 250	-1034	795 G	0.007	0.06	72
-71	250 - 260	-1035	791 H	0.005	0.00	66
-71	270 - 280	-1037	814 I	0.001	0.00	17
-71	280 - 290	-1038	809 G	0.024	0.00	96
76-GLE-72	0 - 10	-1039	748 B	0.022	0.04	1
-72	10 - 20	-1040	475 D	0.039	0.12	27
-72	20 - 30	-1041	748 C	0.015	0.05	11
-72	30 - 40	-1042	748 D	0.021	0.05	18
-72	40 - 50	-1043	834 I	0.037	0.06	22
-72	50 - 60	-1044	748 E	0.122	0.12	14
-72	60 - 70	-1045	480 A	0.078	0.09	15
-72	70 - 80	-1046	484 B	0.381	0.15	17
76-GLE-74	100 - 110	-1101	837 I	0.035	0.47	15
-74	140 - 150	-1105	834 J	0.105	0.25	133
-74	150 - 160	-1106	823 B	0.015	0.21	33
-74	170 - 180	-1108	809 H	0.077	1.04	96
-74	180 - 190	-1109	839 J	0.016	0.25	43

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, pp
76-GLE-75	50 - 60	76-65-1120	795 I	0.036	0.12	13
-75	60 - 70	-1121	809 I	0.049	0.15	45
-75	250 - 260	-1140	837 J	0.049	0.59	217
-75	260 - 270	-1141	814 J	0.023	0.00	72
76-GLE-76	0 - 10	-1144	824 J	0.018	0.05	8
-76	10 - 20	-1145	484 C	0.015	0.06	8
-76	20 - 30	-1146	748 F	0.015	0.05	5
-76	30 - 40	-1147	484 D	0.042	0.07	19
-76	40 - 50	-1148	548 D	0.055	0.14	23
-76	50 - 60	-1149	825 J	0.032	0.08	21
-76	60 - 70	-1150	548 E	0.076	0.07	23
-76	80 - 90	-1152	458 A	0.012	0.04	12
-76	90 - 100	-1153	791 I	0.012	0.06	45
-76	100 - 110	-1154	810 I	0.038	0.04	60
-76	110 - 120	-1155	791 J	0.018	0.04	16
-76	120 - 130	-1156	460 C	0.010	0.00	31
-76	130 - 140	-1157	809 J	0.103	0.10	37
-76	140 - 150	-1158	456 G	0.081	0.05	28
76-GLE-78	0 - 10	-1201	458 B	0.016	0.05	15
-78	10 - 20	-1202	460 D	0.009	0.05	27
-78	20 - 30	-1203	458 C	0.026	0.05	31
-78	30 - 40	-1204	458 D	0.028	0.11	18
-78	40 - 50	-1205	458 E	0.024	0.13	11
-78	50 - 60	-1206	458 F	0.027	0.08	30
-78	60 - 70	-1207	460 E	0.012	0.00	39
-78	70 - 80	-1208	460 F	0.003	0.00	3

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Ag oz/ton	Cu, p
76-GLE-80	50 - 60	76-65-1249	819 C	0.064	0.04	7
-80	60 - 70	-1250	792 J	0.056	0.09	22
-80	70 - 80	-1251	823 G	0.010	0.12	8
-80	80 - 90	-1252	481 C	0.006	0.20	28
-80	90 - 100	-1253	819 E	0.027	0.09	11
-80	100 - 110	-1254	748 G	0.040	0.07	6
-80	110 - 120	-1255	548 F	0.045	0.08	33
-80	120 - 130	-1256	484 E	0.018	0.41	51
-80	130 - 140	-1257	484 F	0.039	0.58	18
-80	140 - 150	-1258	549 H	0.023	0.43	58
-80	150 - 160	-1259	819 G	0.016	0.08	72
-80	160 - 170	-1260	748 H	0.025	0.44	60
76-GLE-81	90 - 100	-1284	458 H	0.014	0.83	14
-81	100 - 110	-1285	460 G	0.010	0.65	21
-81	110 - 120	-1286	456 H	0.014	0.60	34
-81	120 - 130	-1287	456 I	0.007	0.01	34
-81	130 - 140	-1288	456 J	0.008	0.22	34
-81	150 - 160	-1290	459 A	0.014	0.22	33
-81	160 - 170	-1291	459 B	0.013	0.10	34
-81	170 - 180	-1292	459 C	0.021	0.00	34
76-GLE-87	0 - 10	77-GLE-123	795 J	0.009	0.08	17
-87	10 - 20	-124	810 J	0.215	0.20	23
-87	20 - 30	-125	457 A	0.073	0.09	10
-87	30 - 40	-126	457 B	0.013	0.09	10
-87	40 - 50	-127	457 C	0.017	0.20	14
-87	50 - 60	-128	458 G	0.070	0.17	6
-87	60 - 70	-129	479 A	0.041	0.18	69
-87	70 - 80	-130	479 B	0.109	0.18	72
-87	80 - 90	-131	479 C	0.053	0.32	72
-87	90 - 100	-132	479 D	0.066	0.76	72

24-Hr Bottle Roll Tests on Pulverized Drill Hole Samples

RDH No.	Interval (feet)	Cyprus No.	Miller-Kappes Test No.	Cyanide Soluble Au oz/ton	Cyanide Soluble Ag oz/ton	Cyanide Soluble Cu, pp
78-GLE-105	0 - 10	78-GLE-68	463 E	0.028	0.15	13
-105	10 - 20	-69	477 A	0.012	0.11	39
-105	20 - 30	-70	477 B	0.070	0.09	8
-105	30 - 40	-71	477 C	0.045	0.12	11
-105	40 - 50	-72	477 D	0.044	0.17	11
-105	50 - 60	-73	477 E	0.034	0.19	10
-105	60 - 70	-74	477 F	0.021	0.11	8
-105	70 - 80	-75	477 G	0.018	0.07	13
-105	80 - 90	-76	477 H	0.021	0.07	9
-105	90 - 100	-77	478 A	0.040	0.16	8
78-GLE-106	0 - 10	-88	478 B	0.035	0.16	8
-106	10 - 20	-89	478 C	0.018	0.12	67
-106	20 - 30	-90	478 D	0.038	0.37	54
-106	30 - 40	-91	478 E	0.089	0.31	70
-106	40 - 50	-92	463 F	0.038	0.12	14
-106	50 - 60	-93	478 F	0.018	0.31	75
-106	60 - 70	-94	463 G	0.014	0.06	9
-106	70 - 80	-95	478 G	0.013	0.17	75
-106	80 - 90	-96	478 H	0.013	0.23	75
-106	90 - 100	-97	469 E	0.053	0.06	4

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
75-GLE-1	2- 10	0.090	0.056	0.065	0.051	0.079
-1	25- 30	0.135	0.102	0.133	0.076	0.133
-1	40- 50	0.095	0.032	0.099	0.056	0.100
-1	50- 60	0.080	0.030	0.072	0.037	0.073
-1	60- 70	0.100	0.071	0.083	0.049	0.073
-1	70- 80	0.040	0.020	0.044	0.020	0.034
-1	90-100	0.040	0.033	0.048	0.025	0.049
-1	100-110	0.050	0.018	0.043	0.026	0.046
-1	120-130	0.090	0.121	0.195	0.009	0.086
-1	130-140	0.085	0.014	0.090	0.021	0.085
-1	140-150	0.040	0.023	0.033	0.028	0.038
75-GLE-2	2- 10	0.295	0.239	0.263	0.239	0.239
-2	20- 30	0.095	0.042	0.114	0.092	0.116
-2	70- 80	0.050	0.057	0.067	0.036	0.064
75-GLE-3	10- 20	0.040	0.047	0.050	0.051	0.054
-3	50- 60	0.035	0.039	0.042	0.048	0.056
-3	60- 70	0.015	0.011	0.011	0.012	0.108
-3	90-100	0.025	0.033	0.042	0.017	0.035
-3	110-120	0.035	0.040	0.050	0.028	0.052
-3	160-170	0.050	0.006	0.050	0.022	0.059
-3	170-180	0.050	0.028	0.052	0.022	0.056
75-GLE-4	10- 20	0.050	0.070	0.080	0.039	0.049
-4	20- 20	0.065	0.047	0.052	0.043	0.058
-4	50- 60	0.030	0.019	0.039	0.008	0.025
-4	70- 80	0.020	0.014	0.026	0.013	0.022
-4	80- 90	0.040	0.026	0.040	0.015	0.043

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
75-GLE-5	50- 60	0.010	0.011	0.014	0.004	0.007
75-GLE-6	2- 10	0.035	0.024	0.029	0.021	0.021
-6	10- 20	0.090	0.164	0.174	0.143	0.155
-6	20- 30	0.075	0.077	0.082	0.075	0.091
-6	30- 40	0.075	0.059	0.062	0.054	0.054
-6	50- 60	0.070	0.032	0.090	0.028	0.028
75-GLE-7	140-150	0.050	0.035	0.045	0.026	0.038
-7	160-170	0.044	0.044	0.056	0.036	0.048
75-GLE-8	270-280	0.050	0.017	0.051	0.046	0.065
75-GLE-10	140-150	0.016	0.015	0.025	0.008	0.022
-10	150-160	0.028	0.007	0.031	0.010	0.038
75-GLE-11	150-160	0.038	0.016	0.031	0.023	0.038
75-GLE-12	190-200	0.058	0.014	0.034	0.018	0.026
75-GLE-13	25- 30	0.038	0.053	0.060	0.030	0.040
-13	150-160	0.103	0.051	0.062	0.017	0.047
-13	160-170	0.046	0.026	0.048	0.026	0.054
-13	180-190	0.028	0.022	0.033	0.011	0.035
75-GLE-14	2- 10	0.024	0.018	0.021	0.011	0.295
-14	20- 30	0.034	0.028	0.037	0.033	0.038
-14	30- 40	0.065	0.073	0.078	0.052	0.062
-14	50- 60	0.020	0.026	0.032	0.019	0.029
-14	190-200	0.012	0.008	0.013	0.005	0.014

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
75-GLE-15	60- 70	0.012	0.018	0.021	0.011	0.014
-15	80- 90	0.062	0.065	0.068	0.057	0.070
-15	90-100	0.058	0.060	0.063	0.048	0.066
-15	100-110	0.038	0.028	0.031	0.034	0.046
-15	110-120	0.020	0.017	0.020	0.019	0.029
-15	130-140	0.038	0.054	0.057	0.051	0.060
-15	140-150	0.026	0.015	0.018	0.017	0.020
-15	150-160	0.032	0.041	0.044	0.035	0.051
-15	160-170	0.038	0.037	0.042	0.028	0.038
75-GLE-16	140-150	0.054	0.075	0.085	0.040	0.052
-16	170-180	0.076	0.074	0.077	0.067	0.070
-16	200-210	0.102	0.088	0.091	0.072	0.102
-16	220-230	0.022	0.010	0.021	0.021	0.026
75-GLE-20	100-110	0.042	0.044	0.053	0.029	0.047
-20	110-220	0.102	0.090	0.116	0.082	0.110
75-GLE-21	10- 20	0.092	0.088	0.101	0.103	0.126
-21	40- 50	0.450	0.198	0.212	0.042	0.080
-21	70- 80	0.482	0.305	0.327	0.124	0.214
-21	90-100	0.040	0.062	0.067	0.031	0.034
75-GLE-22	110-120	0.016	0.028	0.093	0.049	0.052
75-GLE-23	140-150	0.062	0.157	0.160	0.038	0.060
-23	190-195	0.114	0.023	0.088	0.068	0.112
75-GLE-24	210-220	0.066	0.061	0.061	0.053	0.056
-24	230-240	0.024	0.132	0.135	0.014	0.017
-24	290-300	0.165	0.122	0.135	0.138	0.149

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
75-GLE-25	60-70	0.042	0.043	0.046	0.043	0.055
-25	100-110	0.012	0.008	0.011	0.012	0.017
-25	140-150	0.090	0.100	0.103	0.095	0.098
-25	190-200	0.060	0.046	0.053	0.050	0.055
75-GLE-27	10- 20	0.022	0.022	0.025	0.020	0.023
-27	60- 70	0.014	0.031	0.036	0.011	0.011
-27	80- 90	0.214	0.126	0.169	0.161	0.210
75-GLE-33	90-100	0.072	0.031	0.063	0.030	0.045
75-GLE-34	50- 60	0.024	0.021	0.024	0.019	0.033
-34	80- 90	0.016	0.007	0.010	0.004	0.009
-34	100-110	0.022	0.012	0.018	0.014	0.021
-34	110-120	0.020	0.011	0.014	0.008	0.020
-34	130-140	0.020	0.009	0.020	0.011	0.014
75-GLE-36	60- 70	0.026	0.010	0.013	0.008	0.011
-36	70- 80	0.010	0.006	0.009	0.002	0.012
75-GLE-39	10- 20	0.018	0.017	0.051	0.018	0.021
-39	20- 30	0.044	0.026	0.029	0.022	0.028
-39	30- 40	0.054	0.047	0.057	0.053	0.067
-39	40- 50	0.042	0.043	0.046	0.034	0.044
-39	50- 60	0.030	0.032	0.037	0.021	0.035
-39	60- 70	0.026	0.021	0.024	0.015	0.015
75-GLE-40	10- 20	0.030	0.034	0.041	0.034	0.044
-40	30- 40	0.030	0.016	0.031	0.028	0.033
-40	40- 50	0.020	0.007	0.019	0.012	0.020

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
75-GLE-41	140-150	0.016	0.016	0.016	0.010	0.012
-41	150-160	0.044	0.044	0.047	0.045	0.055
-41	160-170	0.014	0.014	0.017	0.017	0.020
-41	170-180	0.022	0.026	0.029	0.018	0.022
-41	180-190	0.022	0.025	0.028	0.022	0.022
-41	190-200	0.034	0.029	0.034	0.024	0.027
75-GLE-42	60- 70	0.034	0.024	0.031	0.021	0.035
-42	120-130	0.044	0.019	0.043	0.037	0.045
-42	130-140	0.022	0.027	0.027	0.018	0.026
-42	140-150	0.042	0.043	0.049	0.040	0.051
-42	150-160	0.030	0.037	0.040	0.021	0.026
-42	160-170	0.030	0.024	0.027	0.020	0.032
-42	320-330	0.034	0.029	0.032	0.025	0.030
76-GLE-44	50- 60	None	0.006	0.006	0.002	0.005
-44	170-180	Trace	0.005	0.008	0.004	0.004
76-GLE-46	60- 70	0.080	0.133	0.148	0.080	0.150
-46	110-120	0.460	0.499	0.504	0.389	0.415
-46	150-160	0.440	0.102	0.105	0.133	0.147
-46	190-200	0.375	0.451	0.473	0.433	0.513
76-GLE-47	270-280	0.200	0.127	0.131	0.196	0.198
-47	280-290	0.090	0.166	0.192	0.070	0.090
-47	320-330	0.105	0.102	0.112	0.080	0.096
-47	340-350	0.070	0.073	0.084	0.047	0.082
76-GLE-48	40- 50	0.025	0.018	0.028	0.021	0.024

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
76-GLE-52	170-180	0.046	0.053	0.059	0.044	0.058
76-GLE-55	180-190	0.025	0.028	0.031	0.013	0.031
-55	190-200	0.040	0.027	0.044	0.018	0.036
-55	240-250	0.015	0.018	0.025	0.009	0.021
76-GLE-56	30- 40	0.025	0.038	0.043	0.039	0.049
-56	40- 50	0.050	0.054	0.064	0.049	0.059
-56	140-150	0.040	0.024	0.040	0.021	0.041
-56	200-210	0.080	0.009	0.109	0.046	0.100
76-GLE-57	100-110	0.062	0.035	0.139	0.021	0.039
-57	120-130	0.080	0.049	0.075	0.034	0.066
-57	170-180	0.040	0.027	0.043	0.021	0.047
-57	190-200	0.120	0.041	0.075	0.049	0.097
76-GLE-58	30- 40	0.030	0.020	0.029	0.020	0.031
-58	50- 60	0.030	0.019	0.037	0.015	0.035
-58	60- 70	0.025	0.017	0.035	0.012	0.028
-58	80- 90	0.040	0.026	0.057	0.022	0.056
-58	90-100	0.040	0.026	0.037	0.010	0.034
-58	110-120	0.060	0.030	0.042	0.019	0.031
-58	120-130	0.050	0.028	0.065	0.016	0.042
-58	140-150	0.035	0.014	0.017	0.007	0.021
-58	150-160	0.030	0.012	0.019	0.008	0.026
-58	190-200	0.015	0.003	0.009	0.008	0.015
-58	200-210	0.020	0.009	0.015	0.005	0.019
-58	210-220	0.015	0.007	0.016	0.007	0.014

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
76-GLE-59	70- 80	0.035	0.033	0.043	0.026	0.029
-59	170-180	0.020	0.015	0.018	0.015	0.027
-59	180-190	0.020	0.015	0.020	0.018	0.021
76-GLE-60	10- 20	0.030	0.030	0.035	0.030	0.037
-60	20- 30	0.030	0.036	0.039	0.015	0.020
-60	30- 40	0.020	0.011	0.016	0.010	0.015
-60	40- 50	0.040	0.029	0.035	0.018	0.025
-60	50- 60	0.030	0.038	0.043	0.031	0.036
-60	60- 70	0.030	0.018	0.021	0.024	0.027
-60	80- 90	0.010	0.012	0.012	0.014	0.014
-60	90-100	0.055	0.052	0.057	0.030	0.040
76-GLE-62	20- 30	0.020	0.032	0.038	0.033	0.036
-62	80- 90	0.040	0.030	0.033	0.043	0.046
-62	170-180	0.050	0.019	0.047	0.034	0.046
76-GLE-64	70- 80	0.090	0.079	0.082	0.042	0.066
-64	170-180	0.020	0.023	0.026	0.007	0.017
-64	190-200	0.040	0.043	0.059	0.023	0.053
-64	200-210	0.030	0.008	0.036	0.017	0.037
-64	240-250	0.035	0.019	0.075	0.029	0.043
-64	260-270	0.075	0.004	0.067	0.032	0.060
76-GLE-66	50- 60	0.005	0.022	0.025	0.019	0.031
-66	140-150	0.020	0.034	0.037	0.028	0.031
-66	150-160	0.025	0.024	0.030	0.012	0.054
-66	160-170	0.025	0.025	0.036	0.020	0.036
-66	170-180	0.040	0.033	0.045	0.027	0.051

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
76-GLE-67	160-170	0.020	0.022	0.032	0.012	0.015
-67	170-180	0.020	0.014	0.028	0.019	0.035
-67	180-190	0.030	0.056	0.066	0.033	0.045
-67	190-200	0.045	0.004	0.027	0.035	0.045
76-GLE-69	220-230	Trace	0.001	0.001	0.003	0.006
-69	330-340	0.020	0.041	0.053	0.014	0.030
76-GLE-70	100-110	0.060	0.015	0.051	0.039	0.051
-70	110-120	0.020	0.001	0.014	0.019	0.041
-70	120-130	0.050	0.037	0.040	0.047	0.053
76-GLE-71	40- 50	0.020	0.018	0.023	0.010	0.020
-71	50- 60	0.080	0.015	0.072	0.056	0.164
-71	60- 70	0.025	0.019	0.025	0.021	0.024
-71	70- 80	0.025	0.018	0.021	0.018	0.021
-71	110-120	0.040	0.016	0.021	0.021	0.027
-71	210-220	0.030	0.032	0.041	0.019	0.057
-71	220-230	0.030	0.032	0.053	0.010	0.028
-71	250-260	0.010	0.005	0.008	0.003	0.005
-71	270-280	0.010	0.001	0.007	0.007	0.017
-71	280-290	0.010	0.024	0.034	0.001	0.003
76-GLE-72	10- 20	0.077	0.039	0.051	0.050	0.067
-72	60- 70	0.090	0.078	0.089	0.084	0.087
76-GLE-74	100-110	0.005	0.035	0.038	0.028	0.038
-74	150-160	0.045	0.015	0.015	0.009	0.012
-74	170-180	0.050	0.077	0.080	0.031	0.048
-74	180-190	0.020	0.016	0.019	0.012	0.015

24-Hr. Bottle Roll Tests
on Drill Hole Samples

Comparison of Pulverized versus Unpulverized

RDH No.	Interval (feet)	Assay Head oz. Au/ton	Pulverized Tests oz/ton Au Recovered	Pulverized Tests Calculated Head Assay oz/ton	Unpulverized Tests oz/ton Au Recovered	Unpulverized Tests Calculated Head Assay oz/ton
76-GLE-75	250-260	0.065	0.049	0.063	0.045	0.069
76-GLE-76	0- 10	0.015	0.018	0.021	0.014	0.026
-76	50- 60	0.025	0.032	0.035	0.026	0.031
-76	110-120	0.045	0.018	0.021	0.019	0.023
76-GLE-80	90- 100	0.022	0.027	0.030	0.021	0.024
-80	150-160	0.038	0.016	0.045	0.038	0.066
77-GLE-87	0- 10	0.015	0.009	0.009	0.009	0.012
-87	10- 20	0.140	0.215	0.225	0.183	0.471
78-GLE-105	0- 10	0.020	0.028	0.037	0.014	0.020
-105	40- 50	0.020	0.044	0.057	0.038	0.061
78-GLE-106	60- 70	0.020	0.014	0.021	0.015	0.023
-106	90-100	0.020	0.053	0.060	0.055	0.065

APPENDIX C TO GILT EDGE REPORT 1982 B

CYANIDE SOLUBLE COPPER IN BOTTLE ROLL TESTS ON PULVERIZED RDH CUTTINGS

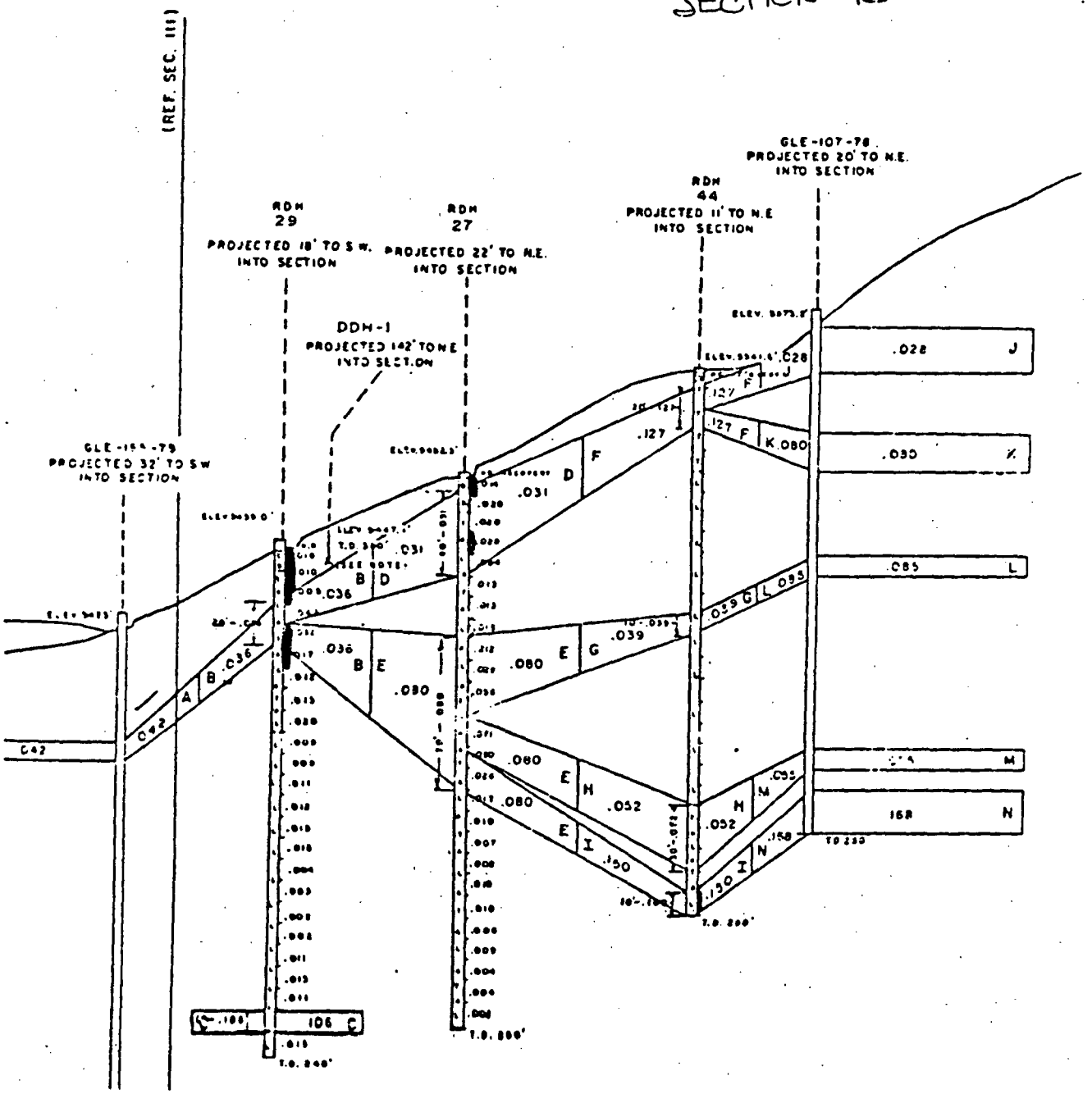
The following orebody cross-sections present the cyanide soluble copper reported in Appendix A and summarized in Figure 3 of the main body of this report.

The color key used in plotting these values is shown below:

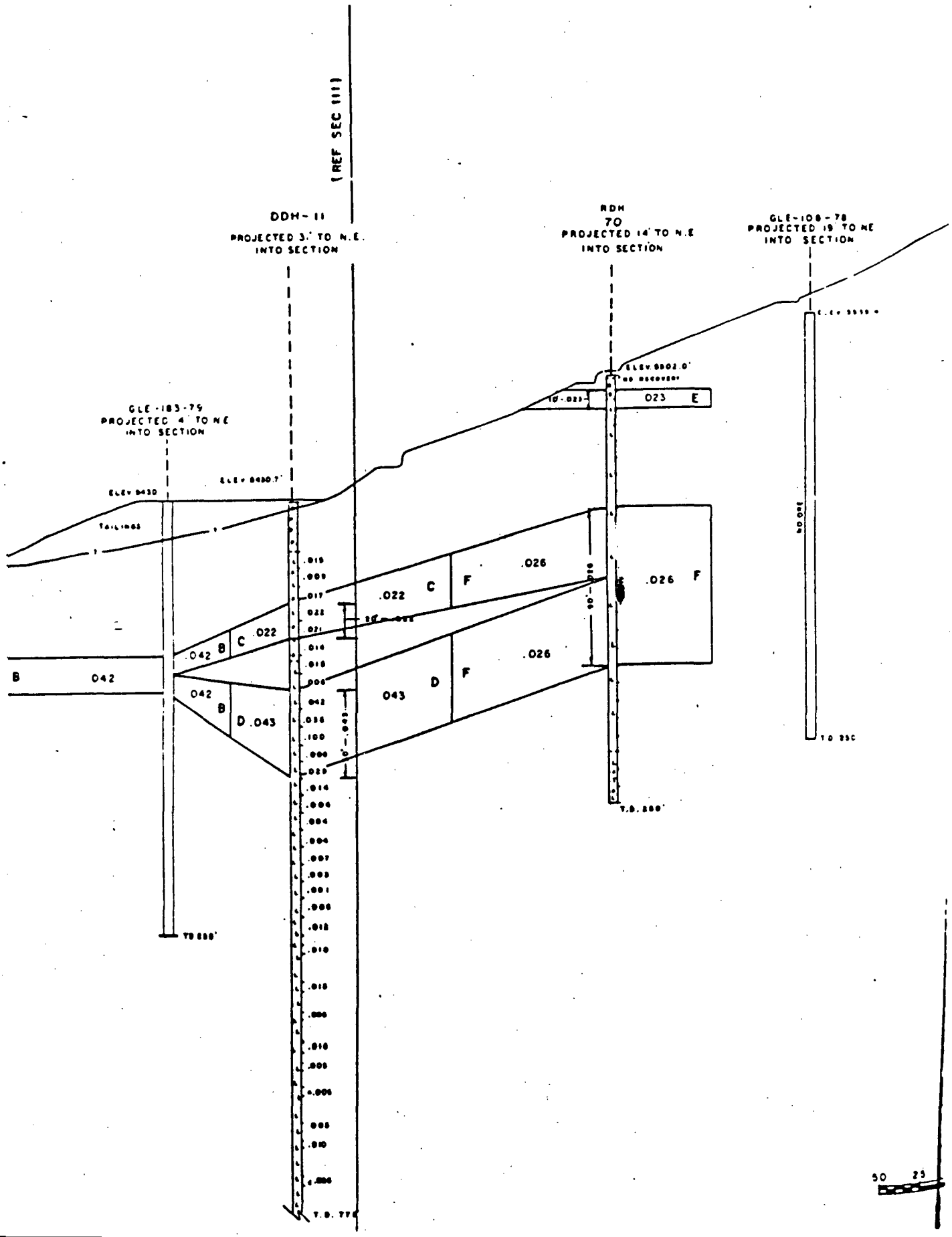
COLOR	CYANIDE SOLUBLE COPPER, ppm
Blue	0 - 10
Yellow	10 - 100
Orange	100 - 1000
Red	> 1000

SUNDAY

SECTION 15



SUNDAY



(REF SEC 111)
DDH-11
PROJECTED 3.' TO N.E.
INTO SECTION
RDH
70
PROJECTED 14' TO N.E
INTO SECTION
GLE-108-78
PROJECTED 19' TO NE
INTO SECTION
GLE-183-79
PROJECTED 4' TO NE
INTO SECTION
TAILINGS
.023
E
.026
F
.022
C
.042
B
.043
D
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.026
.042
B
.042
B
D .043
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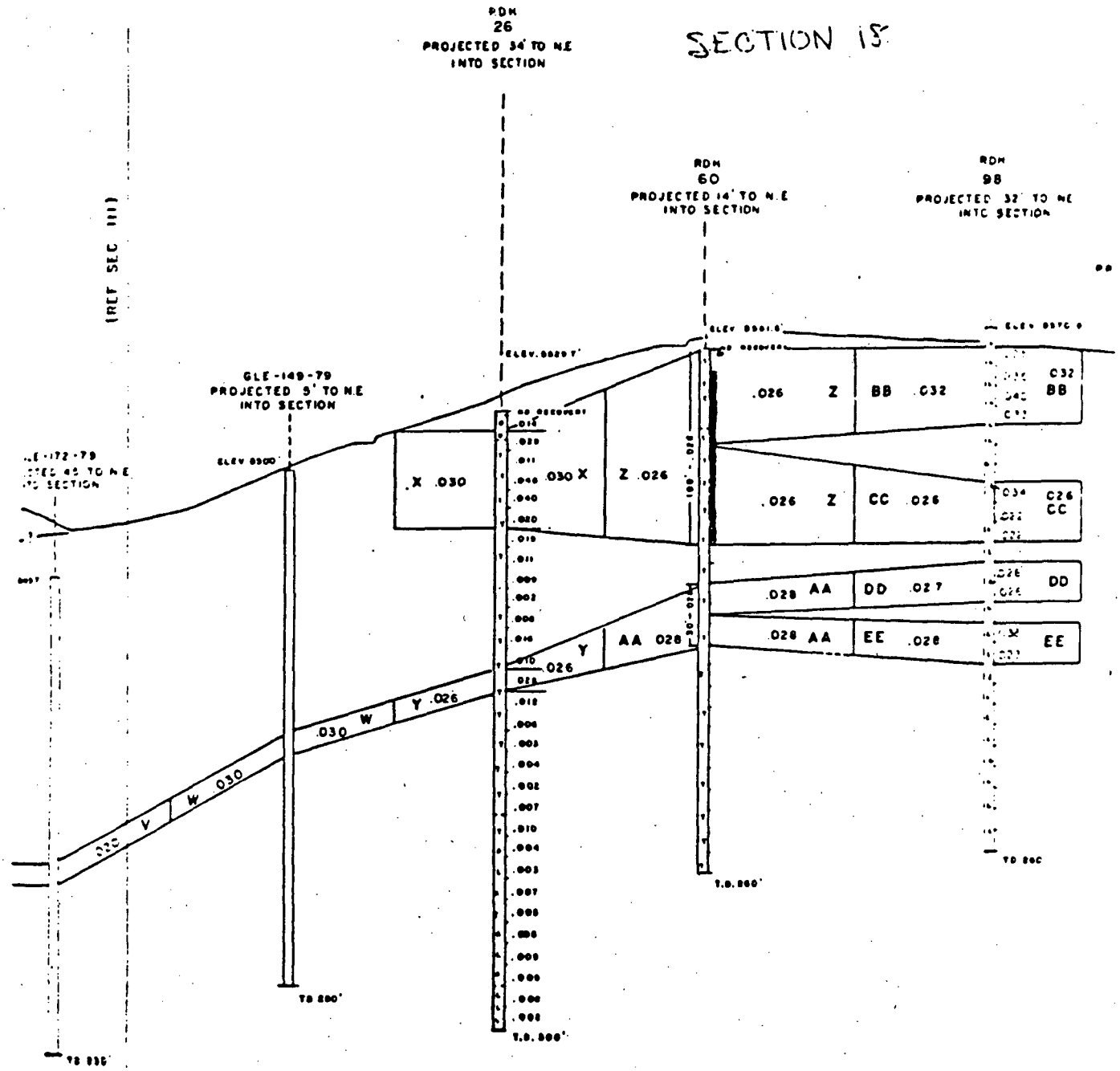
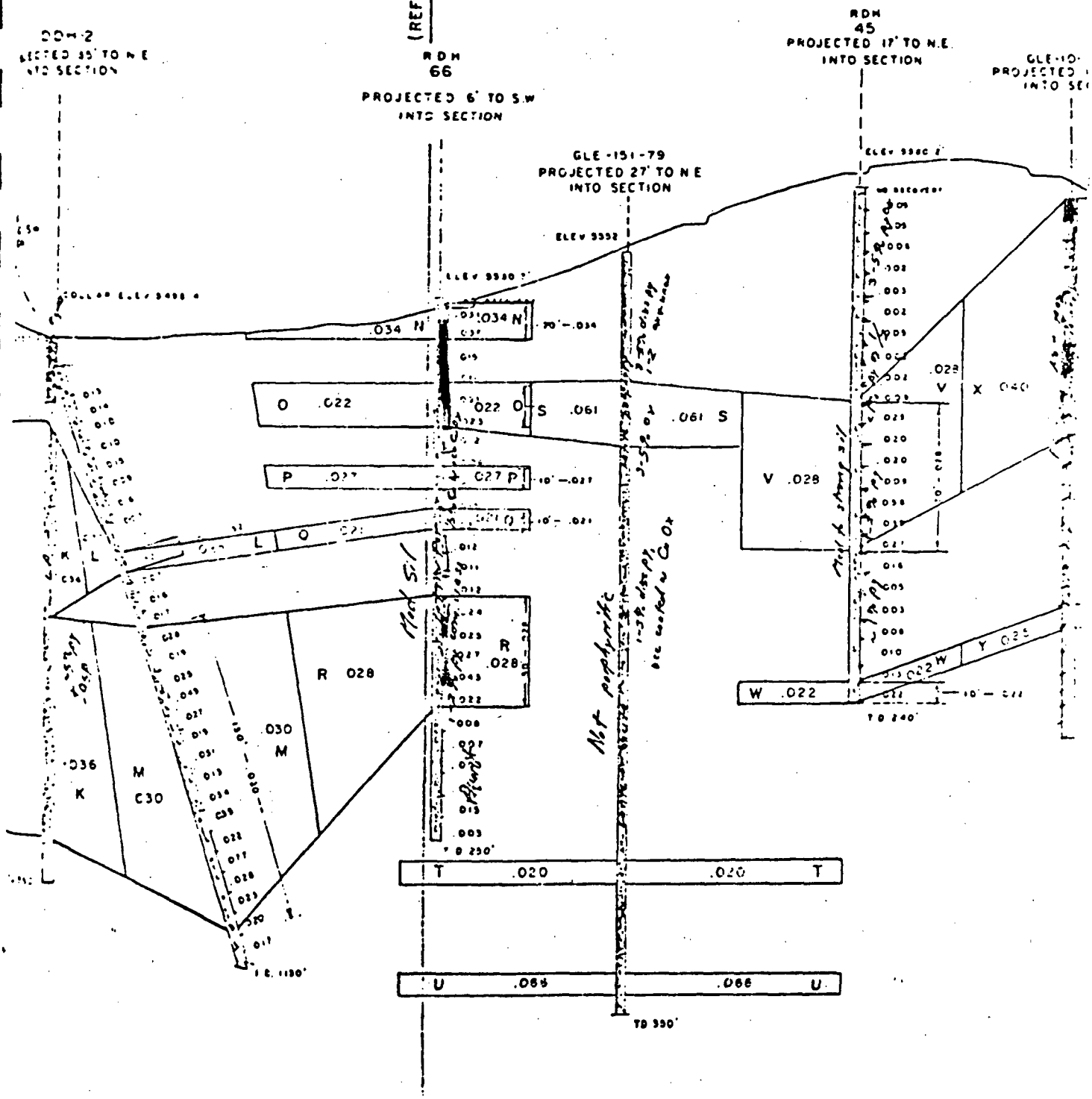
D
F
50
25

SUNDAY

SECTION 15



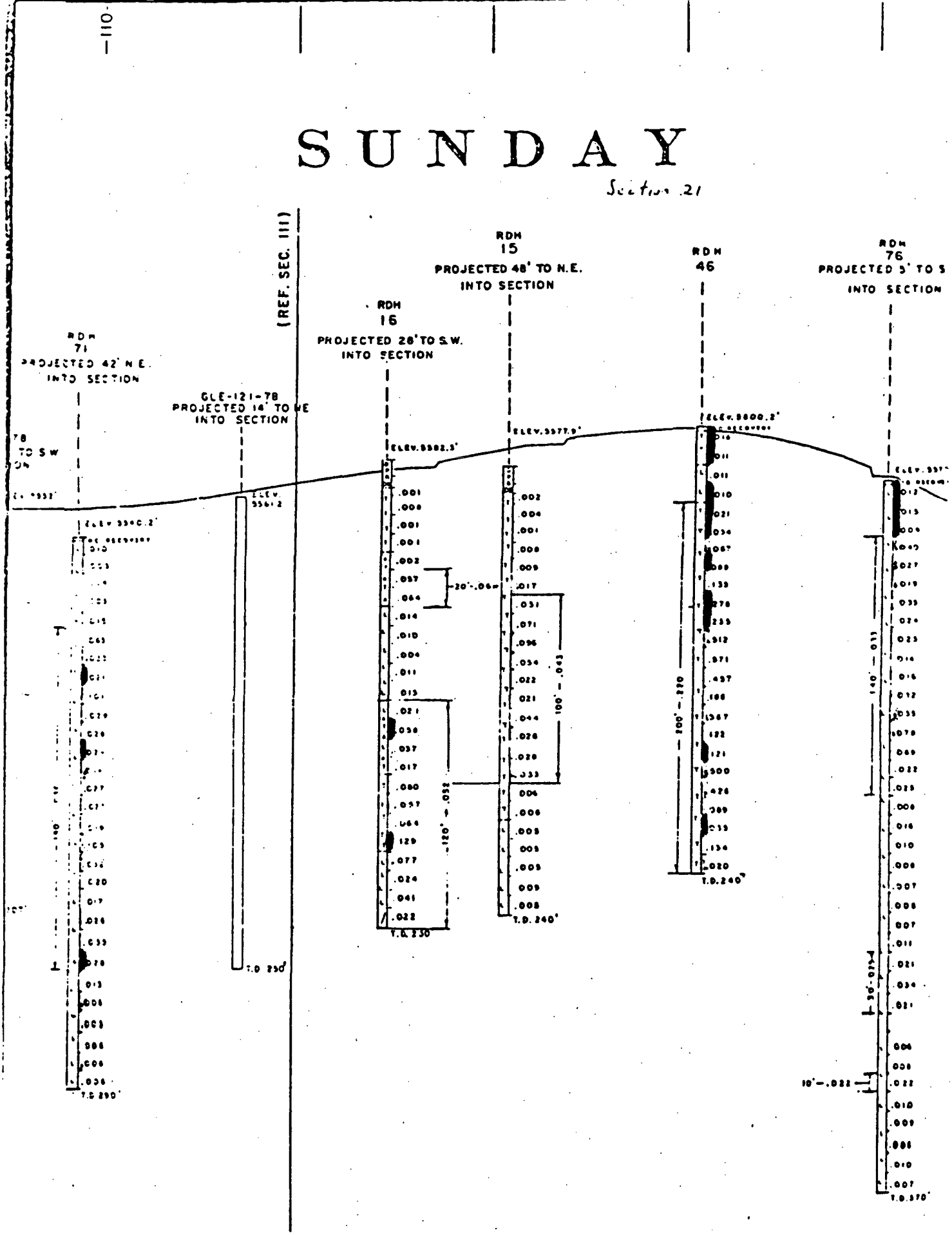
SUNDAY



(REF SEC III)
RDH 66
PROJECTED 6' TO S.W INTO SECTION
RDH 45
PROJECTED 17' TO N.E. INTO SECTION
GLE-151-79
PROJECTED 27' TO N E INTO SECTION
ELEV 5552
.034 N
O .022
S .061
V .028
X .040
P .027
R .028
M .030
K .036
W .022
Y .025
T .020
U .066
Mod Sil
Not porphyritic
TD 350'
TD 240'

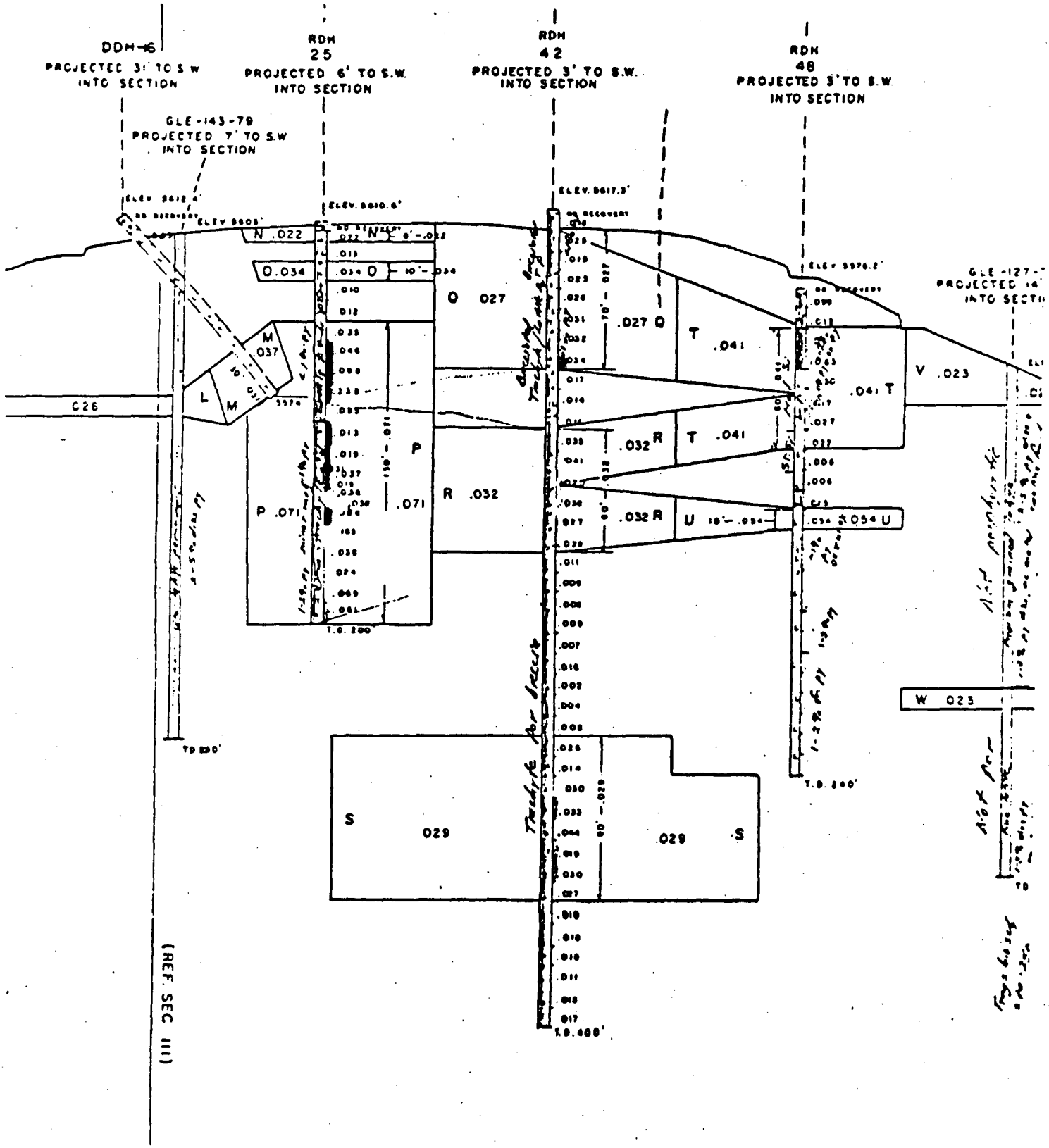
SUNDAY

Section 21



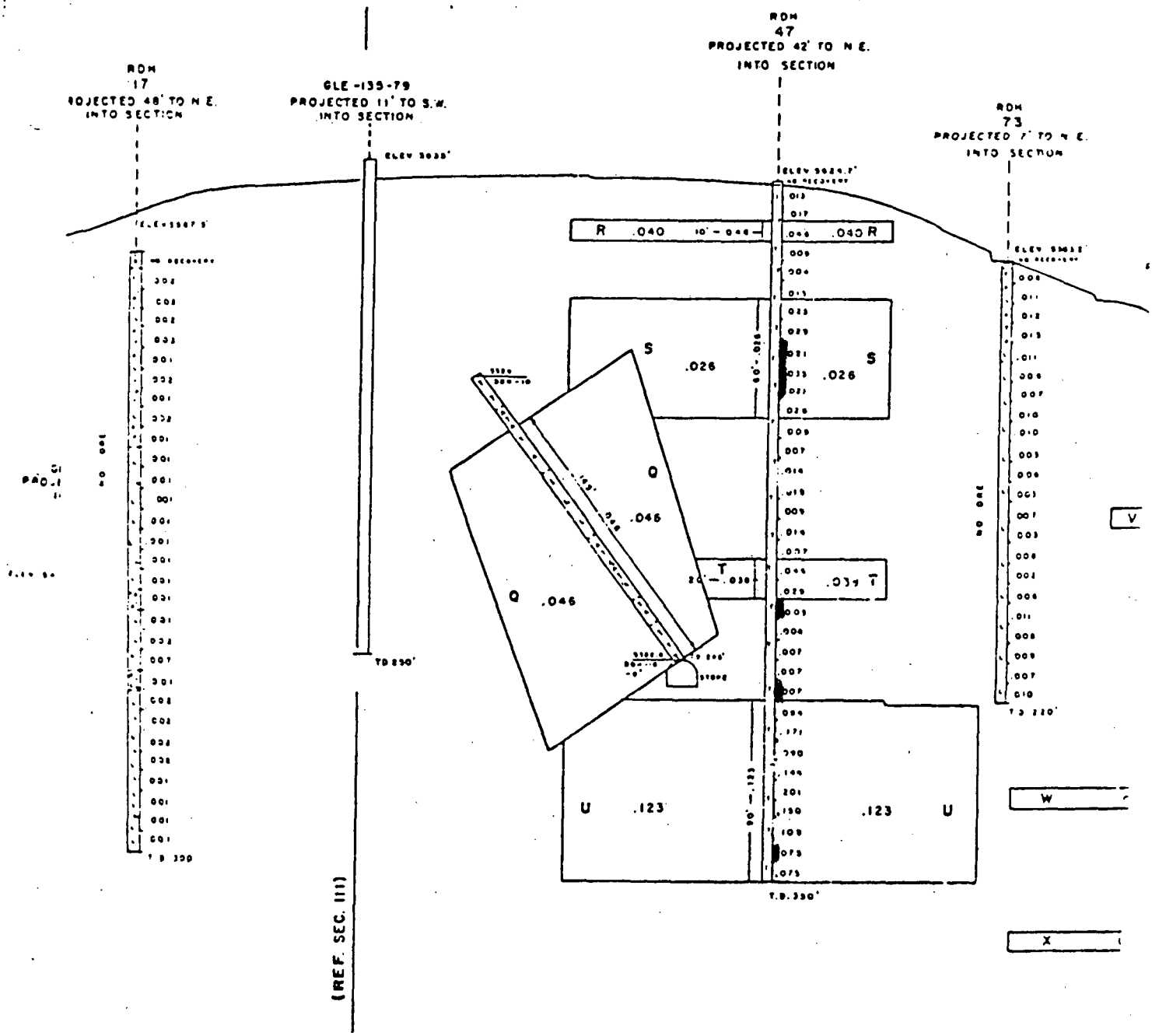
SUNDAY

SECTION 22



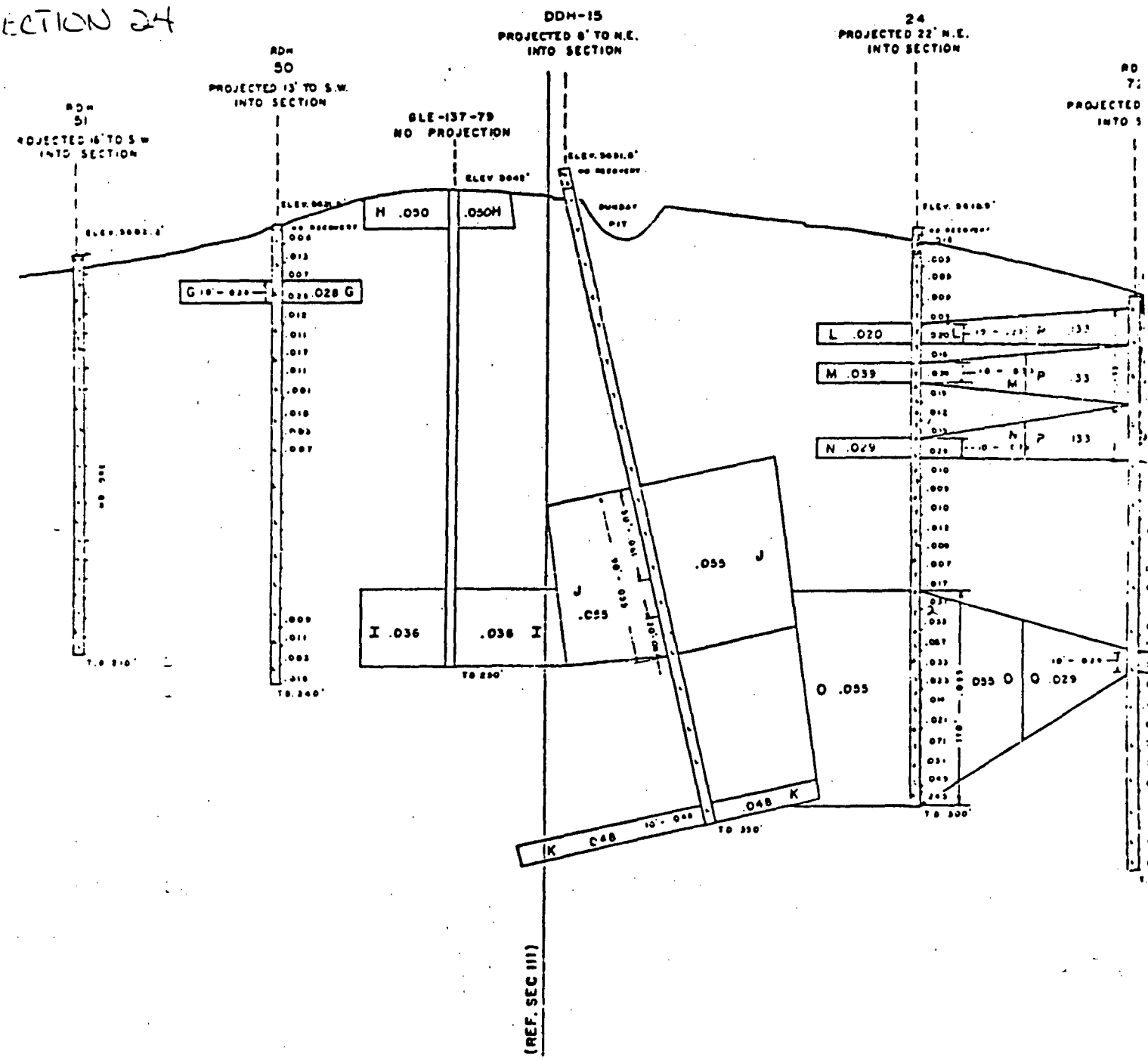
SUNDAY

SECTION D-D



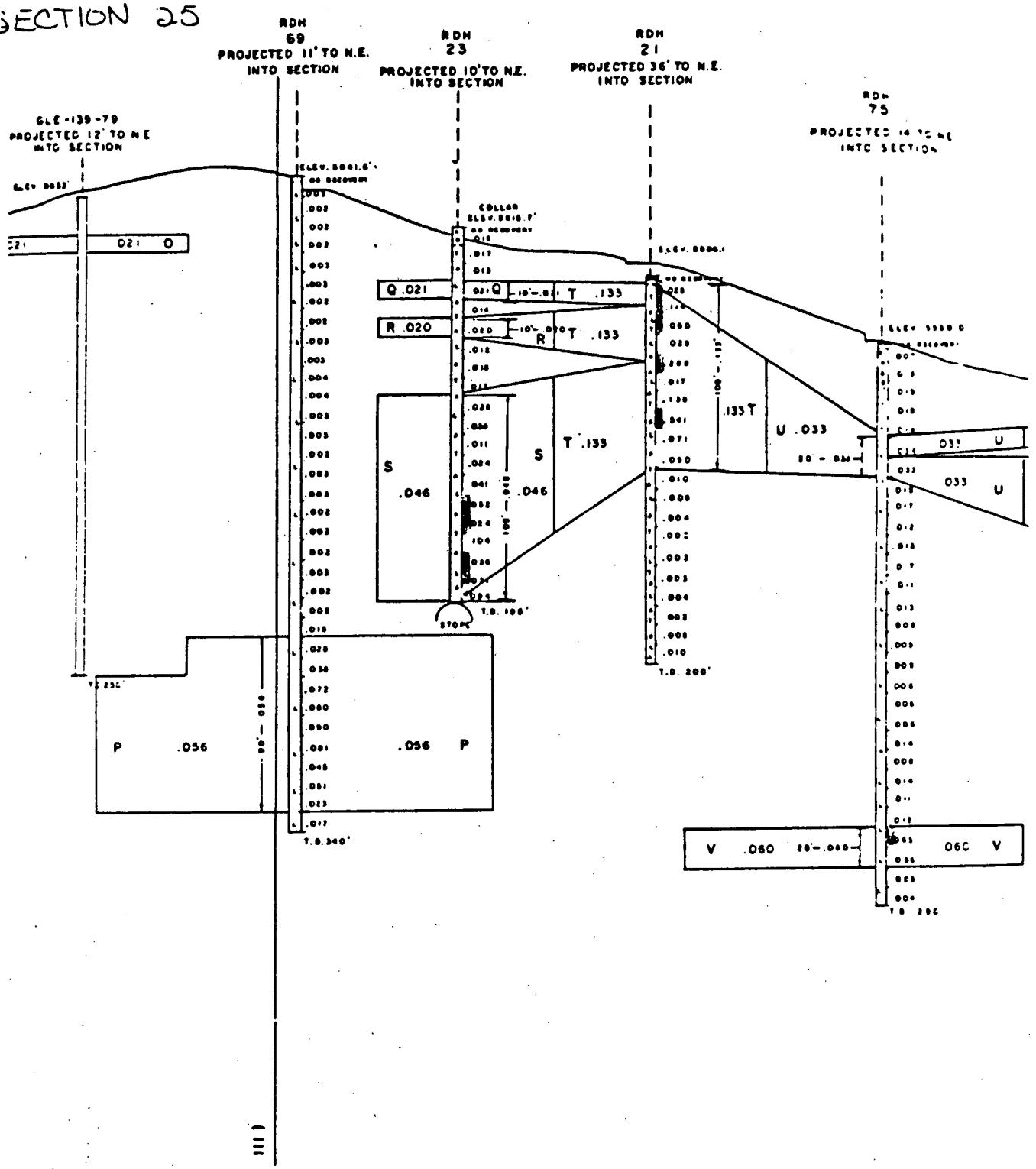
SUNDAY

SECTION 24

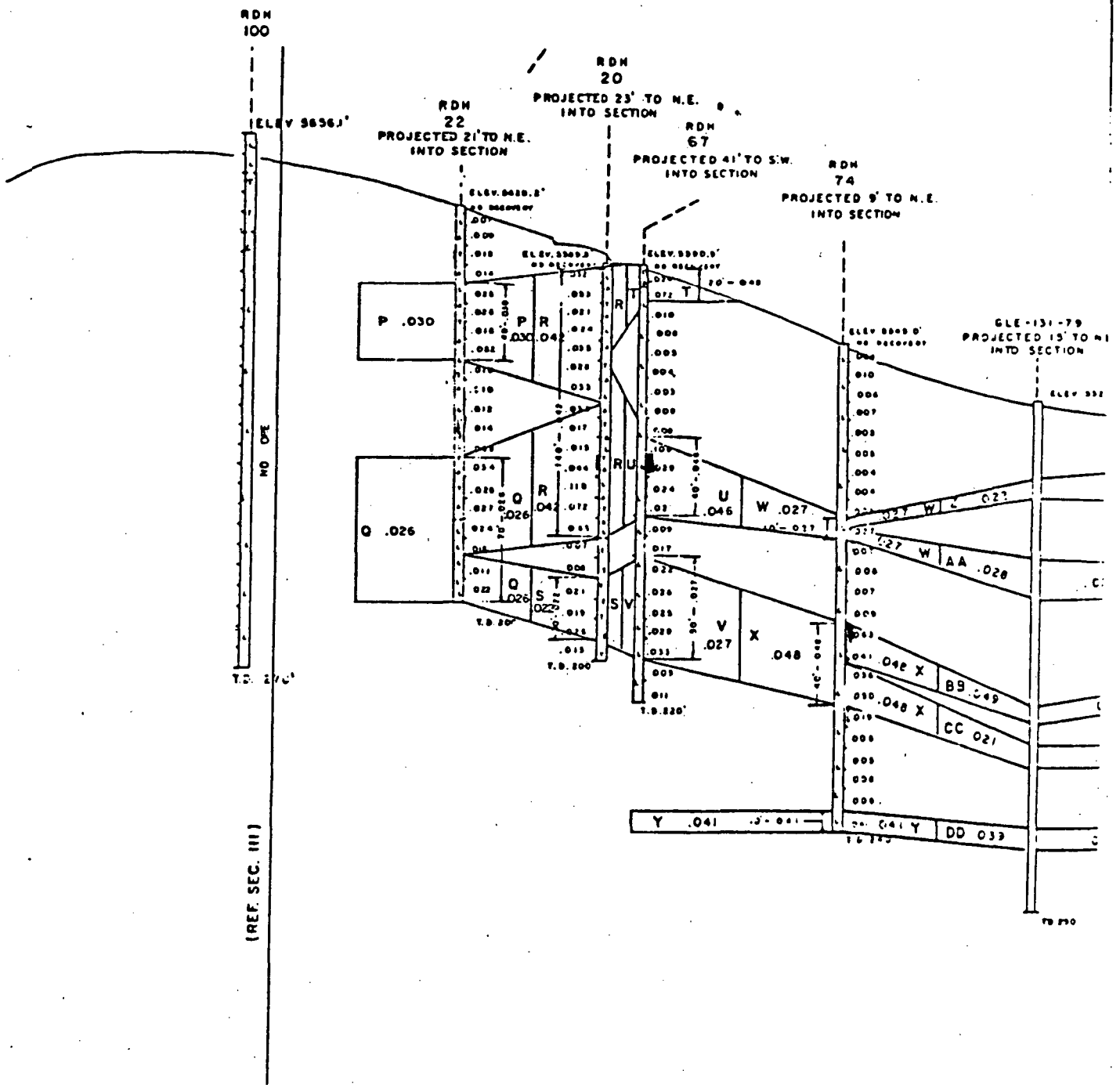


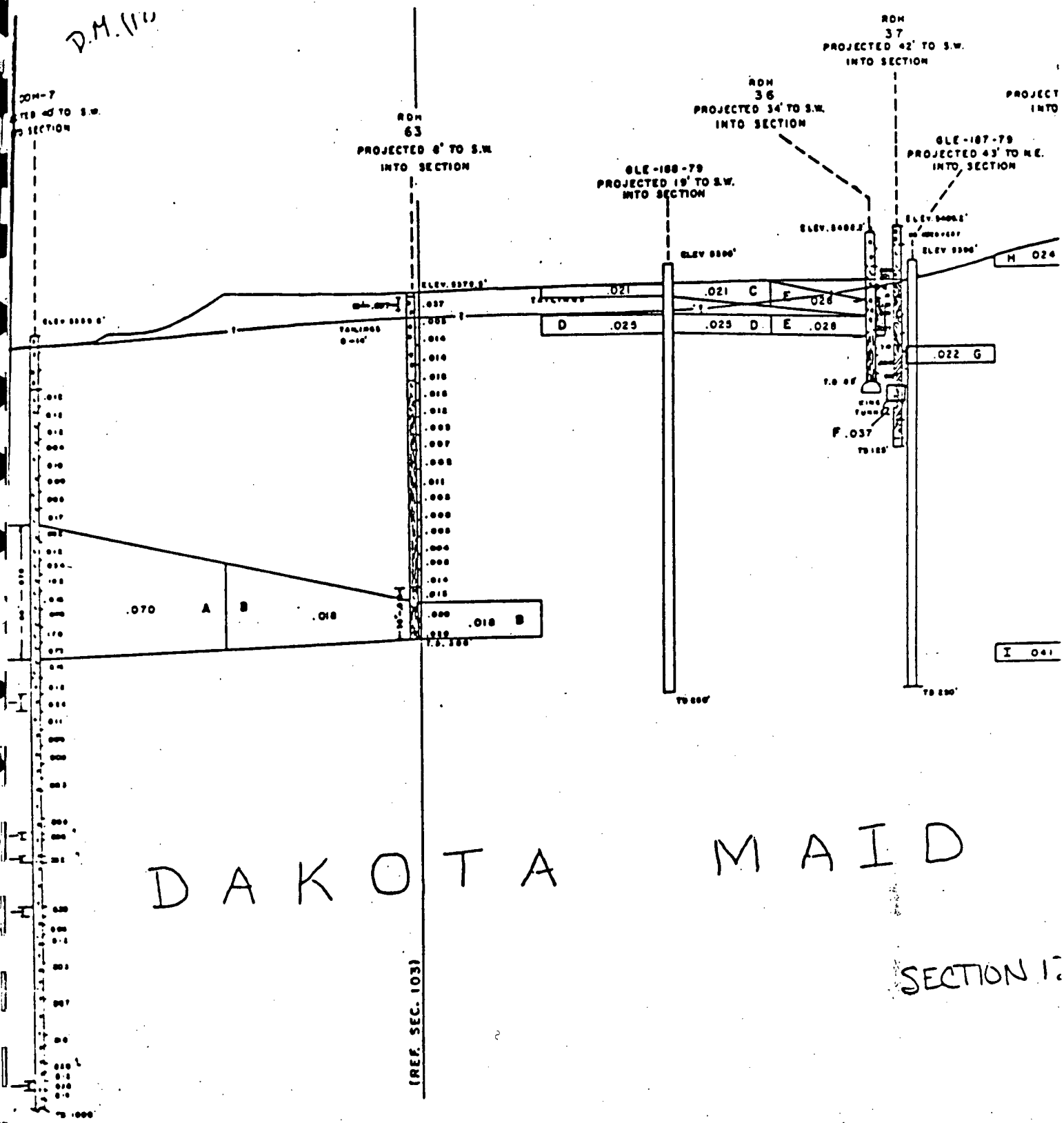
SUNDAY

SECTION 25



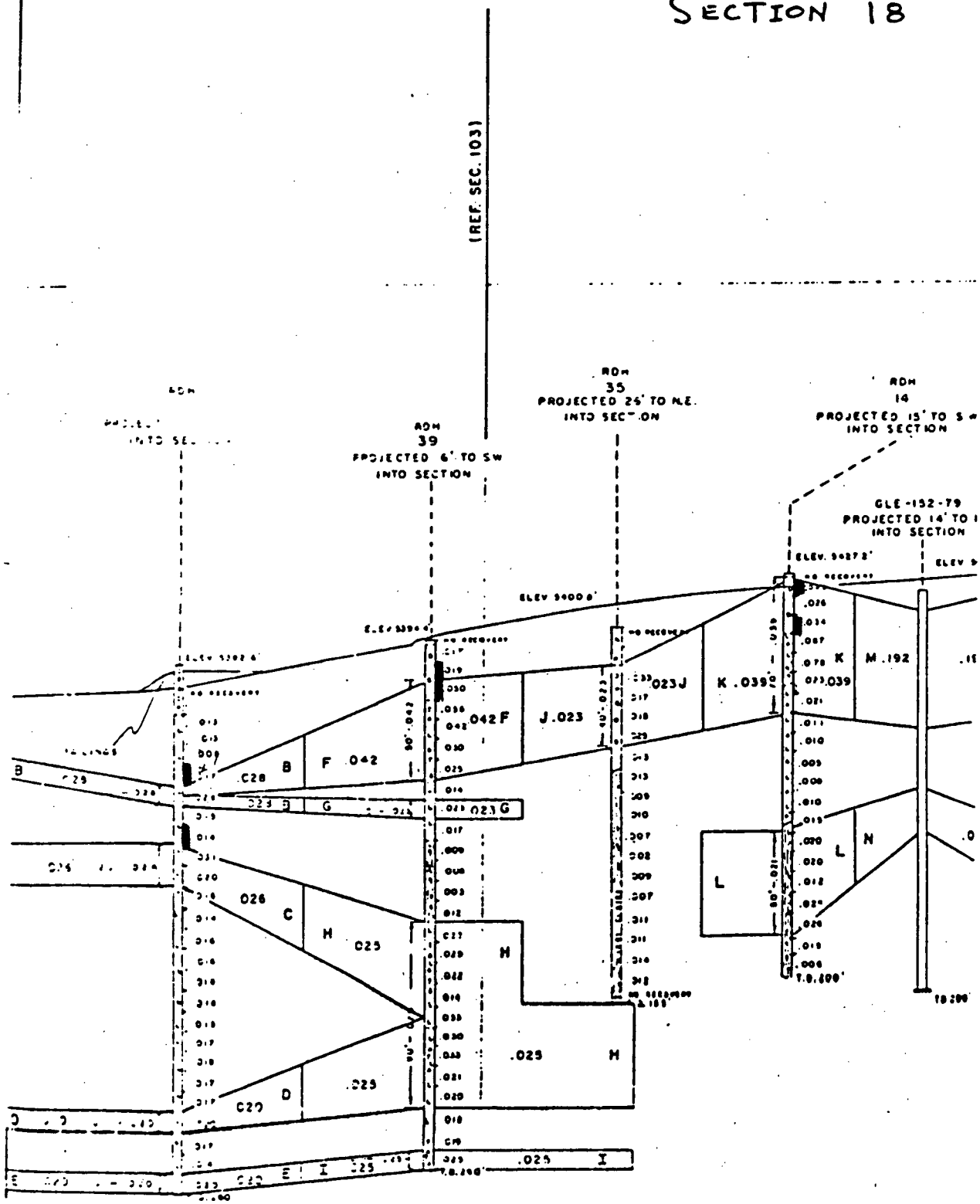
SUNDAY

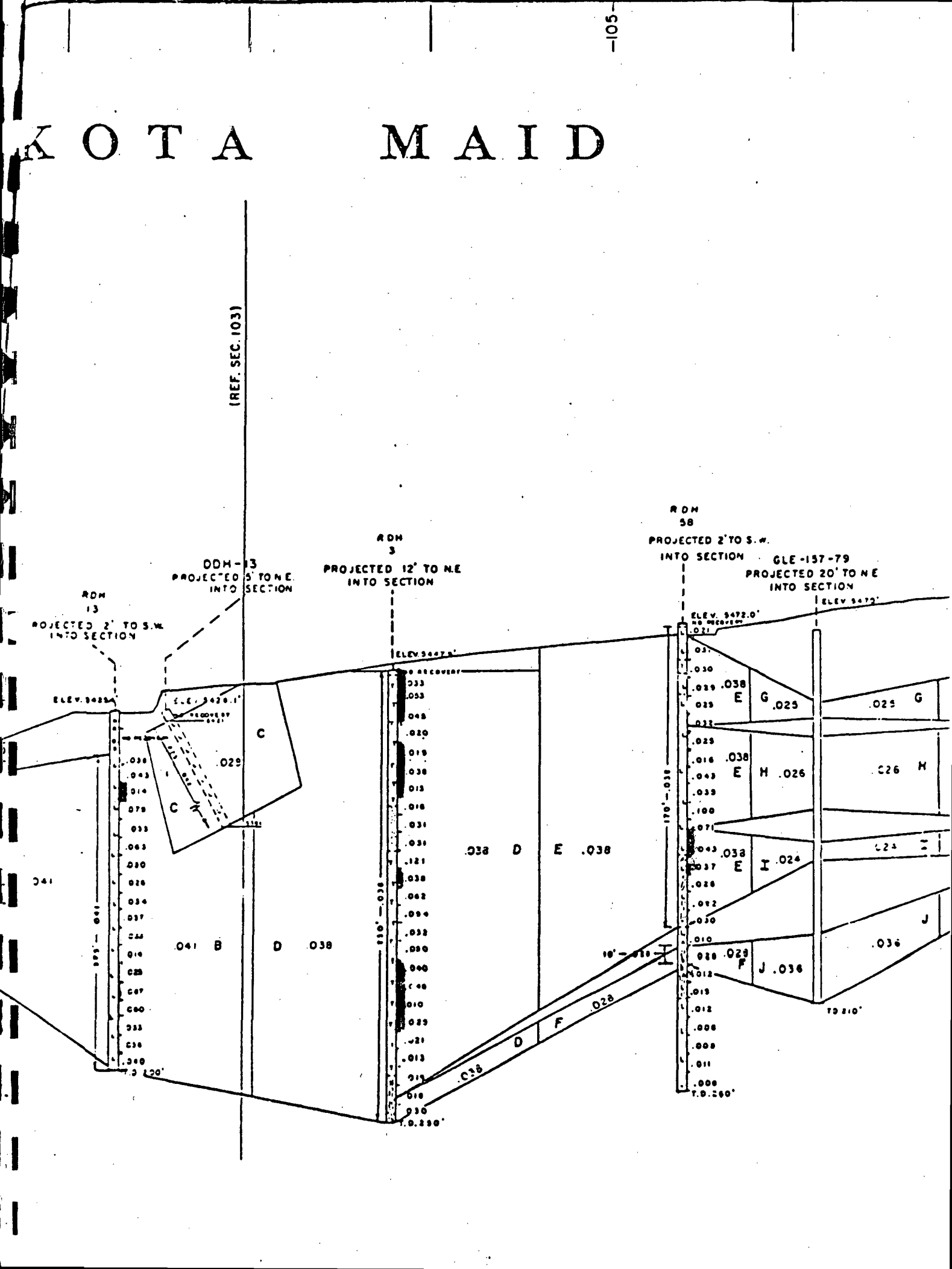

RDH 100
RDH 22
PROJECTED 21' TO N.E. INTO SECTION
RDH 20
PROJECTED 23' TO N.E. INTO SECTION
RDH 67
PROJECTED 41' TO S.W. INTO SECTION
RDH 74
PROJECTED 9' TO N.E. INTO SECTION
GLE-131-79
PROJECTED 15' TO N.E. INTO SECTION
NO OPE
(REF. SEC. III)
P .030
Q .026
R .042
S .022
T
U .046
V .027
W .027
X .048
Y .041
AA .028
BB .049
CC .021
DD .033


D.M. 110
RDH 63
PROJECTED 8' TO S.W.
INTO SECTION
GLE-188-79
PROJECTED 19' TO S.W.
INTO SECTION
RDH 36
PROJECTED 34' TO S.W.
INTO SECTION
RDH 37
PROJECTED 42' TO S.W.
INTO SECTION
GLE-187-79
PROJECTED 43' TO N.E.
INTO SECTION
.070 A
B .018
.018 B
.021 C
E .028
D .025
.025 D
E .028
.022 G
F .037
H 024
I 041
(REF. SEC. 103)
DAKOTA MAID
SECTION 1

DAKOTA MAID

SECTION 18

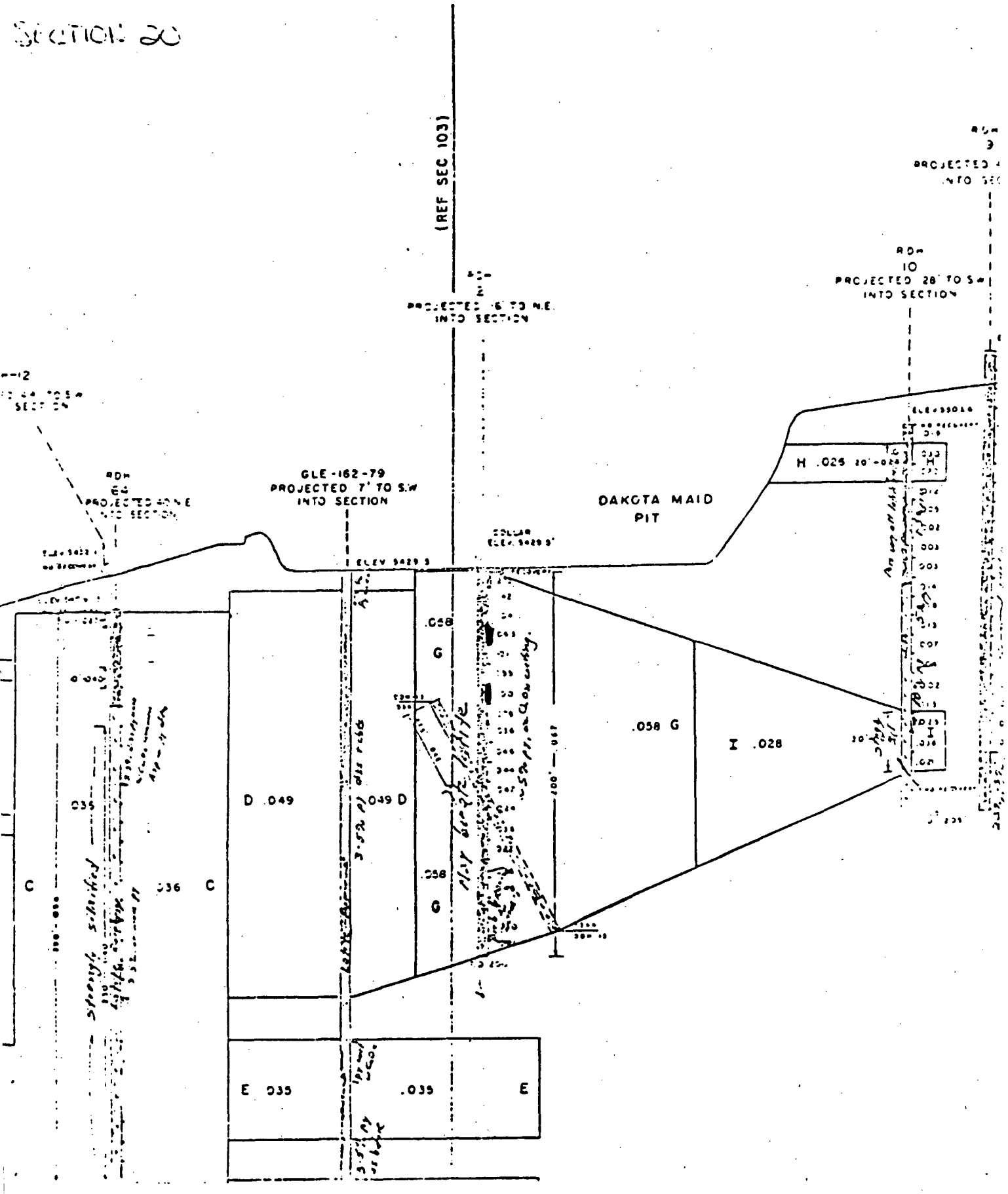




-105-
KOTA MAID
(REF. SEC. 103)
RDH 13
PROJECTED 2' TO S.W. INTO SECTION
DDH-13
PROJECTED 5' TO N.E. INTO SECTION
RDH 3
PROJECTED 12' TO N.E INTO SECTION
RDH 58
PROJECTED 2' TO S.W. INTO SECTION
GLE-157-79
PROJECTED 20' TO N.E INTO SECTION
ELEV. 5472.0'
NO RECOVERY
T.D. 200'
T.D. 250'
T.D. 260'
T.D. 210'

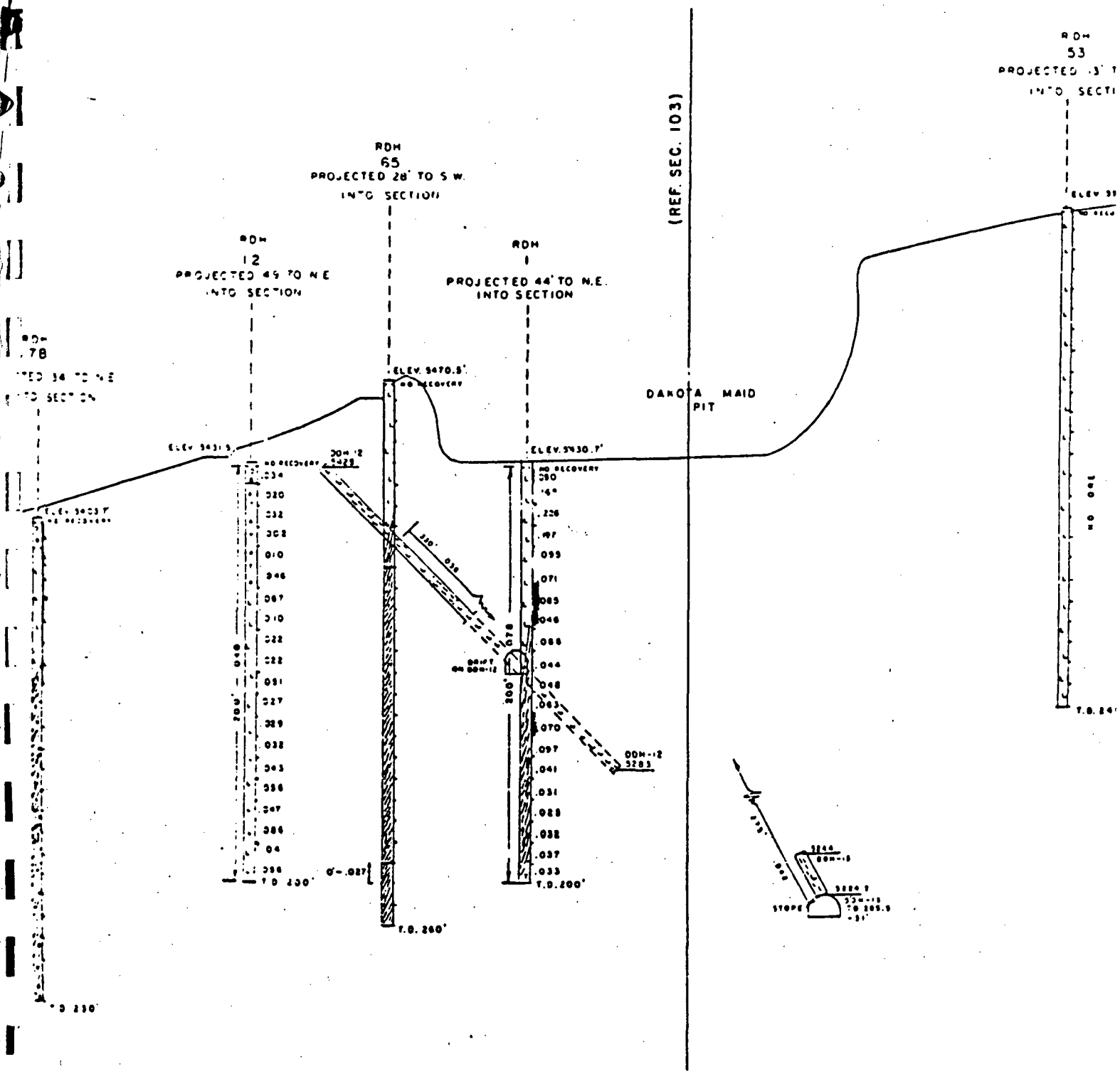
DAKOTA MAID

SECTION 20



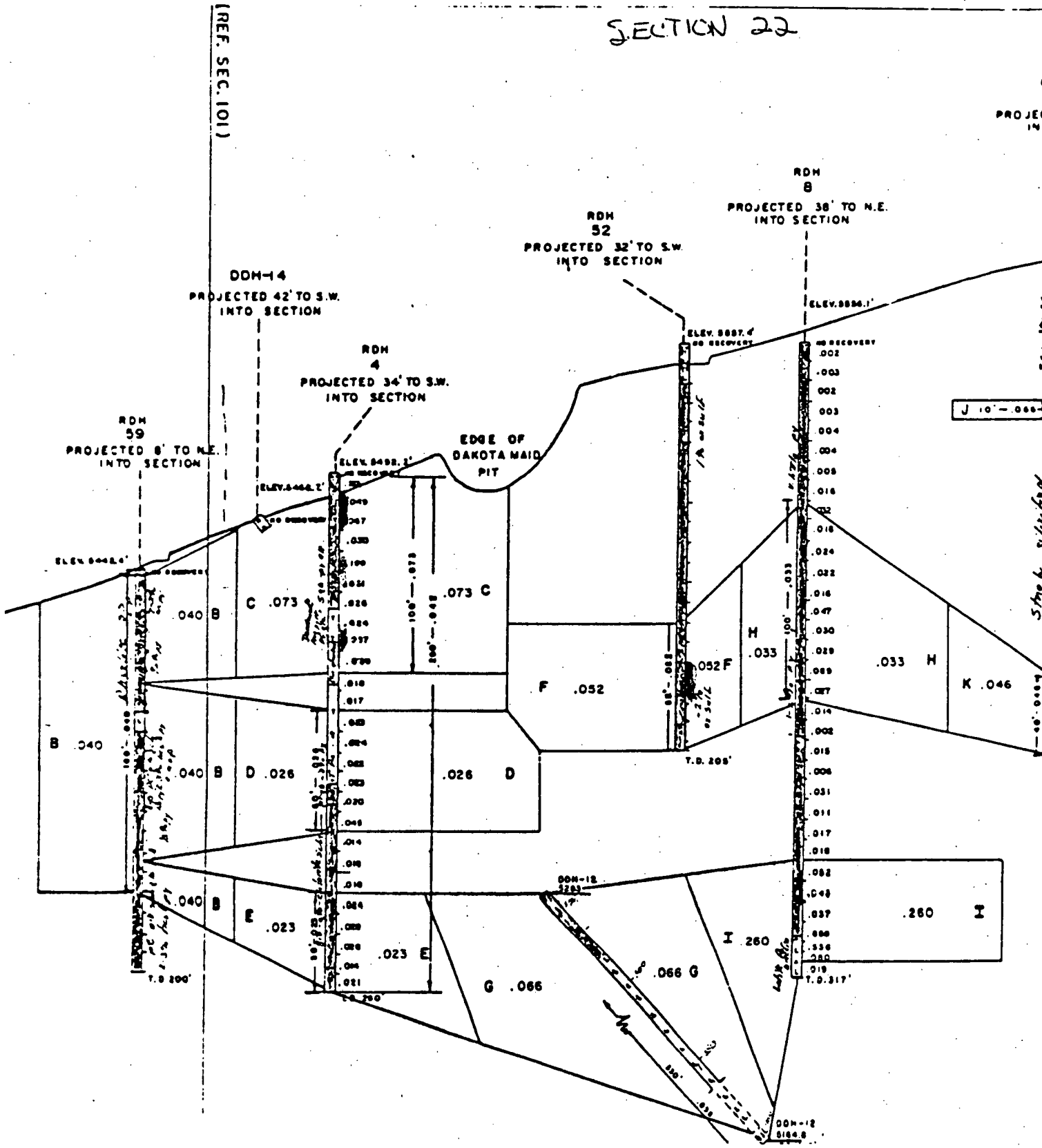
DAKOTA MAID

SECTION 21

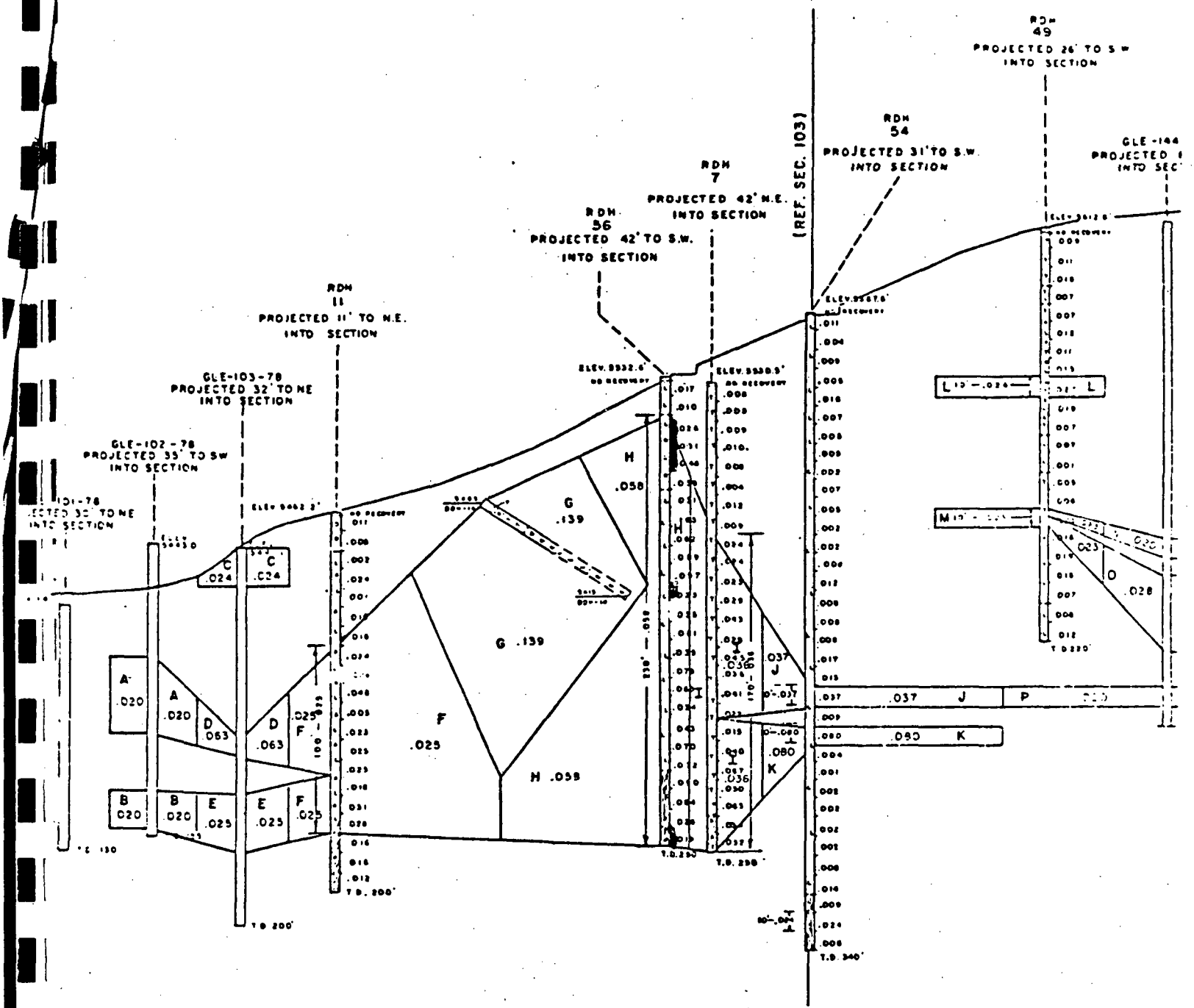


DAKOTA MAID

SECTION 22

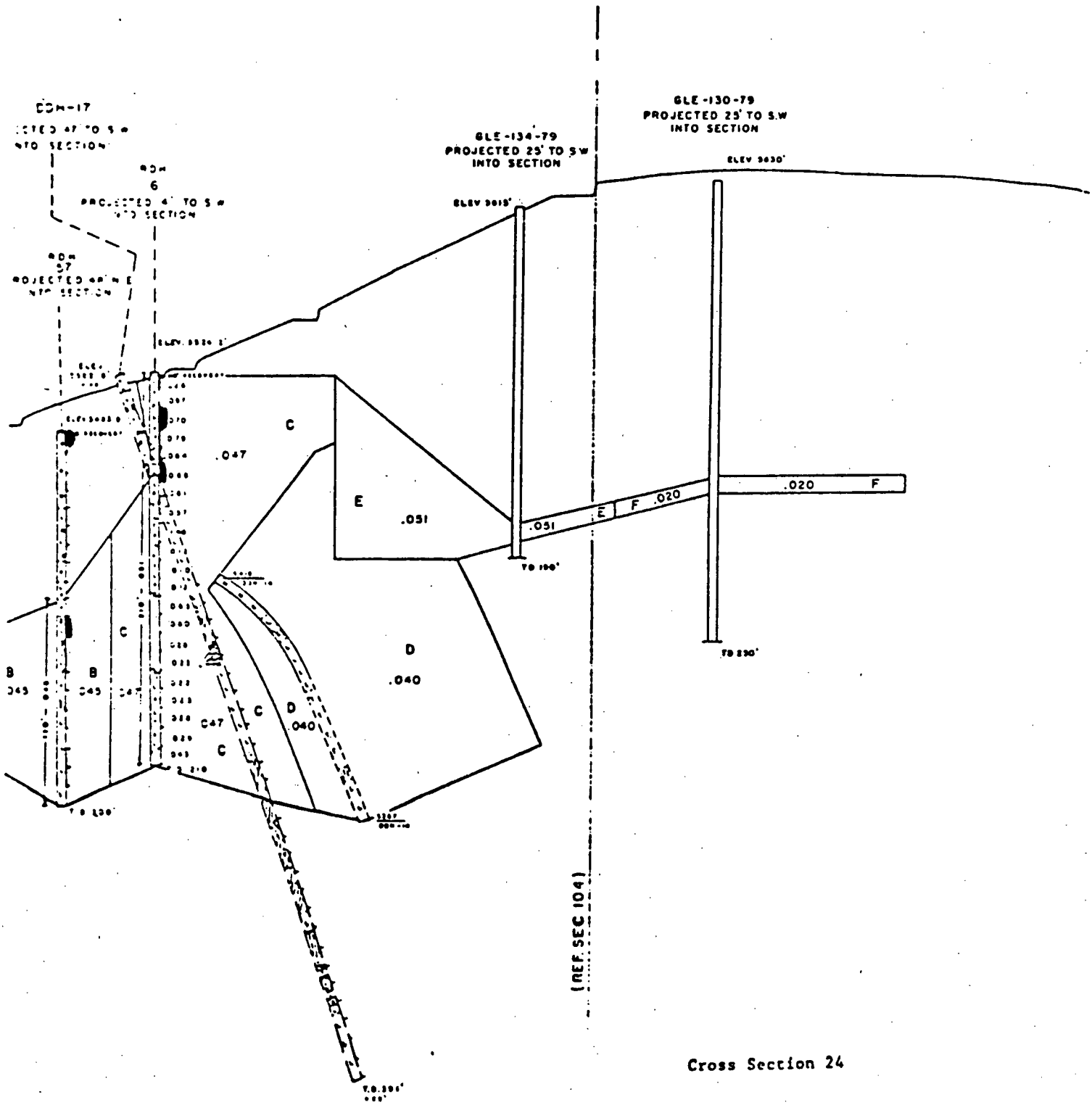


DAKOTA MAID



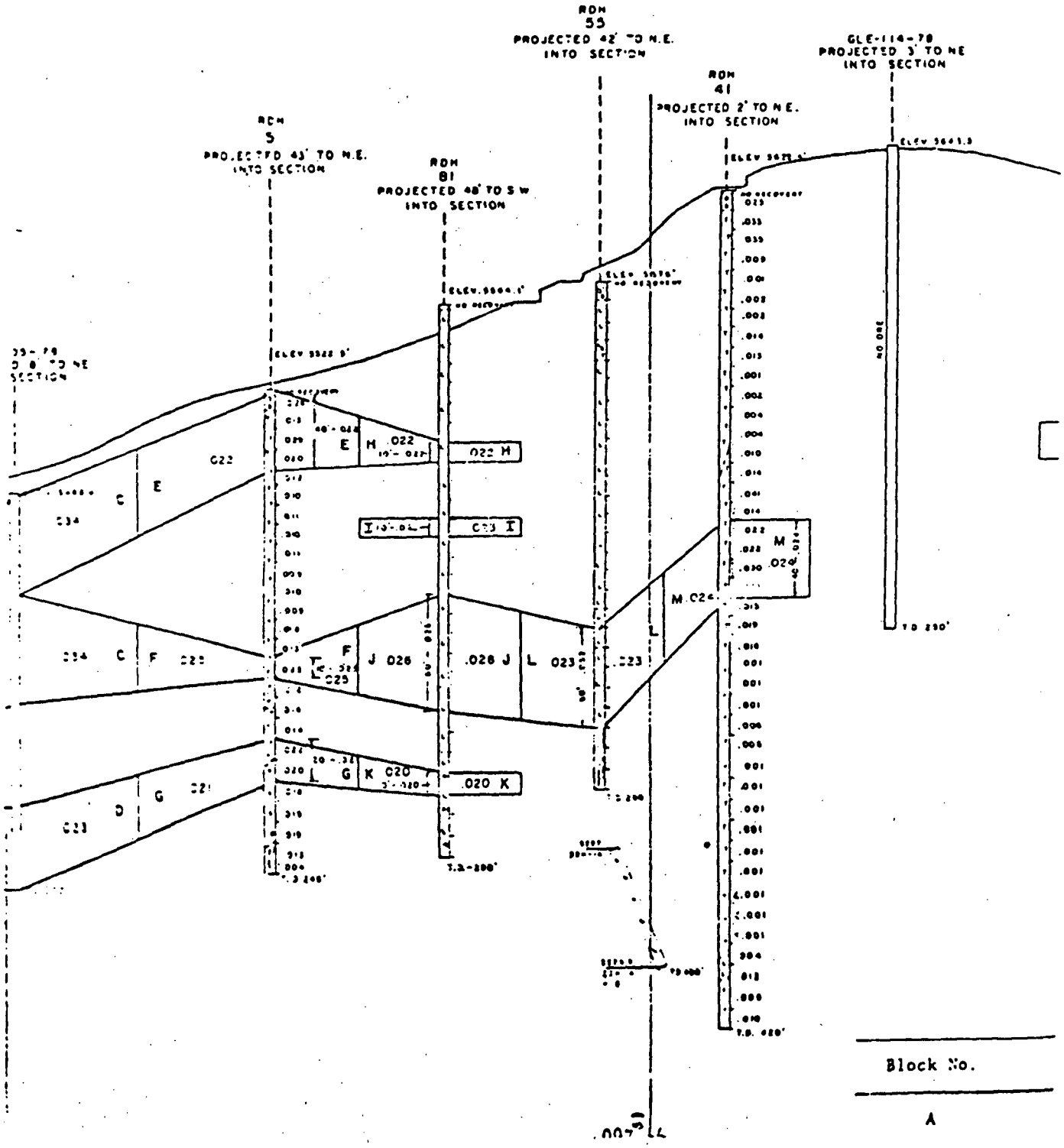
Block No.	Squar
A	C
B	C

DAKOTA MAID

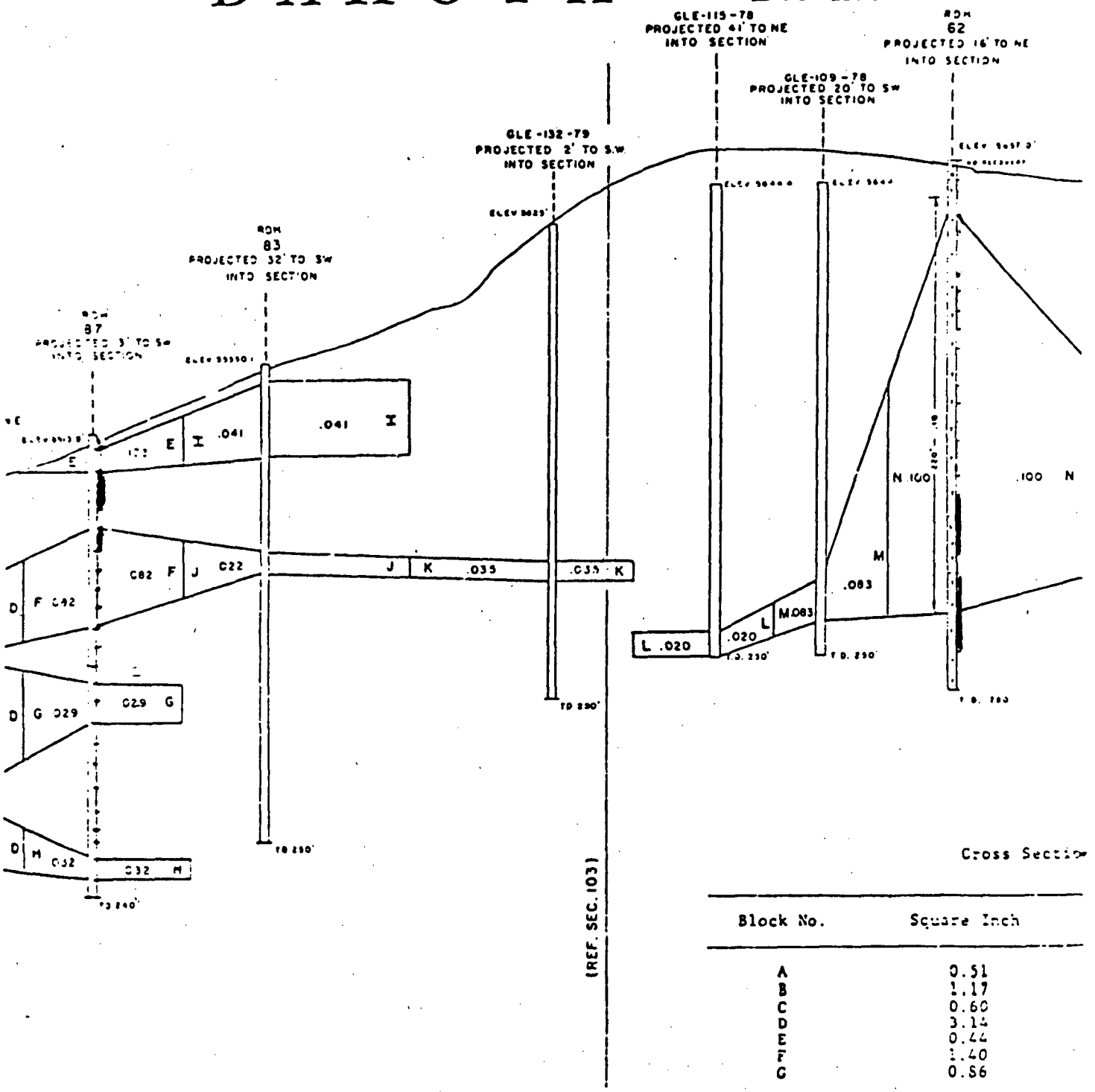


Cross Section 24

DAKOTA MAID


RDH 5
PROJECTED 43' TO N.E. INTO SECTION
RDH 81
PROJECTED 48' TO S.W. INTO SECTION
RDH 55
PROJECTED 42' TO N.E. INTO SECTION
RDH 41
PROJECTED 2' TO N.E. INTO SECTION
GLE-114-78
PROJECTED 3' TO NE INTO SECTION
ELEV 5622.9'
ELEV 5604.1'
ELEV 5643.3
NO RECOVERY
NO ORE
T.D. 250'
T.D. 420'
.022 H
.023 I
.020 K
.026 J
.023
M .024
Block No.
A

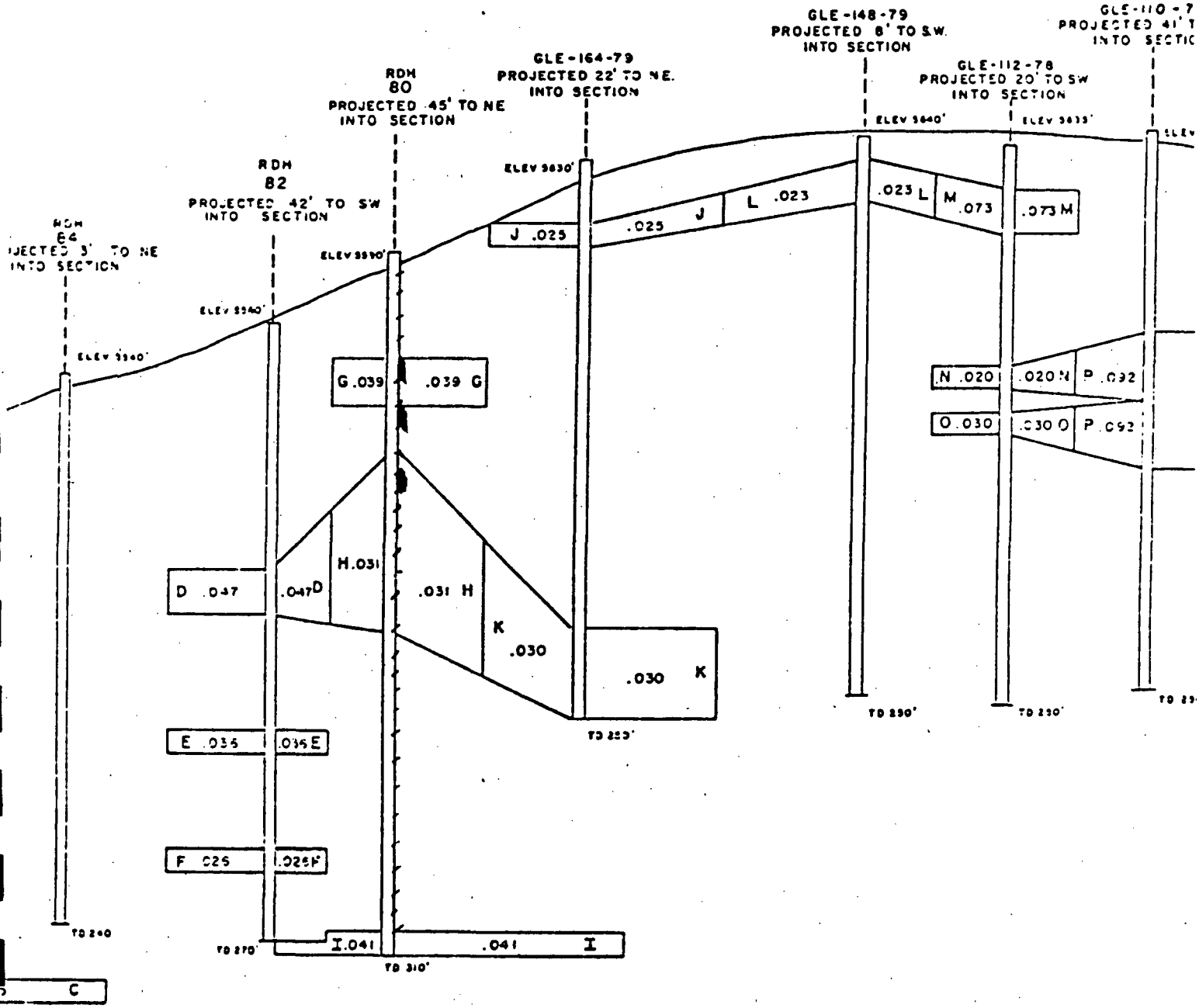
DAKOTA MAID



Cross Section

Block No.	Square Inch
A	0.51
B	1.17
C	0.60
D	3.14
E	0.44
F	1.40
G	0.86

DAKOTA MAID


RDH 84
PROJECTED 3' TO NE INTO SECTION
RDH 82
PROJECTED 42' TO SW INTO SECTION
RDH 80
PROJECTED 45' TO NE INTO SECTION
GLE-164-79
PROJECTED 22' TO NE. INTO SECTION
GLE-148-79
PROJECTED 8' TO S.W. INTO SECTION
GLE-112-78
PROJECTED 20' TO SW INTO SECTION
ELEV 5640'
ELEV 5630'
ELEV 5590'
ELEV 5540'
ELEV 5540'
J .025
.025 J
L .023
.023 L
M .073
.073 M
G .039
.039 G
N .020
.020 N
P .092
O .030
.030 O
P .092
D .047
.047 D
H .031
.031 H
K .030
.030 K
E .035
.035 E
F .025
.025 F
I .041
.041 I
TD 240
TD 270'
TD 310'
TD 250'
TD 250'
TD 250'
C

GILT EDGE BUCKET LEACH TESTS
1979 MINI-BULK SAMPLES

by

MILLER-KAPPES COMPANY

November 10, 1981

Miller-Kappes Company

P. O. Box 13687, Reno, Nevada 89507 702-356-7107

1845 Glendale Avenue, Sparks, Nevada 89431

10 November, 1981

FINAL REPORT

GILT EDGE BUCKET LEACH TESTS

1979 MINI-BULK SAMPLES

This report summarizes the results of a testing program on twelve samples of rock taken from the Gilt Edge, South Dakota properties by D. Kappes and M. Cassiday, during September, 1979.

The twelve mini-bulk samples were taken, using a hammer and moil to chip rock form the surface of the existing mine workings, from areas where one-ton bulk samples were not feasible. Each sample weighed 150 to 200 pounds, Seven of these samples were taken on the R-2 level of the Sunday shaft, 155 feet below the Rattlesnake level at elevation 5350, and represent ore in a deep, damp zone of mixed oxide/sulfide ore with some secondary copper enrichment. Two samples were taken from the walls of the Dakota Maid pit and represent near-surface sulfide ores. Three samples were taken in existing prospect drifts into the extreme north end of the Dakota Maid Zone in areas beyond the prime ore target zones. A set of maps included in this report show the sample locations.

Figure 1 presents sample descriptions and a summary of bucket leach test results. The four pyrite-containing unoxidized samples, 773H-K, averaged 52 percent recovery of contained gold. One of these, K, was from the deep-level Rattlesnake area and contained secondary chalcocite and green copper staining. Two of these, H and I, were highly pyritic Dakota Maid pit samples. The six deep-level Rattlesnake Zone oxidized samples, 773A, B, D-G, averaged 69 percent gold recovery (sample A was actually unoxidized but contained only pyrite).

Three samples, 773 C, I, and L, were taken from the extreme north end of of the Dakota Maid Zone. C and I both achieved 48 percent recovery (head grades were low), while L achieved 84 percent.

The overall conclusions from this series of tests are: 1) that the deep-level Rattlesnake ores, which were taken from a damp, salt-rich, copper-enriched area 50 feet above the present water table, will average about the same recoveries (71 percent) as the nearer-surface ores previously tested; and 2) that highly pyritic, Dakota Maid ores will yield recoveries in the range of about 51 percent, which also correlates with testing of similar ores from the King Tunnel; and 3) that north-end ore probably leaches similar to the ore in the major ore centers. The only major discrepancy noted in the present series of tests is the 60 percent recovery now achieved on sample 773H from the Dakota Maid pit, as compared with a recovery of 30 percent from a previous, identical sample.

Figure 2 presents a bar chart showing head grade and percent recoveries for all the samples.

SCOPE OF THE TESTING PROGRAM

The following tests were run on each of the samples:

1. Fire assays on separate size fractions of head samples.
2. Centrifuge tube cyanide leach tests on pulverized portions of the head samples.
3. Bucket leach tests on rock crushed to 5/8-inches.
4. Fire assays on separate size fractions of the bucket test tails.

SAMPLE PREPARATION

The as-received samples were dry or only slightly damp, and were processed without further drying. Weighing approximately 70 Kg each, the samples were typical rock chip samples with some fines and coarse fragments up to six inches maximum dimension.

Each of the samples, delivered to the lab in 2 - 4 large paper sacks were dumped onto a 5 foot x 5 foot piece of hypalon, the large pieces broken by hand to 3-inches or less, and mixed well. Using a small scoop, the material was split into two or three, approximately identical, 5-gallon portions, all but one of which were stored for future testing.

One 5-gallon portion from each sample was crushed in a jaw crusher to 5/8-inch. A head sample of approximately 6 Kg was split out from the 5/8-inch material using a Jones splitter, and the remainder of each sample was placed in a leach column.

The 6 Kg head sample was screened into various size fractions and weighed. Each of these size fractions was further crushed through a set of rolls, if necessary, to minus 6 mesh; then a 500 gram portion was split out and pulverized. The pulverized samples were used for a series of cyanide centrifuge tube tests and also for fire assays.

FIGURE 1. SAMPLE DESCRIPTIONS AND

BUCKET LEACH TEST RECOVERIES

SAMPLE NO.	LOCATION	DESCRIPTION	CN-SOLUBLE COPPER (ppm)	GOLD CONTENT/% RECOVERY (oz per ton/%)
773 A	Rattlesnake R-2 level. wide belled-out area in east drift	unoxidized, minor pyrite and chalcopyrite (less than 1%) color, pink-white. Moderately soft rock, apparent clay alteration of feldspars.	764	.056/69.79
773 B	same as 773 A	oxidized, otherwise same rock as 773 A. Considerable introduced hematitic staining. Color, 70% of rock fragments, pink-white, 30%, brick red.	50	.061/70.36
773 C	North end Dakota Maid Zone, "C" Adit, face of right hand cross-cut	oxidized, minor very tiny pyrite casts. Color, 70% of pieces brownish-white, 30% chocolate brown. Moderately hard rock, some apparent silicification, apparent clay alteration of feldspars.	1	.020/50.14
773 D	Rattlesnake R-2 level, from walls of caved stope at north end of east drift	oxidized, minor tinv pyrite casts (2%). Color of fragments, pink-white to medium pink. Moderately hard rock, apparent clay alteration of feldspars.	8	.034/79.65
773 E	Rattlesnake R-2 level, from walls of short drift into west wall of timbered stope in west drift (same stope area as 773 D, sample location about 75 ft from 773 D)	oxidized, no pyrite casts. Color, pink-white. Moderately hard rock, very little apparent clay alteration.	14	.016/49.24
773 F	Rattlesnake R-2 level, from walls of shaft station immediately east of shaft	oxidized and heavily acid-leached. About 3% pyrite casts. Color, uniform medium red-grey. Soft, vuggy rock.	5	.030/69.84
773 G	Rattlesnake R-2 level, from four corners of east drift intersection with shaft cross-cut (approx. 30 feet southeast from location of 773 E)	sample descricption identical to 773 D	3	.031/74.22
773 H	Dakota Maid Pit, northeast undercut, 150 feet along wall same location as previous sample DM-5 (see 1979 report, gold recovery from DM-5 29%, head assay 0.165 oz/ton)	mixed, approximately 15% unoxidized light-pink, hard, 5% pyrite, little clay alteration; 85% oxidized, brown-yellow and dark red, moderately hard with 5% pyrite casts.	131	.192/60.0
773 I	North end of property, Nevada Tunnel (Adit B), about half-way along tunnel, near Cyprus original sample location 8	unoxidized, 5% pyrite. Color of fragments 80% ivory, 20% brown, (oxidized fracture fillings). Moderately hard rock, apparent clay alteration of feldspars and probably some silicification.	16	.011/46.11
773 J	Dakota Maid Pit, sulfide zone in south undercut	unoxidized, approximately 30% grey quartzite, 60% moderate hard white volcanics with yellow fracture staining and 5% dissem pyrite, 10% vuggy agglomerates of pyrite crystals (apparently fracture fillings).	8	.091/47.16
773 K	Rattlesnake R-2 level, three-chute stope	unoxidized, 2% pyrite., 1% chalcocite on fractures (not coating pyrite). Hard rock, color light grey with light yellow fracture staining, some green copper staining.	1,188	.072/53.91
773 L	North end of property, Nevada Tunnel (Adit B), walls and back of tunnel near face	mixed oxidized/unoxidized, with minor pyrite	4	.033/84.89

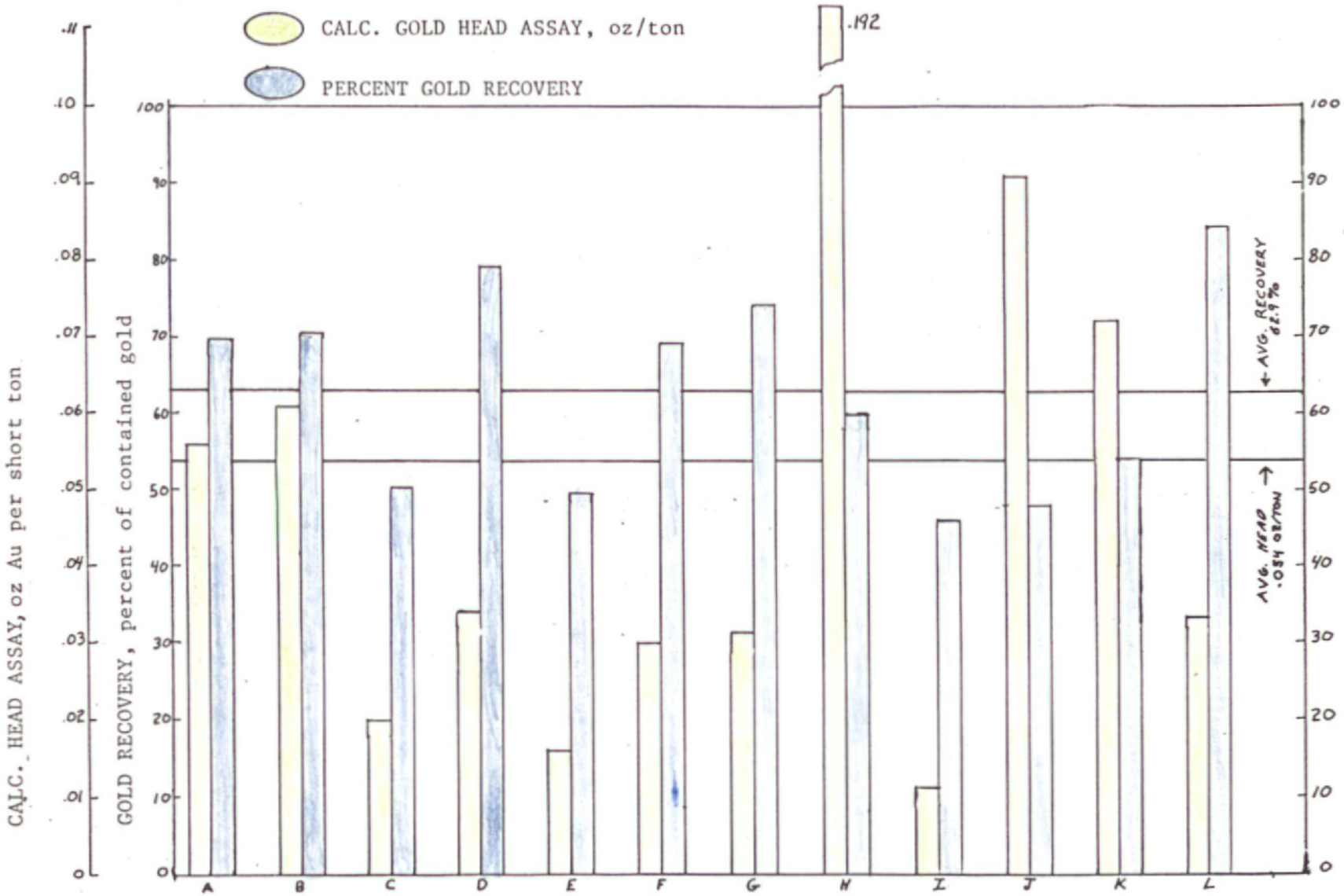


FIGURE 2. GILT EDGE MINI-BULK SAMPLES
12 BUCKET LEACH TESTS ON -5/8" ORE
ORE GOLD CONTENT AND GOLD RECOVERY STATISTICS

CENTRIFUGE TESTS ON HEAD SAMPLES: GOLD DISTRIBUTION IN HEAD SAMPLES

The pulverized pulps from the various screen sizes of the head samples were subjected to cyanide centrifuge tube tests, according to the following procedure:

1. Weigh out 10 grams of pulverized ore and place in centrifuge tube.
2. Add 25 mls of 5 gpl NaCN solution. Adjust pH, if necessary, to pH 10, using lime.
3. Place on wrist-action shaker for 24-hours.
4. Centrifuge and filter through glass wool.
5. Check solution for pH, Au and Ag. Discard residue.

Figure 3 shows head assays of the pulps (fire assays), the percent recovery of contained gold in a 24-hour cyanide centrifuge tube test on pulverized material, and the product fineness (ratio of gold to gold plus silver, times 1000). There are large variations in the reported percent recoveries, probably due to the presence of coarse gold in the samples.

Gold distribution by size fraction is also shown in Figure 3 for each of the 12 samples, and the average distribution for all the samples is shown in the table below. Dakota Maid and Sunday Zone samples were combined in calculating the average distribution, as there appeared to be no significant difference between them.

SIZE FRACTION	WEIGHT PERCENT OF TOTAL SAMPLE IN FRACTION	GOLD oz/ton	PERCENT OF TOTAL GOLD IN SIZE FRACTION
+ 3 mesh	59.5	0.063	63.0
- 3 + 65 mesh	36.4	0.040	24.5
-65 +150 mesh	1.7	0.133	3.7
- 150 mesh	2.4	0.214	8.8

The data clearly indicates that the gold is concentrated into the smaller size fractions, which is an indicator that it occurs primarily on fracture surfaces within the rock.

FIGURE 3. AGITATED CYANIDE LEACH TESTS

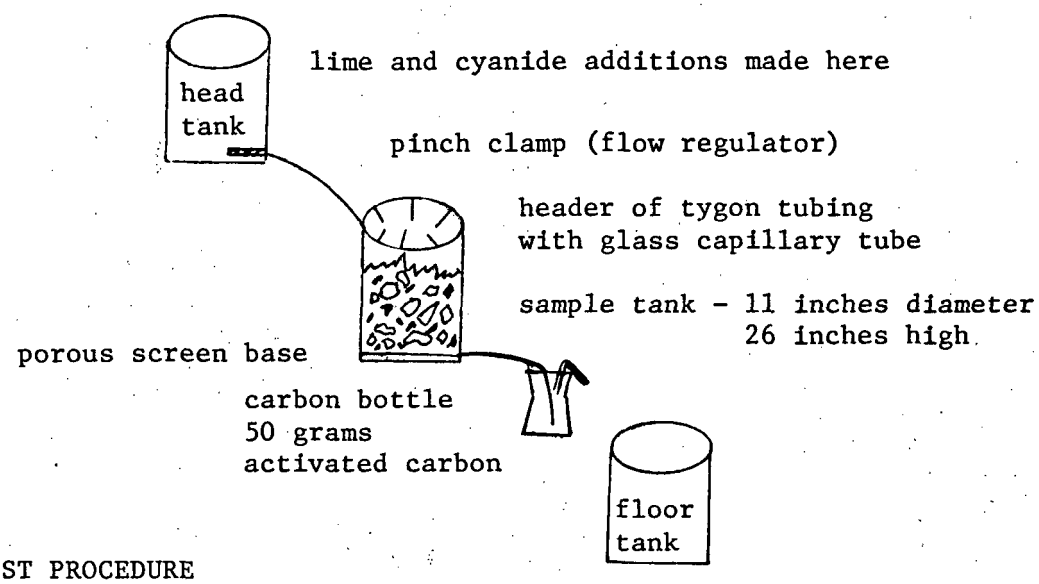
ON PULVERIZED PORTIONS OF SAMPLE SIZE FRACTIONS

(oz gold per ton/percent gold recovery/gold fineness)

SAMPLE NO.	BUCKET TEST NO.	+3 mesh	-3 + 65 mesh	- 65 + 150 mesh	- 150 mesh	WEIGHTED AVERAGE
773 A	774	.072/ 81.9%/655	.042/ 66.7%/583	.062/270.9%/423	.540/107.6%/645	.072/ 86.1%/625
773 B	775	.040/ 70.0%/549	.044/120.4%/726	.078/102.6%/800	.118/111.0%/704	.044/ 90.9%/621
773 C	776	.012/ 75.0%/ 25	.010/240.0%/118	.032/115.6%/ 62	.078/ 98.7%/ 61	.013/130.8%/ 58
773 D	777	.016/ 93.7%/577	.018/161.1%/744	.088/143.2%/863	.212/106.6%/834	.023/117.4%/649
773 E	778	.003/400.0%/571 (tr)	.003/266.7%/444 (tr)	.054/ 85.2%/807	.062/103.2%/780	.005/240.0%/534
773 F	779	.028/110.7%/663	.016/150.0%/706	.068/ 73.5%/833	.122/104.1%/830	.027/114.8%/685
773 G	780	.016/ 93.7%/349	.012/133.3%/348	.066/ 57.6%/731	.106/ 74.5%/687	.018/ 94.4%/363
773 H	781	.120/ 90.8%/122	.160/ 98.1%/194	.430/ 62.3%/221	.530/ 87.4%/177	.150/ 92.0%/151
773 I	782	.012/ 75.0%/ 54	.032/ 37.5%/ 57	.036/ 72.2%/ 43	.046/ 84.8%/ 33	.021/ 52.4%/ 54
773 J	783	.068/ 67.6%/267	.074/ 55.4%/194	.352/ 81.2%/286	.524/ 66.4%/177	.082/ 67.1%/238
773 K	784	.332/ 82.7%/846	.082/ 70.7%/659	.244/ 82.8%/811	.152/ 52.6%/559	.235/ 80.8%/770
773 L	785	.016/150.0%/480	.024/120.8%/491	.076/165.8%/829	.130/109.2%/721	.023/130.4%/496
AVERAGE		.061/ 86.3%/430	.043/ 92.6%/439	.132/ 91.6%/551	.218/ 90.0%/517	.059/ 88.1%/437

BUCKET LEACH TEST APPARATUS

The apparatus for the 5/8-inch rock tests is shown in the drawing below.



LEACH TEST PROCEDURE

In the apparatus shown above, the center tank or leach tank, was filled with the rock to be leached.

Alkaline cyanide solution was continuously distributed onto the rock from the head tank through a set of glass capillary drip tubes. Flowrate of solution dripping onto the rock was controlled using a pinch clamp, to approximately 12,000 ml per 24 hours, or 0.0035 gpm per square foot of heap top surface.

Solutions entering the floor tank were assayed periodically (initially every two days, averaging once/10 days over the life of the tests) for cyanide and lime, and reagents added as necessary to maintain solutions at "target" levels.

Solutions exiting the leach tank flowed continuously through a bottle of activated carbon and then into a floor tank. The 12,000 ml of active solution in the system was recycled to the head tank every 48 to 72 hours, so that the average flowrate over the life of the tests was 0.0015 gpm per square foot of heap top surface.

The tanks were kept covered at all times to minimize evaporation and cyanide loss. No makeup water was required.

The charge of activated carbon was removed three times during the tests and assayed to determine the amount of gold and silver leached from the ore. Carbon changes were made on days 10, 24 and at the end of the tests on day 107 (11 tests) or 281 (1 test on sample 773K). Of the 12 tests begun in April, 1980, 11 were ended in July, 1980. The test on sample 773K was ended in January, 1981 after the fourth carbon was removed.

RATE OF GOLD RECOVERY

A summary of test behavior is tabulated in Figure 4 and discussed below.

Sunday Zone. Samples D, E, F and G all had similar gold recovery rates. On average, 81 percent of recoverable gold (55 percent of contained gold) was recovered by day 10 onto the carbon. Between days 10 and 24, an additional 6 percent of recoverable gold (4 percent of contained gold) was recovered onto the carbon. Between days 24 and 107, an additional 13 percent of recoverable gold (9 percent of contained gold) was recovered onto the carbon.

Sample B showed slow, but similar, overall recovery as the other Sunday Zone samples. After 10 days, 45 percent of recoverable gold (31 percent of contained gold) was recovered onto the carbon. Between days 10 and 24, the test produced an additional 22 percent of recoverable gold (16 percent of contained gold); and between days 24 and 107, an additional 33 percent of recoverable gold (23 percent contained gold).

Samples A and K had chemical problems with leaching that caused delayed gold recoveries. The history of these tests is discussed elsewhere.

Dakota Maid. All the Dakota Maid samples behaved similarly, except for H which had an initial delay in gold recovery due to chemical problems. On average, 52 percent of total recoverable gold (39 percent of contained gold) was recovered by day 10. Between days 10 and 24 and additional 35 percent of recoverable gold (12 percent of contained gold) was recovered onto the carbon. Between days 24 and 107 an additional 13 percent of recoverable gold (7 percent of contained gold) was recovered onto the carbon.

SOLUTION CLARITY AND COLOR

Sunday Zone. Initial solutions from all samples were clear and colorless, with the exception of samples A, B and K.

Initial solution from sample B was cloudy. Initial solution from K was clear with a slight green tint. Initial solution from sample A was clear and purple and remained this color throughout the remainder of the test.

Dakota Maid Zone. Initial solutions from samples H, I and J were all clear and blue. Solutions in these tests became slightly yellow after pH became alkaline and remained so throughout the tests.

Initial solution from sample B was clear and brown while sample L was clear and colorless.

TEST HISTORIES

The twelve tests, seven on samples from the R-2 level of the Sunday Zone, and five on samples from the Dakota Maid Zone, were all run on 5/8-inch rock. A summary of test histories is tabulated in Figure 4 and discussed below.

Start-up of Tests. The initial leach solution for all tests consisted of 16 liters of solution containing 1.0 grams NaCN per liter and 0.50 grams $Ca(OH)_2$ per liter. Initial solutions exiting tests varied in pH and NaCN and are discussed elsewhere in this report.

Sunday Zone Sample A had an initial solution that was clear, lavender purple with an acidic pH. Solution throughout the entire test remained purple (cause of the strong purple color was not determined). Solution became alkaline by day 4 after adding the equivalent of 2.4 lbs/ton lime. Copper content of the test solution was above 700 ppm by day 37. After addition of excess cyanide on day 41, copper content climbed to 1200 ppm by day 68, equivalent to 1.5 pounds copper dissolved per ton of ore. Low gold recovery early in the test was probably a function of the high copper content of leach solution, which serves to complex available cyanide and keep it from dissolving gold.

Sunday Zone Sample B had an initial solution that was murky brown in color with an acidic pH. Solution became alkaline by day 4 after adding the equivalent of 2.9 lbs/ton of lime. Copper content of the leach solution was negligible: 70 ppm by day 58.

Sunday Zone Sample K had an initial solution that was light green with an acidic pH. Solution became alkaline on day 4 after adding the equivalent of 2.4 lbs/ton lime. Copper content of the leach solution was above 700 ppm by day 37. After addition of excess cyanide on day 41, copper content climbed abruptly to 1800 ppm, equivalent to 2.1 pounds copper dissolved per ton of ore.

Initial gold recovery was slow; 30 percent of recoverable gold (16 percent of contained gold) was recovered through the second carbon period on day 26. During the third carbon period, in which excess cyanide was added, an additional 52 percent of recoverable gold (38 percent of contained gold) was recovered. The test was allowed to run through a fourth carbon period and ended after 281 days. An additional 17.7 percent of recoverable gold (13 percent of contained gold) was recovered onto the C-4 carbon.

Dakota Maid Zone Sample H solutions had an acidic pH of 5.2 and a blue precipitate (ferric ferrocyanide). After adding the equivalent of 4.2 lbs/ton lime, solutions became alkaline on day 6. Copper content of the test solution was moderate: 200 ppm by day 58. [NOTE: This sample was taken from same location as 1978 mini-bulk sample DM-5 which had recovery of 28.5 percent.]

Dakota Maid Zone Sample J initial solutions were also acidic and dark blue in color. After the addition of 3.1 lbs/ton lime, solutions became alkaline on day 6. Copper in solution was low, only 14.5 ppm by day 58.

Dakota Maid Samples C and I both had marginal recoveries of 50 and 46 percent respectively, however, calculated head assays were only 0.020 and 0.011 oz/ton gold, respectively.

Dakota Maid Zone Sample L leached well without any problems, and with 88 percent of recoverable gold (74 percent of contained gold) recovered by day 9.

FIGURE 4. SUMMARY OF BUCKET TEST HISTORIES

(See also Figure 6 for overall lime and cyanide consumption)

SAMPLE NO.	LIME ADDED TO ACHIEVE ALKALINITY (lbs $Ca(OH)_2$ per ton)	FINAL COPPER CONTENT OF SOLUTION ppm	INITIAL SOLUTION COLOR	FINAL SOLUTION COLOR	HIGHEST "TARGET" LEVEL OF NaCN IN SOLUTION gpl	% OF TOTAL GOLD RECOVERED (Based on 100 Percent) By Day 10	By Day 24
773 A	2.43	1250	Murky	Purple	9.70	30.0	65.9
773 B	2.85	71	Murky		.76	44.7	67.3
773 C	.76	2	Brown		.78	72.7	88.5
773 D	.94	11	Clear		.98	90.2	96.5
773 E	.87	20	Clear		.92	90.1	97.1
773 F	1.00	7	Clear		.87	52.3	87.1
773 G	1.10	4	Clear		.85	88.7	96.0
773 H	4.23	206	Blue		.76	33.5	72.6
773 I	5.31	28	Blue		.79	62.8	82.8
773 J	2.31	14	Blue		.80	78.3	96.3
773 K	2.37	2200	Green		11.75	15.55	30.1
773 L	.71	7	Clear		.81	87.7	97.2

GOLD AND SILVER RECOVERIES

Figure 5 tabulates gold and silver recoveries from the bucket leach tests. The data is tabulated in actual milligrams gold or silver per sample, and can be converted to oz/ton by reference to the sample weights shown in Figure 5.

In Figure 3, fineness of recovered gold (ratio of gold to gold plus silver, times 1000) was shown for agitated tests on pulverized samples, and averaged 437 fine. Figure 5 shows an average fineness of 619 for the same samples, leached at 5/8-inch in bucket leach tests.

Gold recoveries and head grades are shown in bar chart form, in Figure 2.

TAILINGS ASSAY AND METALLURGICAL BALANCES

At the end of the tests, the test tailings were dried and then screened into various size fractions. The size fractions were crushed, if necessary, to 100 percent minus 6 mesh, and then a portion was split out to be pulverized. Duplicate fire assays were run on the pulverized material. Average tailings assays are reported in Figure 6.

Metallurgical balances for the 12 tests are shown in Figure 7. Seven of the 12 tests showed higher assay heads than calculated heads, three showed the reverse, two showed calculated heads identical with assay heads. Overall metallurgical balance was 91.5 percent, and nine of the twelve tests are considered to have shown moderate or close correlation. The noise in the metallurgical balance calculation is to be expected because of the coarse size (5/8-inch) of rock used for the tests, and because of the occurance of coarse gold in the Gilt Edge ores. The calculated head, based on actual recoveries and assays of the fine-crushed tailings, is considered more accurate than assay head.

FIGURE 5. BUCKET LEACH TESTS

CARBON RECOVERIES AND GOLD FINENESS

SAMPLE NO.	MILLIGRAMS GOLD/SILVER RECOVERED ONTO CARBON DAY 10	DAY 24	DAY 107	DAY 281	TOTAL	MILLIGRAMS GOLD IN ORE TAILINGS	TOTAL CALC. MG. GOLD IN SAMPLE	FINENESS OF RECOVERED GOLD
773 A	7.62/ 4.52	9.14/ 3.72	8.67/ 2.17		25.43/10.41	11.01	36.44	709
773 B	11.06/ 1.12	5.60/ 1.92	8.08/ 0.00		27.74/ 3.04	10.42	35.16	901
773 C	5.14/51.74	1.12/22.04	.81/15.62		7.07/89.40	7.03	14.10	73
773 D	13.60/ 1.58	.94/ .32	.53/ 0.00		15.07/ 1.90	3.85	18.92	888
773 E	4.38/ .46	.34/ .18	.14/ 0.00		4.86/ .64	5.01	9.87	884
773 F	5.98/ 1.10	3.98/ .38	1.48/ 0.00		11.44/ 1.48	4.94	16.38	885
773 G	10.32/ .70	.84/ .48	.47/ 0.00		11.63/ 1.18	4.04	15.67	908
773 H	24.14/35.10	28.16/65.68	19.75/78.01		72.05/178.79	48.03	120.08	287
773 I	2.38/40.40	0.76/14.98	.65/ 1.99		3.79/57.37	4.43	8.22	62
773 J	24.28/70.26	5.58/32.14	1.14/ 4.86		31.00/107.56	34.74	65.74	224
773 K	4.10/ 1.94	3.38/ 1.44	13.83/ 2.09	4.68/ 3.11	26.49/ 8.58	22.65	49.14	755
773 L	18.32/ 2.64	2.00/ .96	.58/ 0.00		20.90/ 3.60	3.72	24.62	853

FIGURE 6. GILT EDGE MINI-BULK SAMPLES

BUCKET LEACH TEST - TAILINGS ASSAYS AND REAGENT CONSUMPTION

SAMPLE NO.	CALC. HEAD ASSAY oz Au/ton	+½"	- ½"+3m	-3+10m	-10+65m	-65+150m	-150m	TOTAL	POUNDS/SHORT TON LIME AND CYANIDE CONSUMPTION $Ca(OH)_2$	NaCN
773 A	0.056	3118	8008	5276	2075	189	225	18,891	6.08	23.10
		0.020	0.018	0.028	0.028	0.028	0.024	0.017		
773 B	0.061	2985	7095	4225	1945	231	401	16,882	7.86	7.14
		0.014	0.016	0.026	0.012	0.018	0.022	0.018		
773 C	0.020	4295	9260	4370	2026	232	325	20,508	6.50	6.17
		0.016	0.003	0.016	0.014	0.010	0.014	0.010		
773 D	0.034	2010	8658	2845	1850	240	445	16,048	2.11	4.93
		0.012	0.003	0.010	0.010	0.022	0.018	0.007		
773 E	0.016	1775	8655	5265	1985	245	350	18,275	2.62	4.15
		0.014	0.010	None	0.016	None	0.012	0.008		
773 F	0.030	795	6453	5530	2540	408	295	16,021	7.48	6.23
		0.010	0.010	0.010	None	0.022	0.020	0.009		
773 G	0.031	2055	6528	3950	1685	203	308	14,729	3.45	5.66
		0.003	0.010	0.003	0.012	0.003	0.036	0.008		
773 H	0.192	2305	8733	4747	1803	242	366	18,196	9.63	6.67
		0.054	0.078	0.068	0.096	0.152	0.054	0.077		
773 I	0.011	4250	9230	4355	2875	335	495	21,540	11.60	6.17
		0.012	0.003	0.003	0.010	0.010	0.003	0.006		
773 J	0.091	3380	9108	4960	2568	530	565	21,111	5.40	6.18
		0.012	0.024	0.044	0.124	0.198	0.188	0.048		
773 K	0.072	4450	8055	4805	1992	302	423	20,027	4.95	28.32
		0.067	0.021	0.026	0.025	0.036	0.033	0.033		
773 L	0.033	2542	8878	6713	3082	251	240	21,706	2.04	4.61
		0.003	0.010	None	0.003	None	None	0.005		

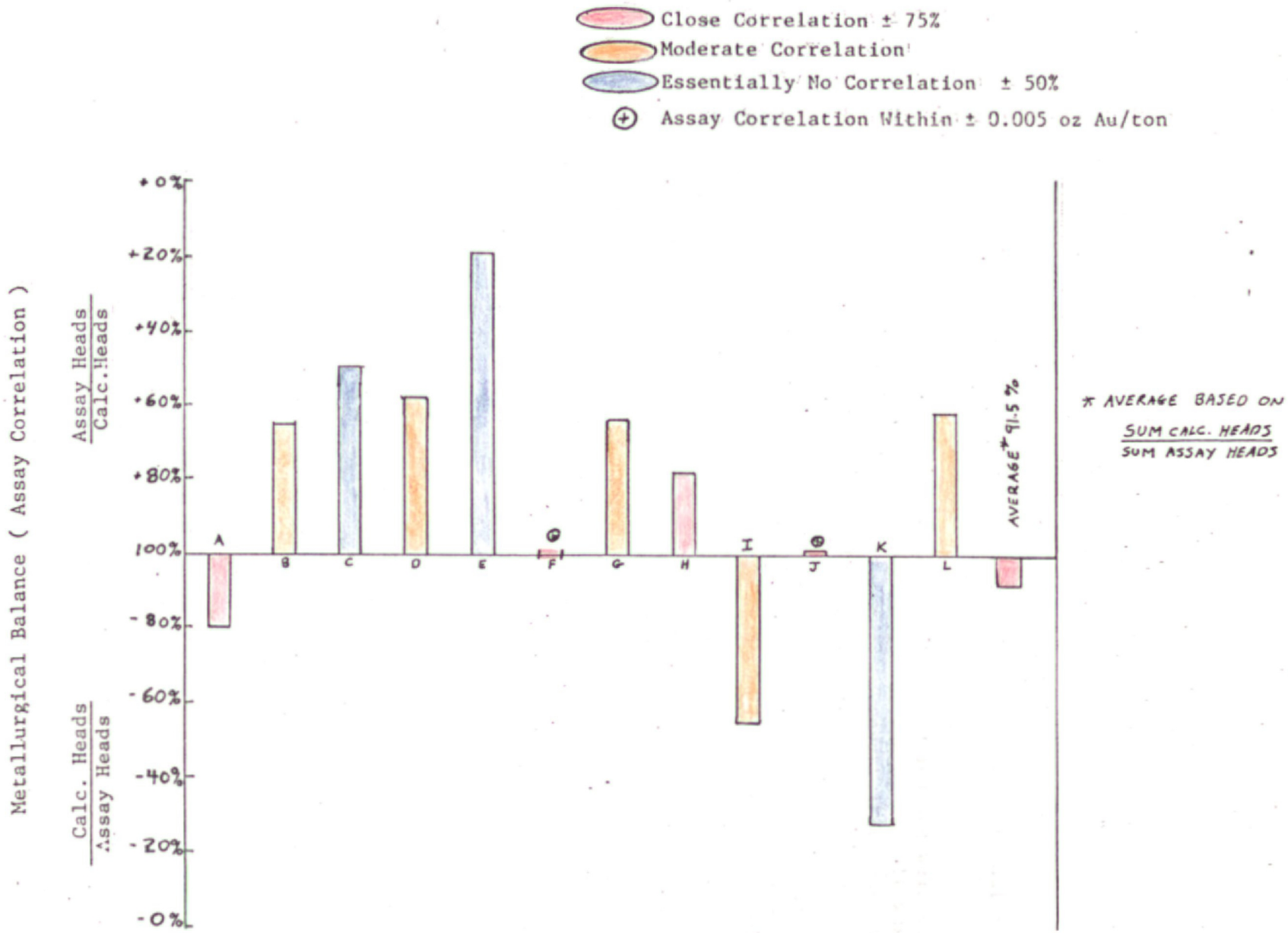


FIGURE 7. GILT EDGE MINI-BULK SAMPLES
12 BUCKET TESTS ON -5/8" ORE
METALLURGICAL BALANCES

ASSAYING PROCEDURES

Heads and Tailings Assays. Heads and tailings assays were all run as half assay ton fire assays.

Carbon Assays. The loaded activated carbon was dried and weighed. One half the carbon was saved for reference and the other half was assayed. The carbon for assay was roasted to convert it to ash, then conventionally fire assayed.

Solution Assays. Approximate solution assays were made periodically on an atomic absorption spectophotometer, using as a standard, a gold cyanide solution which had been calibrated by fire assay. The solution assays were used merely to check on the progress of the leach, since actual recovery was based on fire assay of the activated carbon.

Final solution was checked by AA methods and found to contain negligible amounts of gold.

Submitted by,

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DWK/df