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ALTERATION AND MINERALIZATION OF THE PALEOZOIC SEDIMENTS IN THE EL TIRO PIT AREA SILVER BELL, ARIZONA

A talk presented at the Mining Geology Division of the Arizona Section of A.I.M.E.

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by
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INTRODUCTION

This morning I'll attempt to briefly describe the geology of the sediments and the interrelated igneous rocks in the El Tiro pit area as it is presently known. Much of the data presented is based on a private report to the American Smelting and Refining Company by Mr. Stephen Von Fay. The stratigraphic interpretation of the Union Ridge area has been taken from a recent U of A masters thesis by Joy Merz. My knowledge of the area has been enhanced by discussions and field work with my co-worker Nick Nuttycombe, Mr. Harold Courtright, Mr. Barry Watson, and Mr. Charles Haynes. The views expressed this morning are, however, the responsibility of the speaker.

I'd like to thank the American Smelting and Refining Company for allowing me to present the talk. Speical thanks goes to the Silver Bell Unit of ASARCO, and Mr. D.R. Jameson, Superintendent of the Unit for making this meeting possible.

HISTORY

History of mining in the Silver Bell district possibly dates back as far as Precolumbian times when Indians in the area dug shallow trenches along fracture zones and andesite dikes searching for turquois, hematite and clay. Large saguaro cacti growing in some of these workings indicate they are at least 200 years old.

The area first came to the attention of the white man in 1865 when prospectors located a small pod of enriched silver ore in the altered sedimentary rocks near the present southeast corner of El Tiro pit. Early efforts to mine silver in this area failed, probably due to the fact that Silver Bell has only weak sporadic occurrences of silver mineralization...

Well organized mining activity began in 1899 with the organization of the Silver Bell Copper Company, which evolved into the Imperial Copper Company in 1903. The Southern Arizona Smelting Company or SASCO, built a smelter near the northern end of the Silver Bell mountains giving rise to the small smelter town of Sasco, the ruins of which are still visable today. The Arizona Southern Railway hauled ore between the mines at Silver Bell and the Sasco smelter, both the railway and the smelter being subsidiaries of the Imperial Copper Company.

By 1910 the Company or its predecessors had done some 20.8 miles of underground work in removing about 700,000 tons of ore averaging approximately 3.7% copper. Yet even by accounting all profits including those from the smelter, the railroad, and the company store back to the mine, the company only showed a profit one year. Production ceased in 1910 with the company going into receivership, and later purchase by the American Smelting and Refining Company in 1917.

ASARCO operated the mine on a marginal basis until 1919 when all large scale mining was haulted, sporadic small scale production continuing until 1930. Although the possibilities of disseminated enriched chalcocite ore was recognized in 1909, an extensive drilling program indicated that the tenor was then subeconomic.

In the late forties ASARCO undertook a program of geologic mapping and exploratory and check drilling under the direction of Kenyon Richard and Harold Courtright. This resulted in the delineation of the Oxide and El Tiro enriched chalcocite ore bodies, production from which began in 1952.

The sedimentary rocks which had accounted for the early day production from the district were largely ignored during the first part of the open pit development. This was mainly due to the apparently small amount of mineralized sediments available for open pit operations. In addition, the early miners being interested only in the high grade pods of chalcopyrite, considered anything below 3% copper as waste rock, giving the erroneous impression on old maps that the sediments contained widely scattered pods of high grade material in barren rock.

During a search through old records late in the 1950's, Mr. Steve Von Fay found assay records indicating areas designated as waste ran up to 2% copper. Also found was an old drill log showing mineralized sediments lying below poorly mineralized dacite between the edge of the pit and the outcropping mineralized sediments to the east. A detailed surface and subsurface mapping and sampling program was undertaken. This program was completed with favorable results and a drilling program was initiated. After three years of close spaced drilling, an ore zone of primary chalcopyrite, mineable for open pit methods was delineated.

GENERAL GEOLOGY OF THE EL TIRO PIT AREA

The El Tiro pit exposes all rocks typical of the alteration zone except for the syenodrorite porphyry which is the early phase of the monzonite porphyry. The main geologic features in the pit area are the mafic-free alaskite on the southwest and the dacite intruding Paleozoic sediments to the northeast. These two features are generally separated by the northwest trending El Tiro fault, a wide breccia zone which cuts all Laramide rocks in the pit, and is probably closely related to Richard and Courtrights ancestral "Major Structure". The fault zone appears to have been an important conduit for hydrothermal fluids during the period of mineralization, massive sulfides or strong alteration effects occurring throughout its length. Post mineral movement on the fault is indicated by brecciated and slickensided sulfides.

Two other northwest trending faults indicating recurrent movement along the northwest zone of weakness are defined by post-mineral andesite dikes. These dikes are Mid-Tertiary in age, similar ones being seen throughout length of the alteration zone.

The sediments to the east of the pit are intruded by the dacite in a sill-like manner. Drill hole data indicates the outcropping sediments are floored by the dacite, while the dacite just to the northeast of the El Tiro fault is underlain by sedimentary blocks. Several of these blocks are exposed in the southeastern corner of the pit and along the El Tiro fault zone, and more will be exposed as the pit is deepened.

Two stocks of monzonite porphyry are exposed in the pit. The stock to the southwest of the El Tiro fault is surrounded by alaskite, while the stock to the northeast of the fault intrudes the dacite containing large blocks of sediments and it is essentially bounded on the east, south and southwest sides by these sediments. The two stocks are connected by easterly trending dikes, and are undoubtedly connected in depth to a parent monzonite pluton. These two bodies, inspite of their genetic relationship, are quite differently altered and mineralized. The stock to the southwest of the El Tiro fault is strongly altered showing moderate to strong clay, sericite, secondary potassium feldspar, and silicification along with disseminated pyrite, chalcopyrite and molybdenite and an enriched chalcocite blanket. Moving easterly across the El Tiro fault the clay-sericite alteration becomes perceptably weaker in the monzonite dikes the stock itself being only weakly argillized with sparse to trace amounts of pyrite giving rise to scant limonite staining. There is no chalcocite enrichment, and the rock contains only trace amounts of primary copper. Easterly trending dikes radiating out from this stock, which cut the sediments, are occasionally so weakly altered that twinning of the plagioclase can be seen on a freshly broken surface. This lack of alteration of the monzonite I feel to be due to the fact that it is surrounded by reactive sediments. I theorize that the hydrothermal fluids that so strongly effected the monzonite stock to the southwest of the El Tiro fault were completely absorbed by and reacted with the limy sediments surrounding the stock to the northeast of the fault, leaving that stock almost entirely uneffected by hydrothermal alteration.

The presence of the sediments had a similar effect on the dacite, so that poorly altered and mineralized dacite may be underlain by sediments containing primary chalcopyrite ore.

ALTERATION AND MINERALIZATION OF THE SEDIMENTS IN THE EL TIRO PIT AREA

Although the sediments have been intruded by three different Laramide igneous rocks, namely the alaskite, the dacite and the monzonite, the effects of these intrusions are hard to evaluate. The dacite was relatively cool and gas charged at the time of intrusion and had apparently little metamorphic effect on the sediments. The alaskite and monzonite intrusions may have marbleized the limes and hornfelzed the shaley rocks to some extent. These effects are almost entirely obliterated, however, by the later intense effects which accompanied the hydrothermal mineralization.

In the igneous rocks to the southwest of the El Tiro fault innumerable parallel, easterly trending fractures localized and formed the plumbing for the hydrothermal solutions. This easterly direction of tension fractures also controlled the earlier emplacement of monzonite dike swarms in the El Tiro Oxide pit areas. This same easterly trending tension fracture direction appears to be the structural control for emplacement of ore fluids into the sedimentary rocks.

The ore fluids which permeated the sediments along the easterly trending plumbing system carried silica, iron, sulfur, possibly some potassium, aluminum, and magnesium, along with copper, zinc, molybdenum and minor lead and silver, these last two elements possibly being late stage or even Mid-Tertiary in age.

The hydrothermal fluids altered the sediments to quartzite, marble, hornfels and tactite. The terms quartzite and marble are used in the normal sense and indicate an indurated and silicified sandstone and a recrystalized probably originally rather pure limestone. The terms hornfels and tactite are more ambiguous and have been defined for use at Silver Bell as follows:

> Hornfels is a fine textured rock consisting of varying proportions of lime-silicates such as diopside, epidote, chlorite, feldspar and quartz along with occasional garnet. It is derived from shales, from thinbedded argillaceous limestones and from mudstones and siltstones.
>
> Tactite is a medium textured rock composed of a number of lime-silicate minerals with predominate garnet. It is usually considered to be derived from impure limestones.

In a broad sense the hornfels and tactite units reflect different stratigraphic units, however, locally they are intermingled within individual horizons.

Generally speaking the impure limes and limy argillaceous rocks appear to be more receptative to metasomatism and are therefore good hosts for mineralization.

Garnet (low iron-$CaAl_2(SiO_4)_3$ high iron-$CaFe_2(SiO_4)_3$) diopside, $CaMg(SiO_3)_2$ tremolite-actinolite $Ca(MgFe^{2+})$ $Si_8O_{22}(OH,F)_2$ wollastonite $CaSiO_3$, chlorite and hydrobiotite are the most important gangue minerals.

Garnet in the ore bearing areas is usually brown in color. Chemical analyses indicate it is composed of equal parts of calcium bearing grossularite garnet and iron rich andradite garnet. This appears to represent iron metasomatism and is contrasted with the greenish garnet, probably mainly calcium rich grossularite, which isn't usually associated with ore grade mineralization.

Diopside may represent silica metasomatism of dolometic limes, or merely recrystalization, with the aid of hydrothermal solutions, where sufficient silica as sand was already present.

Wollastonite represents silica metasomatism of limestone. In wollastonite hornfelses it probably represents reaction of the lime and silica already present, assisted by the presence of the hydrothermal fluids.

Increasingly large amounts of metamorphic rock composed mainly of hydrobiotite and chlorite are being found in the pit. This is particularly true along the El Tiro fault and other major hydrothermal channelways, where crystals of hydrobiotite up to two inches across can occasionally be seen. In some places the massive mica-rock is richly intergrown with chalcopyrite. The presence of biotite or chlorite appears to be a fairly good ore guide. The alteration is thought to represent iron, potassium and possibly aluminum metasomatism in a water rich environment.

Chalcopyrite is the only important sulfide copper mineral in the sediments. In areas in the orebody, sphalerite is closely associated with chalcopyrite. In some cases the chalcopyrite is found in solid solution with the sphalerite with exsolution of the chalcopyrite from the sphalerite being clearly seen under the microscope. As no zinc is recovered, where the two minerals are found in solid solution, the sphalerite acts as a diluent because no clean separation can be made between the two minerals in the mill.

Molybdenite occurs in the sediments as disseminations, as "paint" along fractures and with quartz veins in about the same quantity as occurs in the igneous rocks.

Iron as hematite and magnetite, magnetite being most important, appears to be closely associated with the copper mineralization. The occurrence of magnetite varies from disseminated grains to massive pods of the mineral cut by veins of chalcopyrite. This close association with chalcopyrite has made the magnetometer a very useful tool in the search for ore.

Pyrite, though occuring ubiquitously through the metasediments in minor amounts, appears spatially segregated from the chalcopyrite when it occurs in massive form. Drill holes have encountered as much as 100 feet of massive intergrown pyrite and magnetite.

Small occurrences of galena, sometimes quite argentiferous, have been seen, but these are not economically significant. They are either late mineral or may be associated with the Mid-Tertiary activity.

As mentioned previously, the introduction of the hydrothermal fluids was accomplished along easterly trending structures. Most of the disseminated chalcopyrite is associated with small quartz veins, with chalcopyrite disseminating out from the central conduit.

All graduations, from mineralization confined to the vein, to blebs of chalcopyrite disseminated through the rock but associated with vein conduits, to more intense mineralization where most of the gangue minerals have been replaced by sulfides, can be seen in the ore body. Even the massive replacement type mineralization appears to be associated with easterly trending structures, though this relationship is sometimes obscure.

Argillaceous or dirty limestone horizons appear to be particularly favorable for the deposition of ore, where these occur along an ore fluid conduit. Pure limes and of course quartzite horizons are unfavorable for ore deposition, however, along fissures marble will be metasomatized to garnet and carry some chalcopyrite.

OXIDATION OF THE ORE-BEARING SEDIMENTS AND THE OXIDIZED OUTCROP AS A GUIDE TO ORE

The mineralized sediments which outcrop have been oxidized to varying degrees. That is, oxidation may extend to a depth of inches or fractions of an inch or to a hundred feet or more, dependent mainly on permeability due to faulting and fracturing. In most cases, sulfides may be found close to the surface, at least locally.

Because of the high carbonate content of even the most intensely altered rocks, the copper from oxidizing sulfides is almost immediately precipitated in the form of copper carbonates, and therefore, there is no enriched chalcocite blanket. This is important in that copper values found at or near the surface are indicative of the values to be expected in the sulfide zone, providing there is no change in the chemical favorability of the rock.

Important oxide minerals are the black copper oxides, tenorite, and melaconite, the brown amorphous iron-copper complex known as copper pitch, and the copper carbonates malachite and azurite.

Chalcopyrite undergoing oxidation usually first alters to the brown copper pitch. This further reacts with the surrounding carbonate to form malachite. In many cases all that remains of the original chalcopyrite is copper pitch and some malachite, although the copper content of the rock does not change with the oxidation.

The oxidized outcrops of the orebearing metasediments are usually black and lava-like in appearance, with white quartz veins which were the conduits for the mineralizing solutions standing in relief. In many cases copper minerals are not visable except in protected crevices or fissures. The dark coloration is due to manganese and hematite, probably derived mainly from the oxidation of sulfides, but possibly due in part to hydrothermal hematite, and breakdown of iron rich garnet.

On breaking into the rock copper pitch, malachite and possibly even remanent chalcopyrite may be seen.

PALEOZOIC STRATIGRAPHY IN THE UNION RIDGE AREA

Because of the complex structure and intense alteration of the sediments in the El Tiro area, all attempts to work out their stratigraphy by early workers proved fruitless.

The intensity of the alteration is exemplified by the fact that the early workers thought that the Bolsa quartzite, which crops out to the north of the pit, was merely intensely silicified igneous rock.

The stratigraphy was worked out by Mr. Joy Merz in 1967, as his Masters thesis problem at the U of A.

Using the idea that specific sedimentary rocks types when altered would yield specific metamorphic equivalents, and working from the numerous drill logs from holes in the area, Mr. Merz was able to piece together a logical stratigraphic sequence.

His work shows that the pure monomineralic rocks such as quartzite and pure limestone were poor hosts for metasomatism and copper mineralization, the best host rocks being impure limestone and limy siltstones.

The stratigraphic sequence in the Union Ridge area is the Lower Paleozoic sequence of the Bolsa quartzite through the Escabrosa limestone. Because of their impure thinbedded argillaceous nature, the lower Abrigo, the Upper Abrigo, and the Lower Martin limestones are the best hosts for ore, while the Bolsa quartzite, the more pure Middle Abrigo, and the Escabrosa limestone are relatively unmineralized.

SUMMARY

In summary, the Lower Paleozoic sediments in the El Tiro area have been altered and mineralized by the same hydrothermal fluids that altered and mineralized the Laramide igneous rocks.

Although easterly trending structural conduits were important, the mineralizing solutions preferentially metasomatized and mineralized thinbedded units of the Lower and Upper Abrigo formation and the lower part of the Martin limestone. The mineralized sediments form an orebody amenable to open pit mining.

This alteration and mineralization appears typical of limy sediments which are adjacent to bodies of copper bearing porphyrys, similar examples being seen at Santa Rita, New Mexico; Ely, Nevada; the Pima district sout of Tucson, and elsewhere.

UPDATING THE GEOLOGY AND STRUCTURAL ORE CONTROLS
AT SILVER BELL, ARIZONA

by Barry N. Watson
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A talk to be presented to the Mining Geology Division of the Arizona Section of A.I.M.E. on May 20, 1968.

One of the more complete stratigraphic sections in southern Arizona can be pieced together in the Silver Bell area. Much of the geology has been worked out by ASARCO geologists, while a few important areas have been mapped by students as thesis problems. Other portion s of the Silver Bell area have yet to be mapped in any kind of detail, and some of this yet-uncharted geology could well be critical to a better understanding of the complex Mesozoic and Cenozoic stratigraphy.

It is my strong belief that a knowledge of certain of the stratigraphic units in the Silver Bell area--their lithologic characters and structural settings--would be of considerable help to field geologists dealing with similar phenomena elsewhere in southern Arizona. Parts of the Silver Bell stratigraphic section are accessible only by washes or somewhat obscure truck trails, and other portions of the section are on, or readily reachable only by passage through, private property owned by ASARCO.

In the following, I will attempt to briefly describe the geologic history of the Silver Bell area, with particular emphasis on the Mesozoic Era. My knowledge of the area has been greatly enhanced through field excursions and conversations with Harold Courtright, Kenyon Richard, Jim Briscoe, Craig Clarke, Chuck Haynes, Nick Nuttycombe, Joy Merz, Fred Graybeal and Dr. Willard Lacy. I must take, however, the responsibility for the interpretations drawn herein.

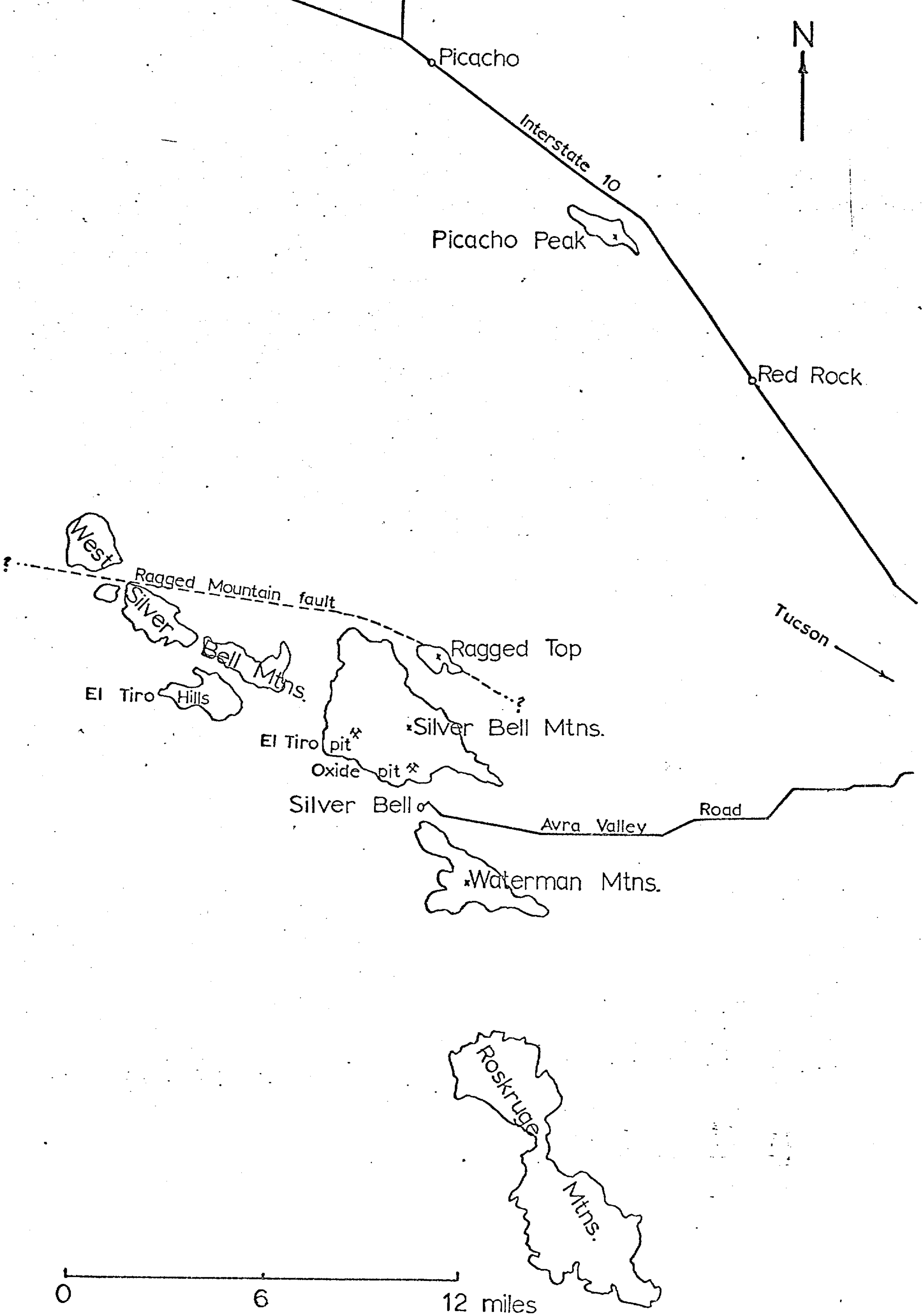
Figure 1 is a location map showing the principal topographic features mentioned below. Figure 2 is my diagrammatic representation of the Silver Bell stratigraphic column.

PRECAMBRIAN

Pinal Schist

The only outcrop of the basement Pinal Schist known to the author in the Silver Bell vicinity straddles the El Paso Natural Gas pipeline road about two miles east of Ragged Top. Relationships with other rock units are obscured by cover, except on the south where the schist is bounded by a mid-Tertiary dike filling the major WNW-trending Ragged Mountain fault.

FIGURE 1. Location map.



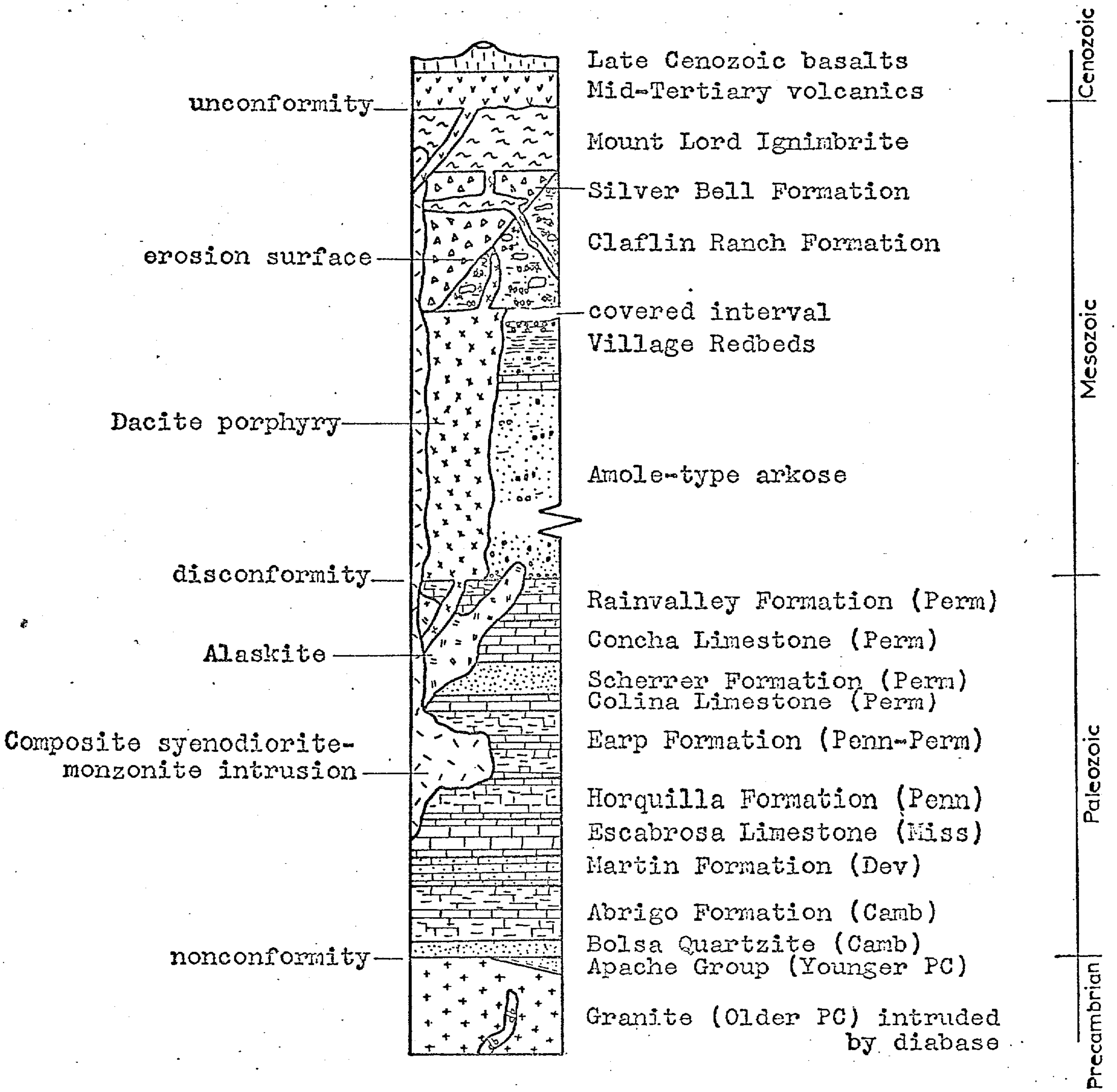


FIGURE 2. Diagrammatic geologic column of the Silver Bell area. Maximum known thicknesses for Paleozoic and Mesozoic rocks are shown. Scale of column: 1"=2000'.

Many fragments (ranging up to boulder size) of Pinal-like schist are seen in Cretaceous sediments just south of Ragged Top, indicating the presence of a considerable area of that schist at the surface in the near vicinity during the Laramide igneous activity.

Granite

A coarse-grained granite is found extensively to the north of the Ragged Mountain fault. Large and numerous quartz grains--frequently .25 inch in diameter--are set among pinkish crystals of feldspar and clumps and books of biotite. In many places orthoclase porphyroblasts up to an inch in length are common. This granite megascopically resembles the Precambrian Oracle granite seen near the town of Oracle.

Paleozoic sediments in the Waterman Mountains southeast of Silver Bell are also underlain by porphyroblastic granite.

Apache Group

Younger Precambrian Apache Group metasediments lie on granite just northeast of Ragged Top. Locally more than 200 feet thick, these south-dipping beds are sharply cut off to the south by the Ragged Top intrusive which wells up along the Ragged Mountain fault. The Apache Group stratigraphy here is not well worked out, but it appears as if a few tens of feet of probable Pioneer Formation (mixed sandy and shaly beds) are overlain by 2-3 feet of Barnes Conglomerate, which is in turn overlain by thin-to moderately thick-bedded quartzites of the Dripping Springs Quartzite.

Apache Group metasediments are missing in the Waterman Mountains where McClymonds (1957) notes Cambrian Bolsa Quartzite to conformably overlie basement granite.

Diabase

Well-altered diabase of possible Precambrian age irregularly intrudes the granite on the northern slopes of Ragged Top. As it is found only within granite, it's relative age cannot be stated with certainty. The principal period of Precambrian diabase intrusion in southern and central Arizona is post-Apache Group.

PALEOZOIC ERA

The Paleozoic stratigraphy of the Waterman Mountains has been deciphered by McClymonds (1957) and Ruff (1951) who mapped a well-faulted pile of limestones, quartzites, siltstones, and shales amounting to a thickness of 4,400+ feet. In the Silver Bell Mountains, Paleozoic stratigraphy was unravelled by Kingsbury, Entwistle and Schmitt in 1941 in a private report to the American Smelting and Refining Co. Merz (1967) undertook the difficult study of the altered and mineralized Paleozoic sediments on Union Ridge east of ASARCO's El Tiro pit. The alteration and mineralization of these Union Ridge sediments will be described in the next paper this morning.

The Paleozoic section in the Silver Bell Mountains is well faulted, locally intensely altered, and generally inundated by various Laramide intrusive units. Although each of the Paleozoic periods represented in the Waterman Mountains also show in the Silver Bell range, the section in the latter is obviously incomplete. A brief tabulation of units with thickness estimates is presented below:

Permian quartzites, limestones, shales..	550 ft. approx.
Pennsylvanian Horquilla Limestone.......	220 ft. max.
Mississippian Escabrosa Limestone.......	275 ft. max.
Devonian Martin Formation...............	300 ft. max.
Cambrian Abrigo Formation...............	430 ft. max.
Cambrian Bolsa Quartzite................	230 ft. min.
Total.....	2,005± ft.

In the El Tiro Hills section of the West Silver Bell Mountains, Clarke (1965) mapped 1,200+ feet of uppermost Permian sediments. Approximately 300 feet of quartzites and dolomitic limestones belonging to the Scherrer Formation are overlain by ±420 feet of Concha Formation Limestone and ±550 feet of Rain-valley Formation limestone and argillite. These Permian rocks protrude from alluvial cover and are overlain by Mesozoic sediments.

MESOZOIC ERA

Amole-type arkose

A clearly exposed contact between Mesozoic and Paleozoic sediments is found in the El Tiro Hills where Clarke (1965) has mapped an estimated 5,000+ feet of probable Cretaceous Amole-type sediments overlying Permian Rainvalley rocks. The basal Amole-type units, lying on a disconformity, is a massive arkosic conglomerate containing rounded quartzite cobbles up to several inches in diameter. This unit of the Cretaceous (?) is several feet thick; the remainder is generally more thinly bedded.

Hayes and Drewes (1968) consider the Amole Arkose of the Tucson Mountains to be more or less a time-equivalent of the lower Middle Cretaceous Bisbee Group sediments. If the Amole-type materials in the El Tiro Hills can be considered correlative with the Amole Arkose, then Clarke's basal quartzite pebble conglomerate qualifies as a far-western equivalent of the basal Bisbee Glance Conglomerate. The presence of Cretaceous (?) beds lying disconformably on the uppermost Permian Rainvalley certainly suggests that the Silver Bell area did not experience, at least locally, the degree of structural unrest manifested farther to the east.

Another interpretation suggested by the near-conformable nature of the Paleozoic-Mesozoic contact related to recent U.S. Geological Survey recognition of Triassic sediments in southern Arizona. Possibly the hiatus between Permian and Mesozoic deposition is not as great as might be thought, and the lowermost Amole-type sediments are of Triassic age?

A few tuffaceous beds are scattered through the Amole-type arkoses, indicating periodic volcanic activity in the general region. Red-colored shales and conglomerates are found here and there through the sequence and are most prevalent in the upper portions. A 20-30-foot thick sandy limestone occurs near the top of the exposed older Cretaceous beds.

The Amole-type sediments are overlain in angular unconformity by interbedded tuffs and coarse clastic sediments of the Claflin Ranch-type. A similar mid-to late Cretaceous unconformity has been noted elsewhere across southeastern Arizona. It is felt that this unconformity reflects initial upheaval related to Laramide deformation.

Amole-type arkoses, conglomerates and sandstones also crop out in the valley between the Waterman and Silver Bell Mountains. Immediately overlying the arkoses near the southeast corner of the older Silver Bell tailings dam is a limestone unit probably exceeding 200 feet in thickness. Donald Bryant of the University of Arizona was able to identify recrystalized pelecypods here as of definite Cretaceous age. Outside of the Bisbee Group Mural Limestone, this localized unit is probably the thickest Cretaceous limestone known in southcentral Arizona.

Village Redbeds and red conglomerates

A sequence of red-colored clastics is found overlying the limestone unit and Amole-type arkoses south of the Silver Bell tailings dams. These clastics, which also underlie Silver Bell village, are locally several hundreds of feet thick, but faulting and alluvial cover prevent thickness determinations. The author originally considered this unit to be an equivalent of the Recreation Redbeds of the Tucson Mountains. However, detailed mapping plus radiometric age-dating have recently proven the Recreation Redbeds to be of pre-Amole age, and evidence is now overwhelming that red coloration represents restricted environmental conditions that could, and do, appear at various times throughout the Mesozoic. Consequently, I am here designating the Cretaceous redbeds and red conglomerates near the Silver Bell townsite the "Village Redbeds".

In places redbeds and light-colored Amole-type arkoses are found interbedded, suggesting a somewhat gradual transition from the Amole to the Village environment. Several hundred feet of red silts, sands and arkoses occur in the lower portions of the Village Redbeds and are seen to grade upward to red conglomerates. At first these conglomerates contain only sedimentary detritus. Higher in the sequence igneous materials begin to appear, however, and in the uppermost known portions the red conglomerate consists almost entirely of purple andesitic fragments set in a detrital matrix. Deformation of an ancient Silver Bell landscape and a gradual increase in volcanic activity is readily evidenced in the continuing deposition of the redbeds and red conglomerates. Thus the transition from normal Cretaceous subaerial sedimentation to coarse and rapid Laramide accumulation is not always marked by an obvious stratigraphic break.

The Village red conglomerates are cut off by a major WNW-trending fault in the tailing pond area, and their relation to overlying units is not presently known.

Claflin Ranch Formation

The Claflin Ranch Formation is something of a catch-all term, and the rocks it represents are not limited to any one specific time of deposition. The formation represents a type of sedimentation associated with a terrane undergoing volcanic upheaval and rapid erosional deformation. Thus, in the Silver Bell Mountains where Richard and Courtright first used the name (1960), the conglomerates, mudflows, landslide blocks, aeolian tuffs, water-lain tuffs and pyroclastic layers included within the Claflin Ranch Formation have ambiguous relationships with associated volcanic units. They are pre-dacite and post-dacite, pre-Silver Bell andesite and post-Silver Bell andesite. In the West Silver Bell Mountains Claflin-like conglomerates are interbedded with pyroclastics and overlie earlier Cretaceous sediments by angular unconformity.

The thickest continuous Claflin Ranch sequence in the Silver Bell Mountains--approximately 1800 feet--occurs southwest of Ragged Top. This accumulation is, at least in good part, pre-dacite porphyry (the earliest of the Laramide volcanic and sub-volcanic rocks in the Silver Bell range). Coarse, greenish clastic materials megascopically identical with parts of the Claflin Ranch Formation are found as a matrix of the Tucson Mountain Chaos in the Tucson Mountains. Claflin Ranch-type rocks also are seen in roadcuts north of Sonoita along Arizona State Highway 83.

It seems reasonable to expect that the Claflin Ranch-type of surface accumulation of detrital and volcanic debris might be found throughout southern Arizona wherever Laramide volcanic piles exist. Such depositional sequences--seemingly thickest in earlier Laramide time--would run the gamut from fairly thin-bedded sands to chaotic masses of landslide-block accumulations.

Alaskite

Richard and Courtright (1966), in accounting for the WNW-striking zone of alteration at Silver Bell, conclude that "indirect evidence suggests a fault representing a line of profound structural weakness existed in this position prior to the advent of Laramide intrusive activity." This line is referred to as the "major structure." They go on to note that this major structure "was largely obliterated by the Laramide intrusive bodies, but it effected a degree of control on their emplacement, as evidenced by their shapes and positions."

The first indication of activity along the Silver Bell fault zone came in early Laramide time with the intrusion of a coarsely granitoid alaskite along the southwest side of the major structure. This alaskite, which contains a very low ferromagnesian mineral content, intrudes Paleozoic sediments and Cretaceous Amole-type arkoses in the El Tiro area. Aplite dikes are found through the alaskite, and, locally, fine-grained border phases of alaskite are found in contact with other rock units.

The alaskite is one of the principal hosts for the later porphyry copper mineralization. This coarse-grained felsic rock locally shows high chalcopyrite-to-pyrite ratios.

Dacite porphyry

The dacite porphyry is a sub-volcanic rock characterized by numerous rounded or triangular quartz "eyes" set in a very fine-grained matrix. Orthoclase and sanidine phenocrysts, vague but consistent flow structure, and up to 20% of xenoliths are also commonly seen. Chemically, the dacite porphyry is more accurately a quartz latite porphyry.

The dacite occurs extensively northeast of the major structure in the form of sills and dikes within Paleozoic and Mesozoic sediments. The largest body of the porphyry--a sill $\pm$ 3,400 feet thick--occupies the stratigraphic interval in the Silver Bell range proper where Amole-type arkose should occur. This sill is floored by Paleozoic sediments and roofed by an 1800-foot sequence of Claflin Ranch materials. The dacite-Claflin Ranch contact is gradational over several feet, but dikes of dacite porphyry are found locally in the overlying Claflin Ranch beds.

An explosive history for the dacite porphyry is strongly suggested by the numerous xenoliths, the large fragments of quartz, and the shards of former glass in the matrix. The nature of the rock is believed to reflect an emplacement by fluidization in the following manner:

The gas-and fragment-charged dacite porphyry magma (actually quartz latite in composition, suggesting greater viscosity and more explosive potential) rose along the Silver Bell fault zone into Paleozoic strata. The higher the porphyry magma ascended, the more the confining pressure decreased, causing exsolution of gases and thus lending an explosive and dilative nature to the intrusive material.

Its extension to the southwest blocked by the large body of alaskite, the dacite porphyry welled up, sending small dikes and sills northeastward into the Paleozoic beds. Damp Amole-type Cretaceous (?) sediments were reached and more gas evolved. The magmatic material, expanding constantly, spread laterally to the northeast in the weak Cretaceous (?) sediments. Dilation occurred, as did the incorporation of fragments broken by churning gas action.

The dacite porphyry probably surfaced in one or more places, venting gases as it did. Gas also escaped laterally through the just-formed sill and vertically into overlying Claflin Ranch sediments. The heat and vapor action altered the immediately overlying quartzo-felspathic clastic sediments, giving rise to the gradational contact seen today.

The dacite porphyry was a poor host rock for porphyry copper mineralization because of its flinty, "tight" nature.

Silver Bell Formation

The Silver Bell Formation (Richard and Courtright, 1960) consists of laharic, autobrecciated, and intrusive andesitic to dacitic breccias, andesitic to dacitic flows, and andesitic intrusions. These materials overlie Claflin Ranch sediments and dacite porphyry in the Silver Bell Mountains. The rugged nature of the basal Silver Bell contact and the fact that it locally lies on unroofed dacite porphyry points to a period of rapid uplift and erosion following intrusion of the dacite porphyry sills.

Purplish Silver Bell-type breccias are seen to be interlayered in places with overlying Mount Lord Ignimbrite. Such a transition from andesitic activity to more felsic and explosive volcanism is seen throughout the world and is commonplace in the Laramide rocks of southern Arizona and southwestern New Mexico.

It is believed that the Silver Bell Formation is roughly correlative with the Demetrie Formation of the Sierrita Mountains, the Picacho Peak volcanics (Briscoe, 1967), the Owl Head volcanics, and that portion of the Cloudburst Formation north and east of the San Manuel mine.

Mount Lord Ignimbrite

A welded ignimbrite lithologically similar to, and stratigraphically a time-equivalent of, the Cat Mountain Rhyolite of the Tucson Mountains overlies the Silver Bell Formation in the Silver Bell Mountains. This quartz latitic ignimbrite is up to 800 feet thick, including an 80-foot thick cap of lithic vitric tuff. As Silver Bell Peak was formerly known to residents of the area as "Mount Lord" and since the peak is composed of the pyroclastic unit, the name "Mount Lord Ignimbrite" has been given to this Cat Mountain-type unit.

Intrusive ignimbrites--genetically related to the Mount Lord Ignimbrite, and megascopically and petrographically identical with it--occur as dikes and sills in the underlying Silver Bell Formation and dacite porphyry. These feeder materials once en route to the surface spread along bedding and formational contacts, apparently when vents became choked.

The Cat Mountain Rhyolite of the Tucson Mountains evinces an average age of 68 million years (Damon, 1968), and it is felt that the Mount Lord Ignimbrite is of similar age.

Syenodiorite porphyry

The syenodiorite porphyry is an early and somewhat extensive pyroxene-bearing phase of the composite intrusive thought to be related to the copper mineralization at Silver Bell. Later phases of this composite intrusive are monzonitic and quartz monzonitic. The syenodiorite porphyry is found principally in the southeastern portion of the Silver Bell Mountains. It occurs as massive bodies in Oxide pit (where it was previously called both "andesite" and "dacite") and east of Oxide pit along the major structure, and is found as east-trending dikes north of Oxide pit in the mountain range.

The syenodiorite porphyry is the best host rock in Oxide pit. It shows the highest primary copper sulfide content of any of the igenous rocks at Silver Bell and has allowed precipitation of a substantial chalcocite blanket.

Only occasional dikes of syenodiorite porphyry are seen in El Tiro pit.

Monzonite porphyry

The later monzonitic and quartz monzonitic phases of the composite intrusion are found as massive bodies scattered along the major structure. They occur also as generally east-trending dikes in the mountain range to the northeast of the major structure.

The principal porphyry copper mineralization followed emplacement of the monzonite porphyry, and a zone of alteration was superimposed on the major structure. K-Ar age-dating (Mauger, Damon and Giletti, 1965) has shown that the solidification of the monzonite porphyry and the subsequent hydrothermal alteration occurred at approximately 65 million years and within a short enough time span so that, considering the limits of error of the age-dates, the two events are radiometrically indistinguishable. I do not mean to imply here that the Silver Bell deposits are to any great extent syngenetic as has been suggested recently (Mauger, 1966). It may be that a small amount of chalcopyrite became trapped as discrete grains in the monzonite magma at the time of solidification. The great preponderance of copper mineralization, however, was emplaced in the various host rocks through veins, veinlets, and hairline fractures with values diffusing into wallrocks, possibly with the aid of a certain amount of igneous rock recrystallization.

It is interesting to note that both the Oxide and El Tiro orebodies occur at structural intersections (see Figure 3). Oxide pit is located at the junction of the WNW-trending major structure with an ENE-trending swarm of syenodiorite and monzonite porphyry dikes. Similarly, El Tiro pit exists at the junction of the major structure with a northeast-trending swarm of monzonite porphyry dikes.

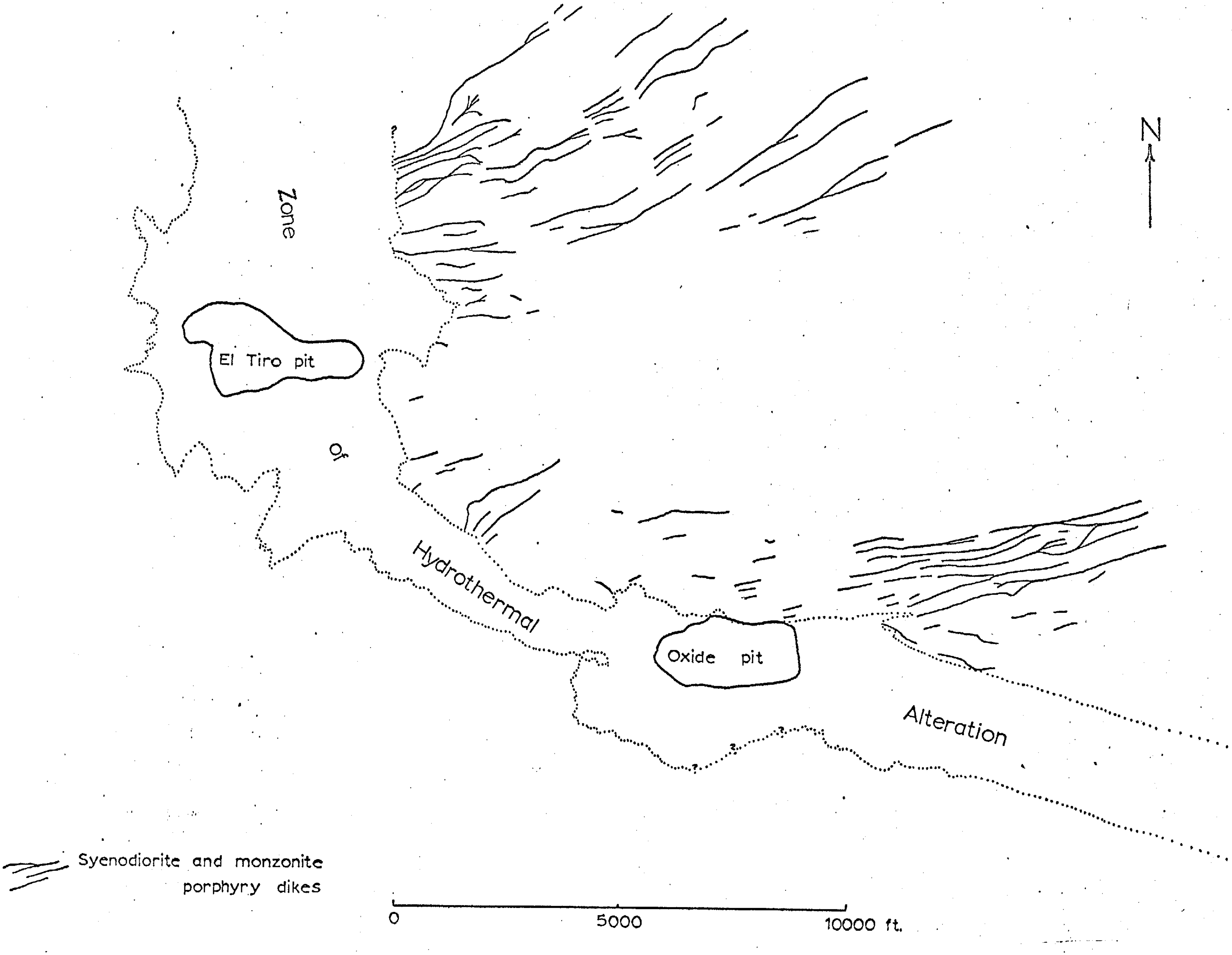


FIGURE 3. Dike swarms related to mineralization at Silver Bell

B. Watson

CENOZOIC ERA

It is preferred here to set the Mesozoic-Cenozoic time boundary at 63 million years as defined by Folinsbee, Baadsgaard, and Lipson (1961). This allows the Silver Bell mineralization to fall at the end of the Cretaceous Period.

Regional northeasterly tilting of 20°-30° occurred sometime between the emplacement of the composite Laramide intrusion and the mid-Tertiary volcanism. It probably was a result of late Laramide upheaval. This tilting, shown by the present orientation of Laramide depositional units, appears to have taken place by rotation of WNW-elongate, fault-bounded blocks in the Silver Bell area.

The mineralized rocks at Silver Bell were exposed to weathering and probably supergene enrichment in early Tertiary time. This is strongly suggested 3 miles east of Oxide pit where pieces of leached capping were found in a conglomerate immediately underlying an andesite flow dated at 28 million years (Damon and Mauger, 1966). A mid-Tertiary period of rhyolitic to andesitic volcanism evinced widely over southern Arizona probably covered and thus preserved the Silver Bell mineralization. This mineralization has been exhumed in more recent times and is presently undergoing destruction through weathering processes.

North-northwest-trending quartz latite porphyry and andesite porphyry dikes of the mid-Tertiary volcanic epoch cut all earlier rock units in the Silver Bell Mountains. The quartz latite dikes have a strangely discontinuous line of outcrop which is caused not by faulting, as has been previously suggested by Schmitt (1941), but by intrusion into a very broken and faulted terrane. A few of the andesite porphyry dikes are conspicuous in El Tiro pit where they are locally collectors of green copper oxide.

The Ragged Top Latite Porphyry dated at 25±1.0 million years (Mauger, Damon and Giletti, 1965) intruded the prominent Ragged Mountain fault which had dropped Laramide rocks on the south some 5,000-7,000 feet against Precambrian granite. Andesitic and rhyolitic flows of probably similar age are seen several miles west of Ragged Top in the northeastern part of the West Silver Bell Mountains.

A late and minor lead-silver-copper mineralization is found in the Silver Bell range. North-trending epithermal veins carrying galena, native silver and cerargyrite with a barite-quartz-calcite-fluorite gangue were mined in the early days. Copper stain is seen on the old dumps. This later period of mineralization has been superimposed very locally on the porphyry copper deposits to the south. On the other hand, a mid-Tertiary quartz latite porphyry dike cuts one of the epithermal veins, thus establishing a general minimum date to this mineralization.

Quaternary-Tertiary basalt cones and flows are found north of the Ragged Mountain fault.

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COPPER DUMP LEACHING AT ASARCO'S SILVER BELL UNIT, ARIZONA

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Dump leaching at Silver Bell started in January, 1960. The dumps now being leached are the results of selective mining during the stripping and active mining phases of the development of the two pits. Barren cap-rock is segregated and dumped separately in waste dumps. The copper bearing material in the leach dumps consists of oxide copper minerals and low grade sulfide copper minerals. Neither of these two classes of material could be profitably treated by flotation in the concentrator. The copper values are amenable to dump leaching in closed circuit with iron launder precipitation.

The more basic criteria for dump leaching are: (a) copper mineralization capable of dissolution in leaching solutions within reasonable lengths of time; (b) a host rock which will not consume inordinate quantities of acid, or decrepitate to prevent proper passage of solutions; and (c) a suitable site for placement of the dumps to insure minimal losses of pregnant solution to seepage and good drainage to a central recovery dam.

Additional advantages which are desirable, but not basically necessary are: (a) sufficient pyrite present in the dump material to generate enough free acid and ferric sulfate to dissolve the copper minerals without acid having to be added to the leaching solutions; and (b) not too much ferric sulfate produced in the dumps, which would make subsequent precipitation of the copper difficult or costly.

All of the above basic criteria are realized in the dump leaching operations at Silver Bell. However, H_2SO_4 is added to the leach solutions to aid in dissolving the copper minerals.

LEACH DUMPS AND DAMS

At the present time, there are four dumps undergoing leaching, (Figure 1). The original dump, upon which the leaching plant started operations, is in a canyon adjacent to the Oxide Pit. This is now called the Upper Oxide Leach Dump (Ox. II). The ravine underlying this dump runs directly to the main pregnant solution dam near the precipitation cells.

The main pregnant solution dam is of concrete construction, abutting in solid rock on both walls of the canyon. It has a storage capacity of about 750,000 gallons.

The lower Oxide Leach Dump (Ox. I) was started in another ravine west of the Upper Dump (Ox. II). It has been formed mainly with leach material developed by stripping and mining after ore production for the concentrator was started in the Oxide Pit. In fact, leach material is still being added to its north-western end while the rest of the dump is being leached. The pregnant solution collecting in the ravine under the northern two-thirds of this dump is diverted with an earth-fill dam and a 16-inch pipeline some 250 feet long, to the main pregnant solution dam.

The diversion dam has a 12-inch thick concrete key and the earth face is sealed and protected with gunite. The footings of this dam are in conglomerate but there has been very little leakage. The inlet to the diversion pipe is provided with slots for weir boards so the dam can be used as emergency storage of about 100,000 gallons of pregnant solution in case of trouble with the pumps at the main pregnant solution dam.

The southern one-third of Oxide I dump drains to the same canyon as the Ox. II dump and the solution goes directly to the main pregnant solution dam.

About 250 yards below the diversion dam and the pregnant solution dam the two ravines from the Oxide Dumps joins as one (Fig. 2). Below this junction another 50 yards lies the barren solution dam, another earth-fill dam with a tamped-earth key. The footings of the key are in solid rock on one side and conglomerate on the other. The storage capacity of this dam is roughly one and one-half million gallons.

In early 1961, the west leach dumps being prepared adjacent to the El Tiro Pit (E.T. I) were ready for leaching. In order to accomplish this, it was necessary to put in a pump and a pipeline from the barren solution dam to the El Tiro #I dumps, construct a dam across the canyon below, and provide pumps and a pregnant solution return line to the main pregnant solution dam. Also, the additional amount of copper to be precipitated required an increase in the number of cells and in drying area at the precipitation cells. Construction was completed and leaching started on the El Tiro #I dumps in July, 1971.

The El Tiro I dumps overlie four branches of a main canyon which drains the area. These four join under the dumps and there is only a single underflow. The pregnant solution dam is about 500 feet downstream from the toe of the dumps. It is of concrete, tied into solid rock and has a storage capacity of about 100,000 gallons.

In 1965, the El Tiro South Dumps (E.T. II) were large enough to allow starting several rows of ponds on the established area while the crest was continuously being advanced by additional leach material. This required a third barren solution pump installation at the barren solution dam, a pipeline to the dump, a pregnant solution and diversion dam across the drainage canyon, pumps and return lines to the main plant, as well as additional drying pad area at the plant to allow for the anticipated additional production. As is explained later, only modest changes were necessary on the precipitation cells, with no increase in number. The construction work was completed and leaching of the El Tiro II dumps started in December, 1965.

: The concrete pregnant solution dam for the El Tiro II dumps is tied into solid rock and has a storage capacity of about 350,000 gallons.

LEACHING SOLUTION DISTRIBUTION

At the barren solution dam, there are two 6-inch vertical centrifugal pumps of type 304 stainless, driven by 100 HP motors, for pumping the solution to the Oxide area and El Tiro #I. The pump to El Tiro #II is an 8-inch pump, driven by a 150 HP motor. They are floated on rafts to maintain constant submergence regardless of the rise and fall of the water level in the dam. The raft for Oxide and El Tiro #I is made up of a wooden deck floating on 24 sealed ten-foot lengths of 12-inch I.D. PVC plastic pipe. The separate raft for El Tiro #II is floated on polystyrene flotation billets. The pumps are connected to their respective discharge lines by flexible hoses. The rate of the flow of barren solution from each of the pumps is measured and recorded by orifice plate meters.

The Oxide dumps receive their leaching solution through a 10-inch pipeline approximately 3950 feet long, with a static head of 260 feet. On the Upper Ox. II dump, the solution was initially distributed from ten lateral pipes six inches in size. These laterals were provided with one and one-half inch plastic valves on each side of the pipe every 50 to 60 feet. The valves regulated the flow of solution to small, irregular ponds which averaged about 50 to 60 feet square.

When the Ox. #I dump was being readied for leaching, it was decided to try a less elaborate method of distribution. In this system, the solution was simply delivered through an open-end 10-inch pipe to a high point and discharged to an open ditch. From the ditch, the solution was cut into one or more 100' square ponds as desired. This "irrigation system" has proven quite successful and the same method is now in use on all dumps. The barren solution is being distributed at a rate of 1 gal/min. to 200 square feet of dump surface.

El Tiro #I barren solution is delivered through 16,940 feet of 8-inch pipeline with a static head of 165 feet to the upper benches. El Tiro II solution travels two miles through a 14-inch line, against a static head of 260 feet.

The El Tiro #I pregnant solution used to be pumped the full 16,940' back to the main plant in its own separate 8-inch line. Since the installation of the El Tiro II pregnant solution dam, however, the E.T.I Pregnant solution is simply pumped by two 4-inch pumps to the new dam where it joins the underflow of the E.T.II dump. The combined underflows are then pumped by two 6-inch and one 8-inch, Type 316 stainless steel pump through an 8-inch and a 12-inch pipeline to the main pregnant solution dam. Magnetic type flow meters on the two lines measure and record the flow rates.

On all ponds, every effort is made to get the solution to spread out and cover as much area as possible and not short-circuit through the dump. In some rocky areas, it has been necessary to bring in concentrator tailings and crusher fines, to be spread in thin layers over the pond's surface, to reduce the porosity. The whole purpose is to gain as wide a distribution base as possible to get drop-by-drop penetration into the dump. This gives maximum expectation of wetting every rock in the dump, maximum contact time for leaching, minimum of channeling, and above all, a higher grade pregnant solution.

There is a practical limit, however, to these large pond areas, because of the high evaporation rate in the desert country which creates an appreciable loss of water. The percent recovery of leaching solutions varies widely but averages 90 to 95 percent. Most of the loss cam be attributed to evaporation, with minor losses to seepage and seeping into the pores of the rock in the dump.

CHEMICAL REACTIONS OF LEACHING

The chemical reactions involved in leaching of copper minerals were thoroughly studied by John S. Sullivan and others, about 1930. From this source and others, the following overall reactions are given for the dissolution of the principal minerals in Silver Bell dump leaching.

Chrysocolla: $Cu\ SiO_3.2H_2O + H_2SO_4 \rightarrow Cu\ SO_4 + SiO_2 + 3H_2O$

Azurite: $Cu_3(OH)_2(CO_3)_2 + 3H_2SO_4 \rightarrow 3CuSO_4 + 2CO_2 + 4H_2O$

Malachite: $Cu_2(OH)_2CO_3 + 2H_2SO_4 \rightarrow 2CuSO_4 + CO_2 + 3H_2O$

Cuprite: $Cu_2O + H_2SO_4 + Fe_2(SO_4)_3 \rightarrow 2CuSO_4 + 2FeSO_4 + H_2O$

Chalcocite: $Cu_2S + 2Fe_2(SO_4)_3 \rightarrow 2CuSO_4 + 4FeSO_4 + S$

In vat leaching or agitated leaching, chalcopyrite is usually considered to be insoluble in leaching solutions or, at best, has a reaction requiring such an extremely long time that the amount dissolved is negligible. For these methods of leaching this is true. However, in dump leaching, time of reaction is measured in years. It is probable, then, that a small amount of chalcopyrite does dissolve slowly but inexorably over the years due to the action of ferric sulfate, oxygen, and water.

Chalcopyrite: $CuFeS_2 + 2Fe_2(SO_4)_3 + 2H_2O + 3O_2 \rightarrow CuSO_4 + 5FeSO_4 + 2H_2SO_4$

At Silver Bell, extra acid is added to the barren solution when it is pumped to the dumps as leaching solution. Experiments indicated that the extra acid addition helped to keep Iron scale from forming in the pipelines; and the leaching mechanisms previously described. A minor source of sulfuric acid and ferric sulfate required by the above reactions is the reaction of pyrite in the dump with water and oxygen to form these solvents, probably assisted by iron-based bacterial action.

$$\text{Pyrite: } 4FeS_2 + 15O_2 + 2H_2O \rightarrow 2Fe_2(SO_4) + 2H_2SO_4$$

Table I shows typical flow data and analysis of solutions to and from the dumps. Assays are reported in grams per liter.

TABLE I

Dump Leaching Data

Barren Solution to Dumps

		Oxide	El Tiro I	El Tiro II	TOTAL
GPM		700	500	1500	2700
Cu	gms/liter				.030
H_2SO_4	" "				.15 - .30
Fe^{++}	" "				.5 - 1.0
Fe^{+++}	" "				.05
pH	" "				2.70

Pregnant Solution from Dumps

		Oxide	El Tiro I	El Tiro II	TOTAL
GPM		690 - 700	410 - 500	1400	2500 - 2600
Cu	gms/liter	0.6 -.800	1.0 - 1.60	.5 - .700	.6 - .900
H_2SO_4	" "	.40	1.00	.15	.50
Fe^{++}	" "	.01	.01	.01	.01
Fe^{+++}	" "	.15	.06	.08	.06
pH	" "	2.50	2.00	3.05	2.3 - 2.50

As can be seen from the above data, the barren solution consists mainly of a slightly acidic solution of iron salts, most of which is in the ferrous form. In passing through the dumps, the pregnant solution has accumulated copper, gained in acidity, and converted its remaining iron content almost entirely to the ferric state. Some of the original iron content was precipitated in the distributing ponds and some in the dump itself. There has been some indication that the iron precipitated in the dump may have a deleterious effect on the leaching of the dump material in the form of an iron coating which seals off rock from additional leaching. This can be miniumized by periodic working of the dumps and additions of acid to the leach solution.

PREGNANT SOLUTION GRADE CONTROL

The copper content of the pregnant solution underflow is maintained by gradual, progressive changes from one pond to another on the surface of the dumps. Usually, several small ponds are being leached at the same time. As the grade of copper in the underflow tends to fall, a new pond is cut in, and the pond which has been leaching the longest is cut out. The length of time during which a particular pond may be covered by solution varies from a few days up to several weeks. Since this time is dependent only on the copper being extracted, it follows that the depth of the leaching column, the copper content of the rock, the type of mineralization, and the efficiency of the leaching solution distribution are all factors in its determination.

After a pond is cut out of the leaching cycle and the excess solution has drained, the material in the leaching column underneath this pond will remain unwetted until the pond is again cut in for leaching in its turn in the progression from pond to pond to maintain copper grade. The time of this drying or rest period varies from six months to a year at the present rate of operation.

During this rest period, there is still enough moisture in the rock to maintain the humid, oxidizing conditions required by serveral of the chemical reactions to create additional solvents and to dissolve the copper minerals. By diffusion, capillary action, and evaporation, these salts concentrate at the surface of the rocks and are readily dissolved by the leaching solutions during the next wetting cycle.

PRECIPITATION OF COPPER

To precipitate the copper from solution, detinned scrap cans, supplied by Proler Steel Corp., of El Paso, Texas are used. The main advantage of cans is the large surface area presented to the solutions per pound of metal. This large surface area promotes more efficient precipitation per unit of cell volume, than with heavier pieces of iron.

Cans are prepared for use in precipitation by burning in kilns to remove the tin plating and paint from the surface, and the solder from the seams. After this, they are passed through a hammer mill, a toothed roll or some other shredder of suitable design to make them more compact and less bulky to handle, and less wasteful of space in the cells.

Untreated cans will weigh only 8 to 12 pounds per cubic foot. After shredding and compacting, they will weigh 20 to 30 pounds per cubic foot. The limiting factor of the compaction is that if it is carried too far, there will not be enough porosity remaining to get adequate solution penetration and efficient precipitation. Baled cans have never been widely used for this reason.

CHEMICAL REACTIONS OF PRECIPITATION

There are three principal reactions taking place simultaneously in the iron launders. Only one of these is profitable. The other two represent a necessary operating loss to achieve the first; the precipitation of cement copper. The overall reactions are:

$$CuSO_4 + Fe \longrightarrow Cu + FeSO_4$$
$$H_2SO_4 + Fe \longrightarrow H_2\uparrow + FeSO_4$$
$$Fe_2(SO_4)_3 + Fe \longrightarrow 3FeSO_4$$

There is also a fourth reaction, between ferric sulfate and metallic copper, which undoubtedly takes place, but yields the same net effect as the overall reactions given.

$$Fe_2(SO_4)_3 + Cu \longrightarrow CuSO_4 + 2FeSO_4$$

The copper sulfate formed in this reaction is reprecipitated on metallic iron as in the first equation. The overall effect is that at equal concentrations, the reduction of ferric iron to ferrous iron is the fastest of the three basic reactions. The reduction of copper is the next in rapidity, followed by the reaction between acid and iron.

PREGNANT SOLUTION PUMPING TO CELLS

At the main pregnant solution dam there are three five-inch vertical centrifugal pumps of 316 stainless, two are driven by 50 HP motors and the third by a 30 HP motor, to pump the solution to the cells. A magnetic type flow meter in the main header of the three pumps measures and records the rate and quantity of flow. The discharge pipe is 16 inches in diameter, and about 400 feet long from dam to cells.

Sulfuric acid is being added to the pregnant solution ahead of the precipitation cells. The amount of acid added is small; only enough to lower the pH to about 2.3. The purpose of this addition is to gain better copper precipitation conditions in the cells by preventing hydrolysis and precipitation of hydrous iron salts in the lower cells where the acid concentration is low. About half of the acid is being added to the pregnant solution at the El TiroI dam to prevent scale forming in the 3-1/2 miles of pipe which return this pregnant solution and combined El Tiro underflow to the main plant. The other half of the acid is added at the pumps on the feed to the precipitating cells.

Sulfuric acid is stored in a 10,400 gallon tank at the plant's office and in a 4,090 gallon tank on a dump above the El Tiro I pregnant solution dam. Deliveries from the tanks are metered by variable-stroke diaphragm acid pumps. In the warmer months all the acid is 98% H_2SO_4 or 66.4° Baume'. In the Winter months 93% or 66.0° Baume' is used, to prevent the acid from freezing in the lines. In the near future plans are being made to use a 75% H_2SO_4 or 55.1° Baume', which will be more plentiful and cheaper to buy than the 98% or 93% acid.

CEMENTATION CELLS

The original plant consisted of six precipitating cells. This number was increased to ten in 1961 to gain the needed capacity for the precipitation of the El Tiro solutions.

However, an economic analysis of the precipitating cell operations is 1964 showed that not all of these ten cells were needed. The study made clear that the iron consumed in the lower cells was costing more than the copper precipitated there was worth. As a result of this investigation, five of the cells were cut out of operation and only the remaining five were used to precipitate the copper from the 1000 GPM of solution flow.

Then, when the El Tiro II dump was brought into production in late 1965, with nearly another 1000 GPM of solution to be treated, it was only necessary to cut these five cells back into operation, with minor modifications, to have adequate capacity to handle the flow rate. The economic study of 1964, therefore, has paid off in continuing savings of can consumption and in preventing an unnecessary over-capitalization during the plant expansion of 1965. The present flow rates of 2500 - 2600 GPM are at times too much for the 10 cells, especially when the heads are high in copper. Therefore, plans are being made for a needed expansion in 1974, in the form of new precipitation cones in service along with the 10 precipitation cells. Each cell is eight feet wide by five feet deep and is divided into two compartments, each twelve feet long, by a dividing center wall. The tops of the concrete walls are protected by 6" x 10" timbers. At the five-foot depth in the cells is a perforated screen made of 3/4 inch polypropelyne which has been locally drilled with one and a quarter inch holes on one and one-half center. The poly-propelyne screens are supported by type 316 stainless steel grids which have approximately two inch by four inch openings. The grids rest on two 6" x 8" timbers which are keyed into the side walls in each compartment. Beneath this screen bottom, the concrete floor of the cells slopes to a twelve inch drain valve. This valve is operated by a bell crank and handwheel from the walkway on top of the cells. Stainless steel grates with 1" x 2" openings were tested, but allowed too many scrap cans to escape into the final precipitate during washing and caused an increase in can consumption and lowering of the grade of the precipitate.

CELL OPERATION

Feed solution to the cells passes through cells 1, 2, 3 and 4 in parallel flow from the feed launder and then returns, in parallel, back through cells 5, 6, 7 and 8. The advantage of parallel flow on these first cells is that the heavy precipitation from the strong solutions is divided and there is less back-pressure or resistance built up to flow of solution. After these cells, the solution passes through cells 9 and 10 in parallel. The discharge of cells 9 and 10 is tailings solution which returns through a sump and a 16-inch pipeline to the barren solution dam by gravity.

In each cell (Fig. 3) the solution enters through a gateway from a launder into the upstream compartment. Most of the solution flows down through the cans in that compartment and through the holes in the screen bottom. The majority of it then passes under the center dividing wall, up through the screen bottom and through the cans in the second compartment before overflowing the discharge gate. Some of the solution will pass longitudinally through the top section of the cans in both compartments by way of the gateway in the center dividing wall. Especially, when the gallonage is extremely high or the cells are full of precipitated copper.

Detinned and shredded cans are delivered to the plant by Proler Steel Corp, in side-dump semi-trailers. The loads are dumped off the side of a ramp about eight feet above the stockpile area. The cans are placed in piles by a diesel-driven mobile rubber-tired crane with a 65 foot boom and a five foot diameter electromagnet. The cans are transferred from the stockpile to the cells as needed, by the crane and magnet. Each magnet load of cans weighs about 650 pounds.

Table II shows typical precipitation cell data. Assays are reported in grams per liter.

TABLE II

PRECIPITATION CELL DATA

		Cell Feed	Cell Tailings
Cu	gms/liter	0.717	.018
H_2SO_4	" "	.67	.06
Fe^{++}	" "	.01	1.00
Fe^{+++}	" "	.05	Tr.
pH		2.32	3.28

Lbs. Acid/Lb. Cu Pptd.	.50 - .80
Lbs. Iron/Lb. Cu Pptd.	1.3 - 1.5
Manshifts/week	18

The above data illustrates the salient features of the precipitation cell operations. 97.5 percent of the copper is stripped from the solution, the acid content is decreased, and the ferric iron content is reduced to ferrous iron.

A comparison of the cell tailings solution which returns to the barren solution dam with the barren solution being pumped to the dumps, shown in Table I, demonstrates that a certain amount of the iron content is precipitated in the barren solution dam. This is advantageous in preventing an excessive build-up of iron in the leaching solutions. Unfortunately, some iron precipitates out in the lines and pump intakes in the form of scale. This can be readily controlled by descaling of the barren solutions lines twice a month with chain-covered, rubber balls and the use of acid, which is added to the barren solution.

CELL WASHING

The first four cells always receive the strongest solution and must be washed the most often. They precipitate about 70 percent of the total production and at present are being washed twice a week. The next group, cells 5, 6, 7 and 8, make about 25 percent more of the total and are washed once a week. Cells 9 and 10, producing the remaining 5 percent, are washed once a week.

Situated below the ten drain valves from the cells are five settling tanks, each 16 feet, 10 feet square by four feet deep. When the cells are being washed, the slurry of copper precipitate and wash water flows to these tanks. After allowing time for the copper to settle, the clear water is decanted, and pumped, by a three inch vertical centrifugal pump of 316 stainless, from a recovery sump back to the cells, to entrap any fine particles of copper.

When a cell is to be washed, wooden gates cut the flow of solution, and the drain valve is opened to the settling tanks. The magnet transfers any loose cans which were not covered by solution to an adjoining cell. When the mass of copper and partially consumed cans is exposed, the copper is washed off the cans through the polypropelyne screen bottom and out the drain valve. Washing is done with two one and one-half inch high pressure hoses equipped with quick shut-off fire nozzles. Water for washing is furnished from the tailings solution sump by a two and one-half inch vertical centrifugal pump of 316 stainless. Driven by a 350 RPM, 25 HP motor, this pump can deliver 200 GPM to the wash hose at 100 pounds pressure.

As the cans are washed clean, the magnet lifts them to the next cell. When the cell is empty, the screen bottoms are inspected and repaired, if needed. Earlier in the plant's operation, drilled, plywood sheets were used, but the polypropelyne sheets proved to have a much longer life with less maintenance troubles and eventually replaced them. The polypropelyne sheets have an infinite life and have to be replaced only when accidently broken or cracked by the magnet. When repairs are complete, the washed cans are replaced, new cans are added, the drain vlave is closed, and the gates are removed to put the cell back in the circuit. Ordinarily, a cell can be washed in 1-1/2 to 2 hours by two men on the hoses and one crane-man.

The operating crew consists of two operators, one helper, and a crane-man. All operations, from leaching solution distribution changes to cell washing, are performed on day shift only. Shift bosses from the concentrator check the plant on afternoon and night shifts to see that the pumps are running properly.

Beside dump work and cell washing, the operators are responsible for sampling of the solutions, controlling the acid addition by pH measurements, and miscellaneous oiling and maintenance around the plant.

PRECIPITATION DRYING AND SHIPMENT

In order to reduce the weight of the cement copper shipped to the smelter and more importantly, to improve its handling characteristics, a drying pad of concrete has been provided on the opposite side of the settling tanks from the cells. The original pad was 35 feet by 105 feet and at times was not quite adequate for the amount of precipitate to be dried. With the extension for the El Tiro west production (E.T.I), the pad was enlarged to 60 by 145 feet. For the El Tiro South dump production (E.T.II), it was enlarged to its present size of 115 by 195 feet.

Once each week, the settling tanks, which have copper in them from washing the cells, are bailed out with the crane and a clamshell bucket.

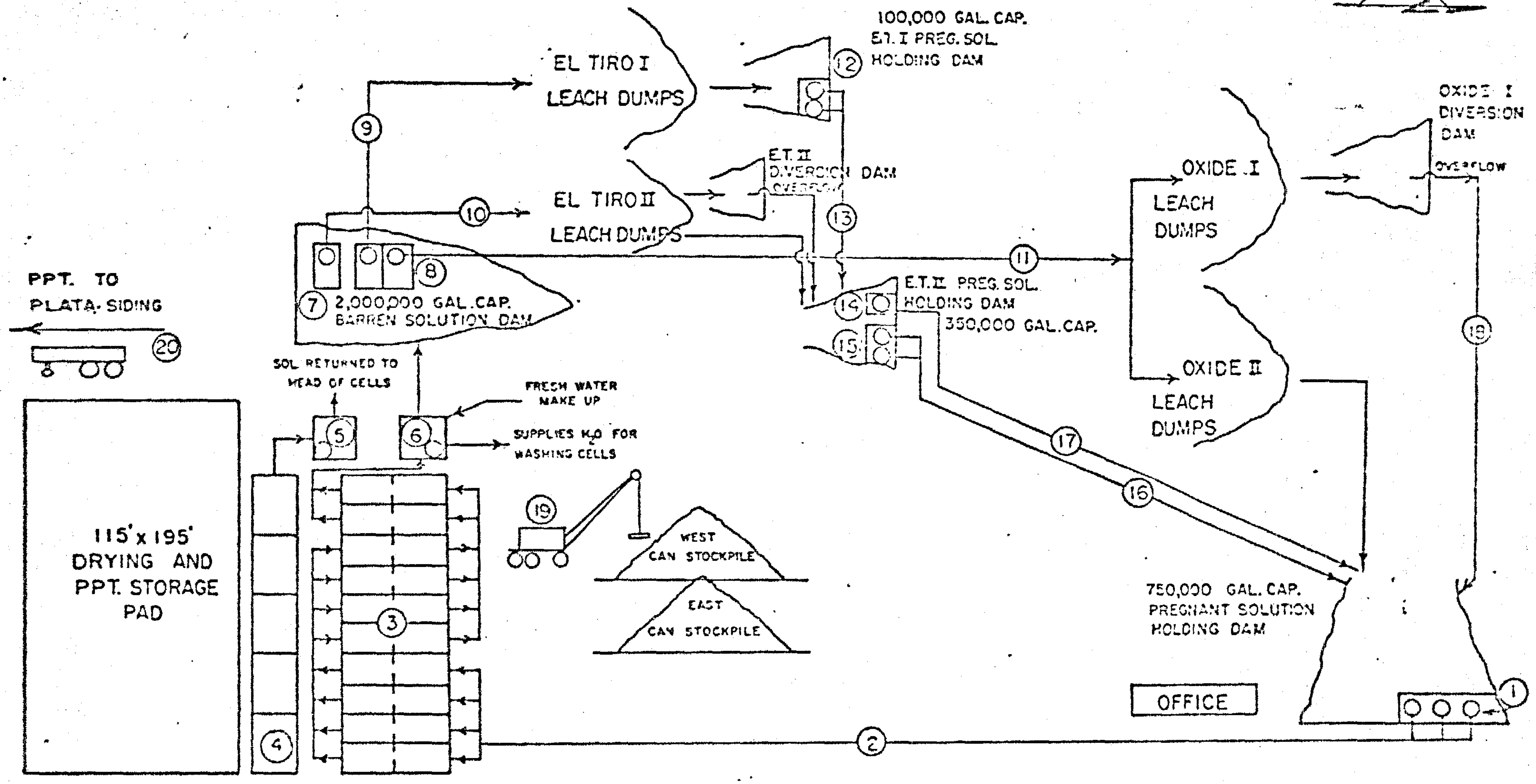
When first placed on the pad, the precipitate will contain 35 to 40 percent moisture. It is placed in an irregular pile and allowed to drain for a day and a half. It is then picked up in small bucket-loads with a front-end loader and laid out in rows about eight to ten inches deep. Occasionally, especially in the winter, there is further drainage of free water from these rows. The drying pad is sloped toward the settling tanks to help in the drainage.

The precipitate is left in rows for 2-3 days in summer and up to 5 or 6 days in winter until the desired shipping consistency is reached. In winter, it is sometimes reworked with the loader to turn the material over and hasten the drying process. When dry enough, the precipitate is placed in a stockpile for ease of loading for shipment.

Carload lots of cement copper average 80 to 82 percent copper and 10 to 15 percent moisture.

The Company railroad siding is a spur off the Southern Pacific mainline at Plata, near the Tucson Casa Grande Highway, a distance of 23 miles from Silver Bell. The same trucks and trailers which haul the concentrates from the mill are used to haul the precipitate to the siding. Usually three trailers, each hauling 35,000 to 40,000 pounds of precipitate, are sufficient to fill a carload. At the siding, the end-dump trailers are unloaded by means of a head-frame and winch into a hopper. From there, conveyor belts deliver the material to open gondolas. The loaded cars are sampled for moisture and copper assay, weighed, and sent to ASARCO's Smelters at Hayden or El Paso.

- - -



1. 3 B-H 5" VERT. PUMPS 316 SS.
2. 10" EPOXY LINED C-A PIPE.
3. 10 8' X 24' CONCRETE PRECIPITATION CELLS.
4. 5 6' X 10' X 4' DEEP DRAIN SUMPS.
5. 3" B-H VERT. RECOVERY PUMP 316 SS.
6. 2 1/2" B-H VERT. WASH DOWN PUMP 316 SS.
7. 8"-H B-H VERT. RAFT MOUNTED PUMP 316 SS.
8. 2 6"-C B-H VERT. RAFT MOUNTED PUMPS 304 SS.
9. 3.5 MILE 8" EPOXY LINED C-A PIPE.
10. 2.0 MILE 14" EPOXY LINED C-A PIPE.
11. 10" EPOXY LINED C-A PIPE.
12. 2 4"-C B-H VERT. PUMPS 316 SS.
13. 8" EPOXY LINED C-A PIPE.
14. 8"-E B-H VERT. PUMP 316 SS.
15. 2 6"-C B-H VERT. PUMPS 316 SS.
16. 2.0 MILES 8" EPOXY LINED C-A PIPE.
17. 2.0 MILES 12" EPOXY LINED C-A PIPE.
18. 16" EPOXY LINED C-A PIPE.
19. UC 78-A LINK BELT CRANE (30 TON CAP) WITH 5' SHRADER MAGNET.
20. 20 TON CAP. PPT. TRAILERS.

B-H = BARRET HAENTJENS
C-A = CEMENT ASBESTOS (JOHNS MANSVILLE TRANSITE) PIPE

SILVER BELL FLOWSHEET

FIG. I

PRECIPITATION PLANT

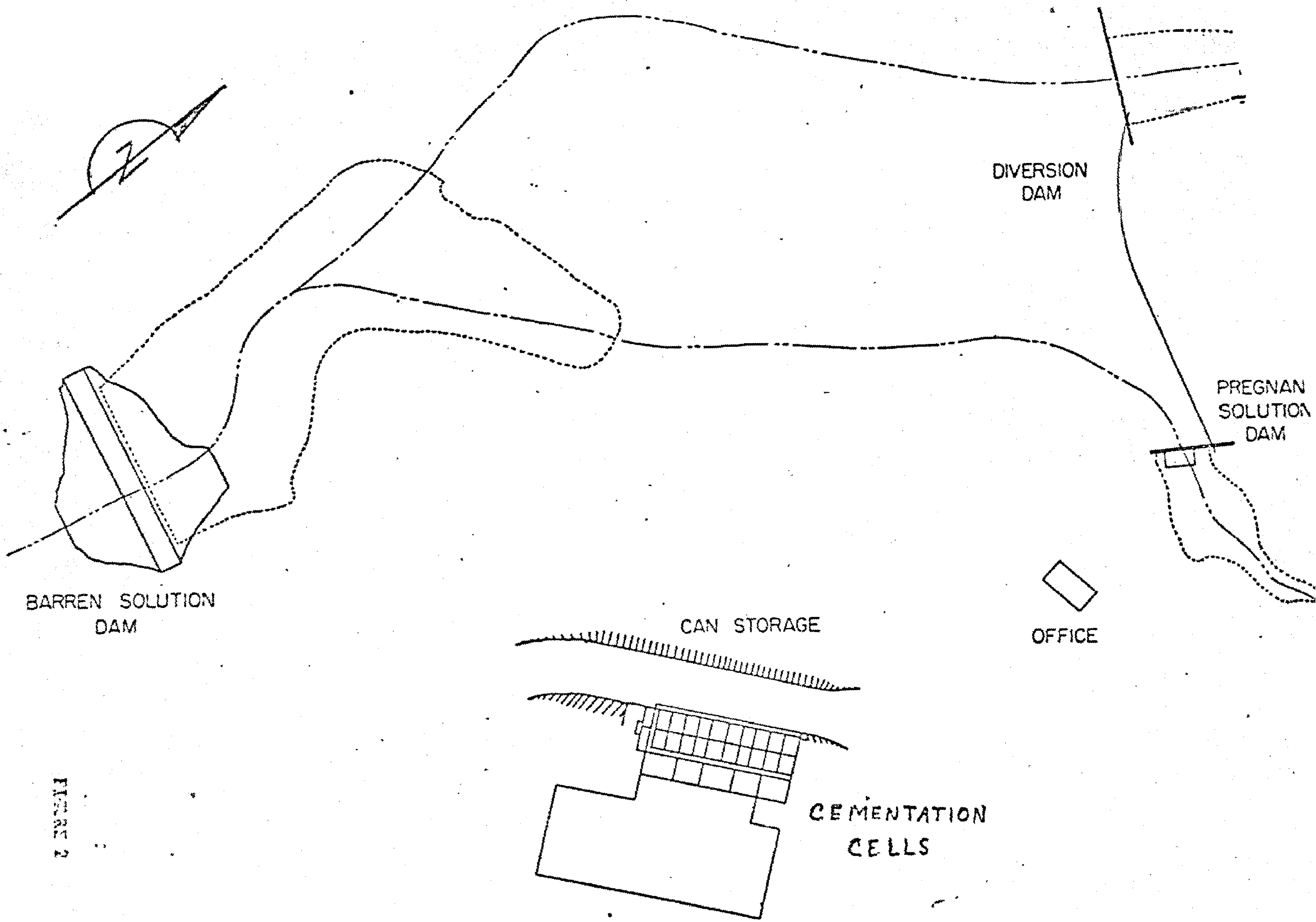

DIVERSION
DAM
PREGNAN
SOLUTION
DAM
BARREN SOLUTION
DAM
CAN STORAGE
OFFICE
CEMENTATION
CELLS

FIGURE 2

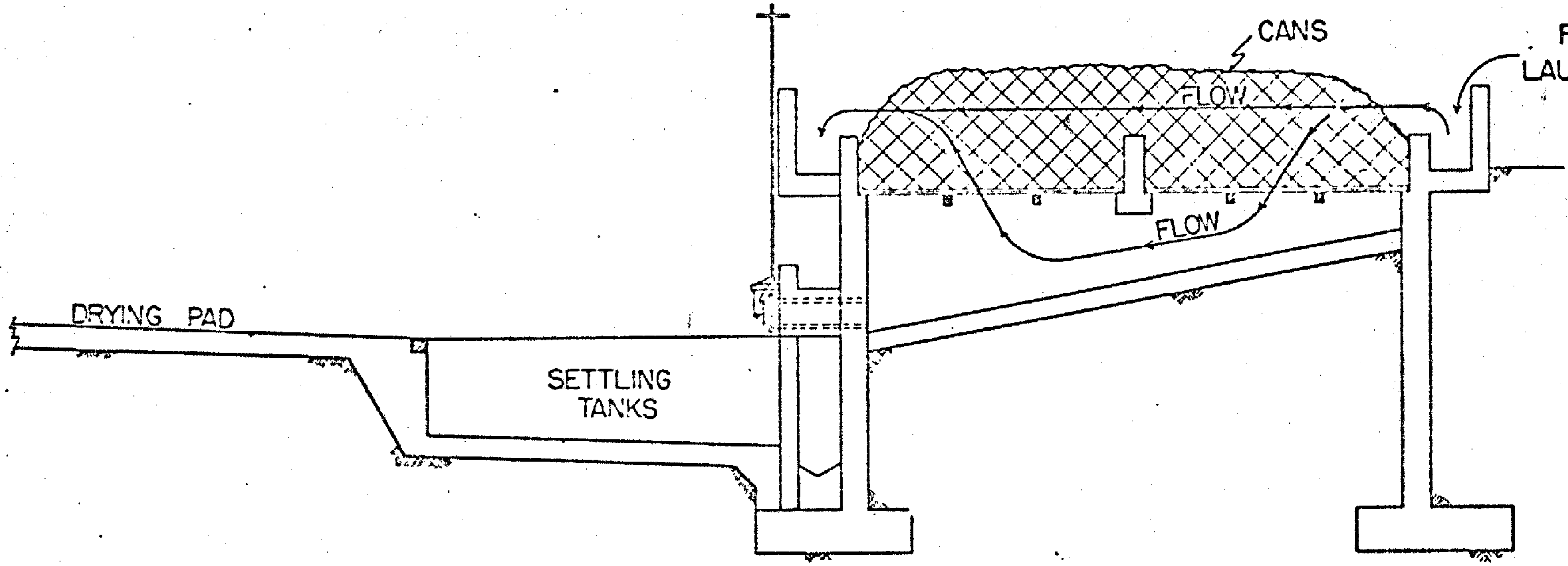
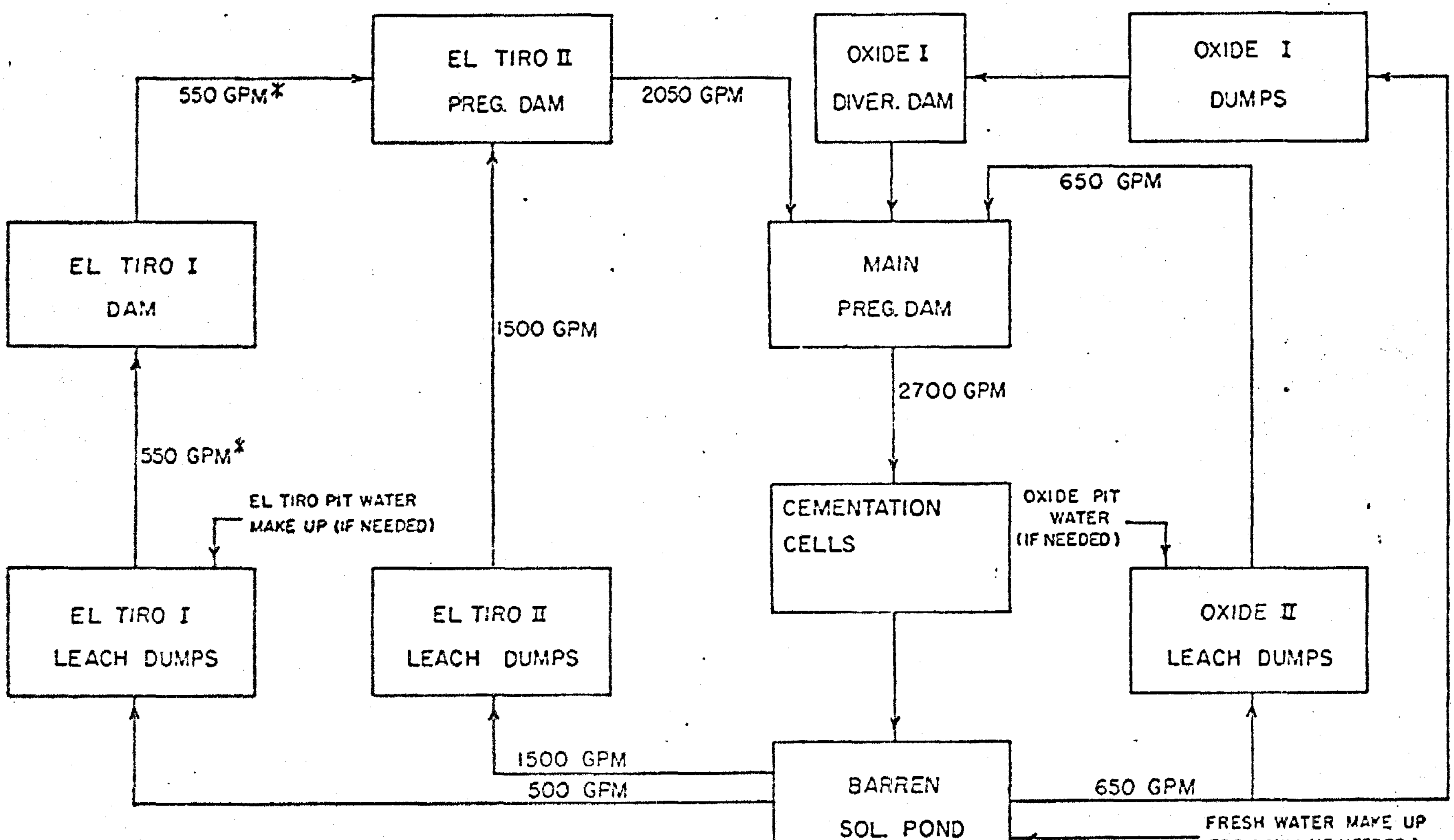


FIGURE 3



* (50-150 GPM FROM SPRING IN DUMP)

SILVER BELL FLOWSHEET

FIG. 4

PRECIPITATION PLANT FLOW RATE DIAGRAM

LEACH PLANT PUMPS

er red	Pump Type*	Serial Number	Purchase Order No.	Purchase Order Date	Motor Size	Present Location	Remarks
	6" C-VSM	N-5957	S-58-2082	11-26-58	100 HP - AC*	Oxide Barren	
		N-6538	61-333-3	2-14-61	100 HP - AC	ET I Barren	
		N-6539	61-333-3	2-14-61	75 HP Rel.*	ET II Preg.	Installed w/50 HP motors,
		N-6540	61-333-3	2-14-61	75 HP "	ET II Preg.	at ET I Preg. in '61. Changed to present location and motors in 1965.
	5" - VS	N-6049	M-870-3	3-30-59	50 HP - AC	Feed to Cells	Originally installed w/10 HP
		N-6050	M-870-3	3-30-59	50 HP - AC	Feed to Cells	motors, replaced w/50 HP motors from ET II Preg. pumps in 1965.
		N-10181	69-632-3	4-29-69	30 HP - GE*	Feed to Cells	
	8" E-VNM	N-8222	65-945-3	5-10-65	125 HP Rel	ET II Preg.	
	8" H-VNM	N-101-80	69-632-3	4-29-69	200 HP-GE	ET II Barren	
	4" C-VSM	N-88220	65-1104-3	6-10-65	25 HP Rel.	ET I Preg.	Converted from B type to C type in 1965.
		N-88221	65-1104-3	6-10-65	25 HP Rel.	ET I Preg.	Converted from B type to C type in 1972.
	3" VS	N-6051	M-870-3	3-30-59	15 HP - US*	Sump Pump	Converted from 3 HP to 15 HP motor in 1971.
	2½" B-VS	N-6048	M-870-3	3-30-59	25 HP - AC	Wash Down	
	LDMDL-31-74SP	59833	S-60-2007	9-12-60	1/3 HP MAS.*	Main Acid Tank	
	VMDL-25-41R	50310	57-213	3-18-57	1/4 HP - US	ET Acid Tank	Originally used in Moly Circuit in Mill - brought to Precip. plant in 1963.

TE: * AC - Allis Chalmers Motors
Rel - Reliance Motors
MAS - Masters Motors

GE - General Electric Motors
US - United States Electrical Motors

* All vertical pumps manufactured by Barrett-Haentjen and of 316 or 304 S.S.
All Acid Pumps manufactured by Milton Roy.

COPPER DUMP LEACHING AT ASARCO'S SILVER BELL UNIT, ARIZONA

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Dump leaching at Silver Bell started in January, 1960. The dumps now being leached are the results of selective mining during the stripping and active mining phases of the development of the two pits. Barren cap-rock is segregated and dumped separately in waste dumps. The copper bearing material in the leach dumps consists of oxide copper minerals and low grade sulfide copper minerals. Neither of these two classes of material could be profitably treated by flotation in the concentrator. The copper values are amenable to dump leaching in closed circuit with iron launder precipitation.

The more basic criteria for dump leaching are: (a) copper mineralization capable of dissolution in leaching solutions within reasonable lengths of time; (b) a host rock which will not consume inordinate quantities of acid, or decrepitate to prevent proper passage of solutions; and (c) a suitable site for placement of the dumps to insure minimal losses of pregnant solution to seepage and good drainage to a central recovery dam.

Additional advantages which are desirable, but not basically necessary are: (a) sufficient pyrite present in the dump material to generate enough free acid and ferric sulfate to dissolve the copper minerals without acid having to be added to the leaching solutions; and (b) not too much ferric sulfate produced in the dumps, which would make subsequent precipitation of the copper difficult or costly.

All of the above basic criteria and additional advantages are realized in the dump leaching operations at Silver Bell.

LEACH DUMPS AND DAMS

At the present time, there are three dumps undergoing leaching, (Figure 1). The original dump, upon which the leaching plant started operations, is in a canyon adjacent to the Oxide Pit. This is now called the Upper Oxide Leach Dump. The ravine underlying this dump runs directly to the main pregnant solution dam near the precipitation cells.

The main pregnant solution dam is of concrete construction, abutting in solid rock on both walls of the canyon. It has a storage capacity of about 160,000 gallons.

The lower Oxide Leach Dump was started in another ravine west of the Upper Dump. It has been formed mainly with leach material developed by stripping and mining after ore production for the concentrator was started in the Oxide Pit. In fact, leach material is still being added to its northern end while the rest of the dump is being leached. The pregnant solution collecting in the ravine under the northern two-thirds of this dump is diverted with an earth-fill dam and a 16-inch pipeline some 250 feet long to the main pregnant solution dam.

The diversion dam has a 12-inch thick concrete key and the earth face is sealed and protected with gunite. The footings of this dam are in conglomerate but there has been very little leakage. The inlet to the diversion pipe is provided with slots for weir boards so the dam can be used as emergency storage of about 100,000 gallons of pregnant solution in case of trouble with the pumps at the main pregnant solution dam.

The southern one-third of the Lower Dump drains to the same canyon as the Upper Dump and the solution goes directly to the main pregnant solution dam.

About 250 yards below the diversion dam and the pregnant solution dam the two ravines from the Oxide Dumps join as one (Fig. 2). Below this junction another 50 yards lies the barren solution dam, another earth-fill dam with a tamped-earth key. The footings of the key are in solid rock on one side and conglomerate on the other. The storage capacity of this dam is roughly one and one-half million gallons.

In early 1961, the West leach dumps being prepared adjacent to the El Tiro Pit were ready for leaching. In order to accomplish this, it was necessary to put in a pump and a pipeline from the barren solution dam to the El Tiro West Dumps, construct a dam across the canyon below, and provide pumps and a pregnant solution return line to the main pregnant solution dam. Also, the additional amount of copper to be precipitated required an increase in the number of cells and in drying area at the precipitation cells. Construction was completed and leaching started on the El Tiro West Dumps in July, 1961.

The El Tiro West Dumps overlie four branches of a main canyon which drains the area. These four join under the dumps and there is only a single underflow. The pregnant solution dam is about 500 feet downstream from the toe of the dumps. It is of concrete, tied into solid rock and has a storage capacity of about 100,000 gallons.

In 1965, the El Tiro South Dumps were large enough to allow starting several rows of ponds on the established area while the crest is continuously being advanced by additional leach material. This required a third barren solution pump installation at the barren solution dam, a pipeline to the dump, a pregnant solution dam across the drainage canyon, and pumps and return lines to the main plant, as well as additional drying pad area at the plant to allow for the anticipated additional production. As is explained later, only modest changes were necessary on the precipitation cells, with no increase in number. The construction work was completed and leaching of the El Tiro

South Dumps started in December, 1965.

The concrete pregnant solution dam for the El Tiro South Dumps is tied into solid rock and has a storage capacity of about 750,000 gallons.

LEACHING SOLUTION DISTRIBUTION

At the barren solution dam, there are two 6-inch vertical centrifugal pumps of type 304 stainless steel, driven by 100 H.P. motors, for pumping the solution to the Oxide area and El Tiro West. The pump to El Tiro South is an 8-inch pump, driven by a 150 H.P. motor. They are floated on rafts to maintain constant submergence regardless of the rise and fall of the water level in the dam. The raft for Oxide and El Tiro West is made up of a wooden deck floating on 24 sealed ten-foot lengths of 12-inch I.D. PVC plastic pipe. The separate raft for El Tiro South is floated on polystyrene flotation billets. The pumps are connected to their respective discharge lines by flexible hoses. The rate of the flow of barren solution from each of the pumps is measured and recorded by orifice plate meters.

The Oxide Dumps receive their leaching solution through a 10-inch pipeline approximately 2,000 feet long, with a static head of 250 feet. On the Upper Oxide, the solution used to be distributed from ten lateral pipes six inches in size. These laterals were provided with one and one-half inch plastic valves on each side of the pipe every 50 to 60 feet. These valves regulated the flow of solution to small, irregular ponds which averaged about 50 to 60 feet square.

When the Lower Oxide was being readied for leaching, it was decided to try a less elaborate method of distribution. In this system, the solution is simply delivered through an open-end 10-inch pipe to a high point and discharged to an open ditch. From the ditch, the solution is cut into one or more ponds as desired. This irrigation system has proven quite successful and the same method is now in use on all dumps.

El Tiro West barren solution is delivered through three and one-half miles of 8-inch pipeline with a static head of 165 feet to the upper benches. El Tiro South solution travels two miles through a 12-inch line, against a static head of 260 feet.

The El Tiro West pregnant solution used to be pumped the full three and one-half miles back to the main plant in its own separate 8-inch line. Since the installation of the El Tiro South pregnant solution dam, however, the West solution is simply pumped by a 4-inch pump to the new dam where it joins the underflow of the South dump. The combined underflows are then pumped by two 6-inch Type 316 stainless steel pumps through two parallel 8-inch pipelines to the main pregnant solution dam. Magnetic type flow meters on the two lines measure and record the flow rates.

On all ponds, every effort is made to get the solution to spread out and cover as much area as possible and not short-circuit through the dump. In some rocky areas it has been necessary to bring in concentrator tailings to be spread in thin layers to reduce the porosity. The whole purpose is to gain as wide a distribution base as possible to get drop-by-drop penetration into the dump. This gives maximum expectation of wetting every rock in the dump, maximum contact time for leaching, minimum of channeling, and above all, a higher grade pregnant solution.

There is a practical limit, however, to these large pond areas, because of the high evaporation rate in the desert country which creates an appreciable loss of water. The percent recovery of leaching solutions varies widely but averages 85 to 90 per cent. Most of the loss can be attributed to evaporation, with minor losses to seepage and soakage into the pores of the rock in the dump.

CHEMICAL REACTIONS OF LEACHING

The chemical reactions involved in leaching of copper minerals were thoroughly studied by John D. Sullivan and others, about 1930. From this source and others, the following overall reactions are given for the dissolution of the principal minerals involved in Silver Bell dump leaching.

Chrysocolla: $Cu\ SiO_3.2H_2O + H_2SO_4 = Cu\ SO_4 + SiO_2 + 3H_2O$

Azurite: $Cu_3(OH)_2(CO_3)_2 + 3H_2SO_4 = 3CuSO_4 + 2CO_2 + 4H_2O$

Malachite: $Cu_2(OH)_2CO_3 + 2H_2SO_4 = 2CuSO_4 + CO_2 + 3H_2O$

Cuprite: $Cu_2O + H_2SO_4 + Fe_2(SO_4)_3 = 2CuSO_4 + 2FeSO_4 + H_2O$

Chalcocite: $Cu_2S + 2Fe_2(SO_4)_3 = 2CuSO_4 + 4FeSO_4 + S$

In vat leaching or agitated leaching, chalcopyrite is usually considered to be insoluble in leaching solutions or, at best, has a reaction requiring such an extremely long time that the amount dissolved is negligible. For these methods of leaching this is true. However, in dump leaching, time of reaction is measured in years. It is probable, then, that a small amount of chalcopyrite does dissolve slowly but inexorably over the years due to the action of ferric sulfate, oxygen, and water.

Chalcopyrite: $CuFeS_2 + 2Fe_2(SO_4)_3 + 2H_2O + 3O_2 = CuSO_4 + 5FeSO_4 + 2H_2SO_4$

At Silver Bell, no extra acid is added to the barren solution when it is pumped to the dumps as leaching solution. Experiments early in the operation of the plant indicated that extra acid addition was neither necessary nor particularily beneficial at this point in the circuit. The source of the sulfuric acid and ferric sulfate required by the above reactions is the reaction of pyrite in the dump with water and oxygen to form these solvents, probably assisted by iron-based bacterial action.

$$\text{Pyrite: } 4FeS_2 + 15O_2 + 2H_2O = 2Fe_2(SO_4) + 2H_2SO_4$$

Table I shows typical flow data and analyses of solutions to and from the dumps. Assays are reported in grams per liter.

TABLE I

Dump Leaching Data

Barren Solution to Dumps

		Oxide	El Tiro West	El Tiro South	Total
GPM		710	450	870	2030
Cu	gms/liter				.030
H_2SO_4	"				.06
Fe^{++}	"				1.18
Fe^{+++}	"				.04
pH	"				3.51

Pregnant Solution from Dumps

		Oxide	El Tiro West	El Tiro South	Total
GPM		630	410	770	1810
Cu	gms/liter	0.622	1.613	.337	.717
H_2SO_4	"	.53	.33	.22	-
Fe^{++}	"	.01	.01	.01	.01
Fe^{+++}	"	.41	.25	.31	.31
pH	"	2.50	2.65	3.05	-

As can be seen from the above data, the barren solution consists mainly of a slightly acidic solution of iron salts, most of which is in the ferrous form. In passing through the dumps, the pregnant solution has accumulated copper, gained in acidity, and converted its remaining iron content almost entirely to the ferric state. Some of the original iron content was precipitated in the distributing ponds and some in the dump itself. To date, there has been no indication that the iron precipitated in the dump may have a deleterious effect on percolation through the dump material.

PREGNANT SOLUTION GRADE CONTROL

The copper content of the pregnant solution underflow is maintained by gradual, progressive changes from one pond to another on the surface of the dumps. Usually, several small ponds are being leached at the same time. As the grade of copper in the underflow tends to fall, a new pond is cut in, and the pond which has been leaching the longest is cut out. The length of time during which a particular pond may be covered by solution varies from a few days up to several weeks. Since this time is dependent only on the copper being extracted, it follows that the depth of the leaching column, the copper content of the rock, the type of mineralization, and the efficiency of the leaching solution distribution are all factors in its determination.

After a pond is cut out of the leaching cycle and the excess solution has drained, the material in the leaching column underneath this pond will remain unwetted until the pond is again cut in for leaching in its turn in the progression from pond to pond to maintain copper grade. The time of this drying period varies from six months to a year at the present rate of operation.

During this drying period, there is still enough moisture in the rock to maintain the humid, oxidizing conditions required by several of the chemical reactions to create additional solvents and to dissolve the copper minerals. By diffusion, capillary action, and evaporation, these salts concentrate at the surface of the rocks and are readily dissolved by the leaching solutions during the next wetting cycle.

PRECIPITATION OF COPPER

To precipitate the copper from solution, detinned scrap cans are used. The main advantage of cans is the large surface area presented to the solutions per pound of metal. This large surface area promotes more efficient precipitation per unit of cell volume than with heavier pieces of iron.

Cans are prepared for use in precipitation by burning in kilns to remove the tin plating and paint from the surface, and the solder from the seams. After this, they are passed through a hammer mill or a toothed roll or some other shredder of suitable design to make them more compact and less bulky to handle, and less wasteful of space in the cells.

Untreated cans will weigh only 8 to 12 pounds per cubic foot. After shredding and compacting, they will weigh 20 to 30 pounds per cubic foot. The limiting factor of the compaction is that if it is carried too far, there will not be enough porosity remaining to get adequate solution penetration and efficient precipitation. Baled cans have never been widely used for this reason.

CHEMICAL REACTIONS OF PRECIPITATION

There are three principal reactions taking place simultaneously in the iron launders. Only one of these is profitable. The other two represent a necessary operating loss to achieve the first; the precipitation of cement copper. The overall reactions are:

$$CuSO_4 + Fe = Cu + FeSO_4$$

$$H_2SO_4 + Fe = H_2 + FeSO_4$$

$$Fe_2(SO_4)_3 + Fe = 3FeSO_4$$

There is also a fourth reaction, between ferric sulfate and metallic copper, which undoubtedly takes place but yields the same net effect as the overall reactions given.

$$Fe_2(SO_4)_3 + Cu = CuSO_4 + 2FeSO_4$$

The copper sulfate formed in this reaction is reprecipitated on metallic iron as in the first equation. The overall effect is that at equal concentrations, the reduction of ferric iron to ferrous iron is the fastest of the three basic reactions. The reduction of copper is the next in rapidity, followed by the reaction between acid and iron.

PREGNANT SOLUTION PUMPING TO CELLS

At the main pregnant solution dam there are two five-inch vertical centrifugal pumps of 316 stainless driven by 50 H.P. motors to pump the solution to the cells. A magnetic type flow meter in the discharge piping measures and records the rate and quantity of flow. The discharge pipe is ten inches in diameter, about 400 feet long from dam to cells.

The only sulfuric acid being added to the entire circuit at present is going into the pregnant solution ahead of the precipitation cells. The amount of acid added is small; only enough to lower the pH to about 2.4. The purpose of this addition is to gain better copper precipitation conditions in the cells by preventing hydrolysis and precipitation of hydrous iron salts in the lower cells where the acid concentration is low. About half of the acid is being added to the pregnant solution at the El Tiro West dam to prevent scale forming in the 3-1/2 miles of pipe which return this pregnant solution and combined El Tiro underflow to the main plant. The other half of the acid is added to the pumps on the feed to the precipitating cells.

Sulfuric acid is stored in a 10,300 gallons tank at the plant and in a 4,090 gallon tank on a dump above the El Tiro West pregnant solution dam. Deliveries from the tanks are metered by variable-stroke diaphragm acid pumps. Until 1967, virtually all the acid was 98% H_2SO_4 or 66° Baumé. Recently, because of a shortage of sulfur, the acid has been 78% H_2SO_4 or 60° Baumé.

CEMENTATION CELLS

The original plant consisted of six precipitating cells. This number was increased to ten in 1961 to gain the needed capacity for the precipitation of the El Tiro solutions.

However, an economic analysis of the precipitating cell operations in 1964 showed that not all of these ten cells were needed. The study made clear that the iron consumed in the lower cells was costing more than the copper precipitated there was worth. As a result of this investigation, five of the cells were cut out of operation and only the remaining five were used to precipitate the copper from about 1000 GPM of solution flow.

Then, when the El Tiro South dump was brought into production in late 1965, with nearly another 1000 GPM of solution to be treated, it was only necessary to cut these five cells back into operation, with minor modifications, to have adequate capacity to handle the flow rate. The economic study of 1964, therefore, has paid off in continuing savings of can consumption and in preventing an unnecessary over-capitalization during the plant expansion of 1965.

Each cell is eight feet wide by five feet deep and is divided into two compartments, each twelve feet long, by a dividing center wall. The tops of the concrete walls are protected by six by ten timbers. At the five-foot depth in the cells is a perforated screen made of one-inch exterior grade plywood which has been locally drilled with three-quarter inch holes on one and one-half centers. The plywood screens are supported by type 316 stainless steel grids which have approximately two inch by four inch openings. The grids rest on two six by eight timbers which are keyed into the side walls in each compartment. Beneath this screen bottom, the concrete floor of the cells slopes to a twelve inch drain valve. This valve is operated by a bell crank and handwheel from the walkway on top of the cells.

CELL OPERATION

Feed solution to the cells passes through Cells 1, 2, 3 and 4 in parallel flow from the feed launder and then returns, in parallel, back through Cells 5, 6, 7 and 8. The advantage of parallel flow on these first cells is that the heavy precipiation from the strong solutions is divided and there is less back-pressure or resistance built up to flow of solution. After these cells, the solution passes through cells 9 and 10 in parallel. The discharge of cells 9 and 10 is tailings solution which returns through a sump and a 16-inch pipeline to the barren solution dam by gravity.

In each cell (Fig. 3) the solution enters through a gateway from a launder into the upstream compartment. Most of the solution flows down through the cans in that compartment and through the holes in the screen bottom. It then passes under the center dividing wall, up through the screen bottom and through the cans in the second compartment before overflowing the discharge gate. Some of the solution will pass longitudinally through the top section of the cans in both compartments by way of the gateway in the center dividing wall.

Cans are delivered to the plant by side-dump semi-trailers. The loads are dumped off the side of a ramp about eight feet above the stockpile area. The cans are placed in piles by a diesel-driven crawler crane with a 65-foot boom and a five foot diameter electromagnet. The cans are transferred from the stockpile to the cells as needed, by the crane and magnet. Each magnet load of cans weighs about 500 pounds.

Table II shows typical precipitation cell data. Assays are reported in grams per liter.

TABLE II

PRECIPITATION CELL DATA

		Cell Feed	Cell Tailings
Cu	gm/liter	0.717	.018
H_2SO_4	"	.67	.06
Fe^{++}	"	.01	1.28
Fe^{+++}	"	.31	Tr
pH		2.32	3.28

Lbs. Acid per lb. Cu Pptd	.50
Lbs. Iron per lb. Cu Pptd	1.65
Man Shifts per Week	18

The above data illustrate the salient features of the precipitation cell operations. 97.5 per cent of the copper is stripped from the solution, the acid content is decreased, and the ferric iron content is reduced to ferrous iron.

A comparison of the cell tailings solution which returns to the barren solution dam with the barren solution being pumped to the dumps, shown in Table I, demonstrates that a certain amount of the iron content is precipitated in the barren solution dam. This is advantageous in preventing an excessive build-up of iron in the leaching solutions.

CELL WASHING

The first four cells always receive the strongest solution and must be washed the most often. They precipitate about 70 per cent of the total production and at present are being washed twice a week. The next group, cells 5, 6 7 and 8, make about 25 percent more of the total and are washed every other week. Cells 9 and 10, producing the remaining 5 per cent, are washed once a month.

Situated below the ten drain valves from the cells are five settling tanks, each 16-feet, 10-inches square by four feet deep. When the cells are being washed, the slurry of copper precipitate and wash water flows to these tanks. After allowing time for the copper to settle, the clear water is decanted, and pumped by a three inch vertical centrifugal pump of 316 stainless from a recovery sump back to the cells to entrap any fine particles of copper.

When a cell is to be washed, wooden gates cut the flow of solution, and the drain valve is opened to the settling tanks. The magnet transfers any loose cans which were not covered by solution to an adjoining cell. When the mass of copper and partially consumed cans is exposed, the copper is washed off the cans through the plywood screen bottom and out the drain valve. Washing is done with two one and one-half inch high pressure hoses equipped with quick shut-off fire nozzles. Water for washing is furnished from the tailings solution sump by a two and one-half inch vertical centrifugal pump of 316 stainless. Driven by a 350 RPM, 25 H.P. motor, this pump can deliver 200 GPM to the wash hose at 100 pounds pressure.

As the cans are washed clean, the magnet lifts them to the next cell.

When the cell is empty, the screen bottoms are inspected and repaired, if needed. Average life of the screens is about four months in the first four cells; longer in the others. When repairs are complete, the washed cans are replaced, new cans are added, the drain valve is closed, and the gates are removed to put the cell back in the circuit. Ordinarily, a cell can be washed in about an hour and a half by two men on the hoses and one crane-man.

The operating crew consists of two operators and a crane-man. All operations, from leaching solution distribution changes to cell washing, are performed on day shift only. Shift bosses from the concentrator check the plant on afternoon and night shifts to see that the pumps are running properly.

Beside dump work and cell washing, the operators are responsible for sampling of the solutions, controlling the acid addition by pH measurements, and miscellaneous oiling and maintenance around the plant.

PRECIPITATE DRYING AND SHIPMENT

In order to reduce the weight of the cement copper shipped to the smelter and more importantly, to improve its handling characteristics, a drying pad of concrete has been provided on the opposite side of the settling tanks from the cells. The original pad was 35 feet by 105 feet and at times was not quite adequate for the amount of precipitate to be dried. With the extension for the El Tiro West production, the pad was enlarged to 60 by 145 feet. For the El Tiro South dump production, it was enlarged to its present size of 115 by 195 feet.

About once each week, the settling tanks which have copper in them from washing the cells are bailed out with the crane and a clamshell bucket.

When first placed on the pad, the precipitate will contain 35 to 40 per cent moisture. It is placed in an irregular pile and allowed to drain for a day and a half. It is then picked up in small bucket-loads with a front-end loader and laid out in rows about eight to ten inches deep. Occasionally, especially in the winter, there is further drainage of free water from these rows. The drying pad is sloped toward the settling tanks to help in the drainage.

The precipitate is left in rows for five more days in summer and up to 12 or more days in winter until the desired shipping consistency is reached. In winter, it is sometimes reworked with the loader to turn the material over and hasten the drying process. When dry enough, the precipitate is placed in a stockpile for ease of loading for shipment.

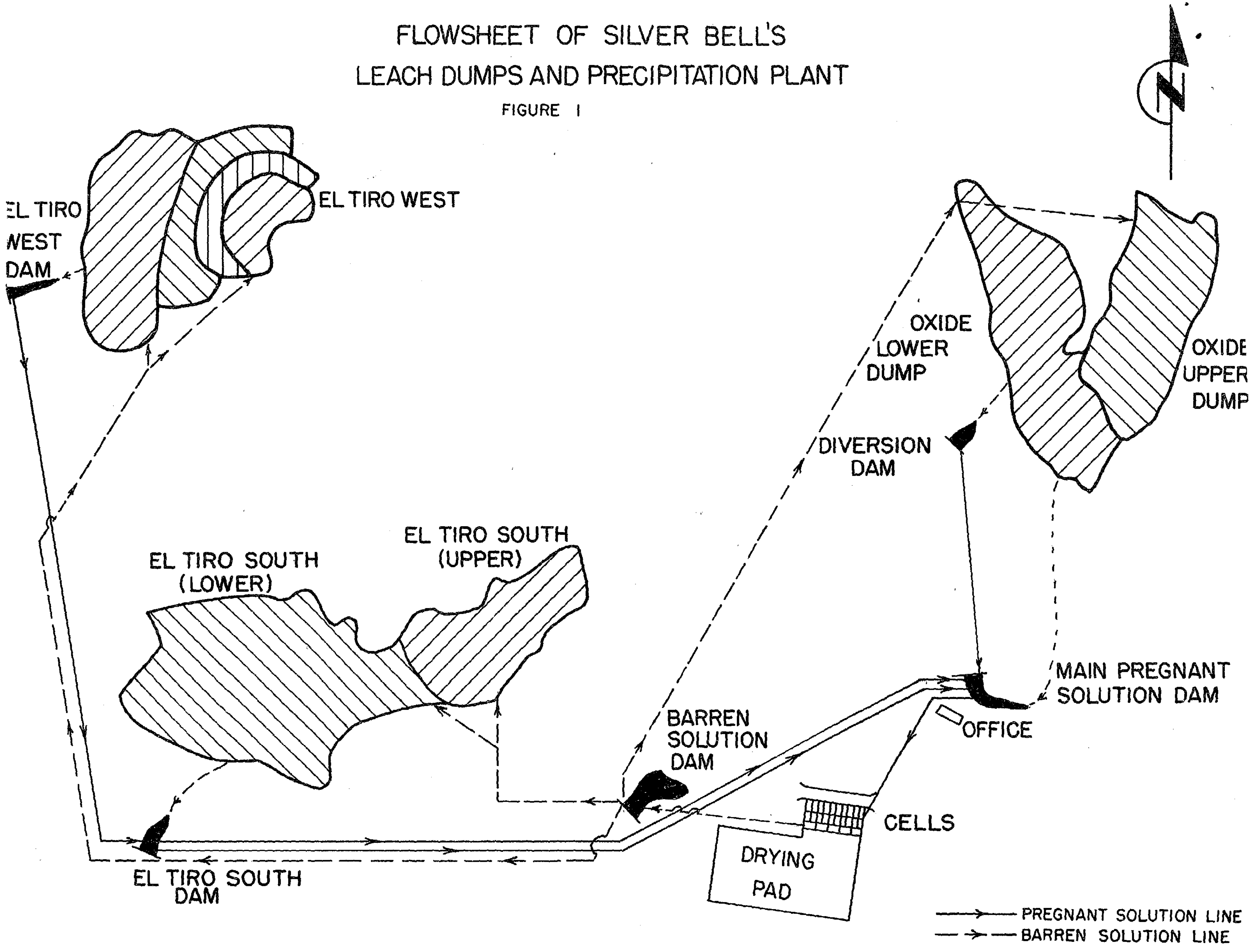
Carload lots of cement copper average 82 per cent copper and 15 per cent moisture.

The Company railroad siding is a spur off the Southern Pacific main-line at Plata, near the Tucson-Casa Grande Highway, a distance of 23 miles from Silver Bell. The same trucks and trailers which haul the concentrates from the mill are used to haul the precipitate to the siding. Usually three trailers, each hauling 35,000 to 40,000 pounds of precipitate, are sufficient to fill a carload. At the siding, the end-dump trailers are unloaded by means of a head-frame and winch into a hopper. From there, conveyor belts deliver the material to open gondolas. The loaded cars are sampled for moisture and copper assay, weighed, and sent to ASARCO's Smelter at El Paso.

FLOWSHEET OF SILVER BELL'S

LEACH DUMPS AND PRECIPITATION PLANT

FIGURE 1



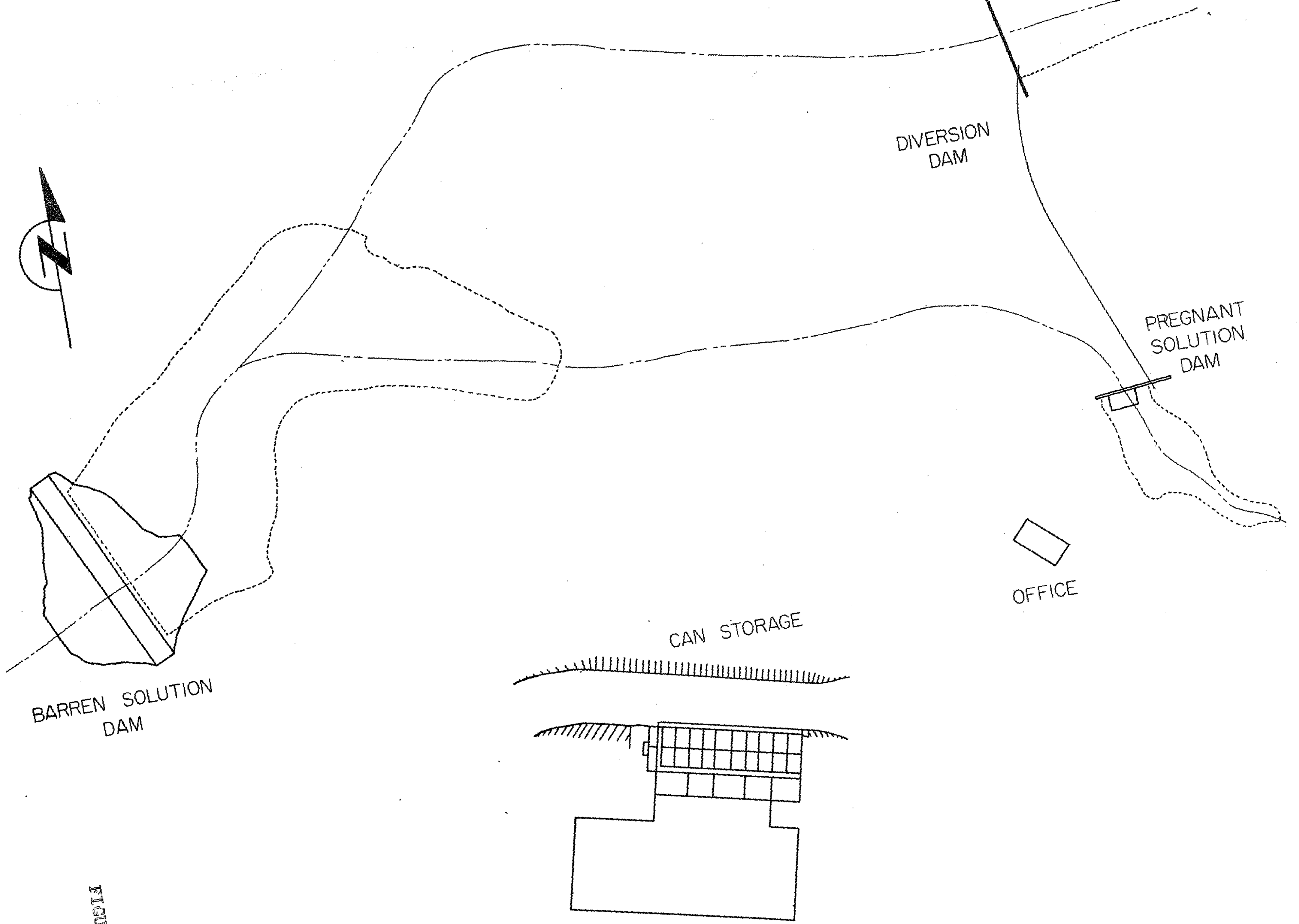

DIVERSION DAM
PREGNANT SOLUTION DAM
OFFICE
CAN STORAGE
BARREN SOLUTION DAM

FIGURE 2

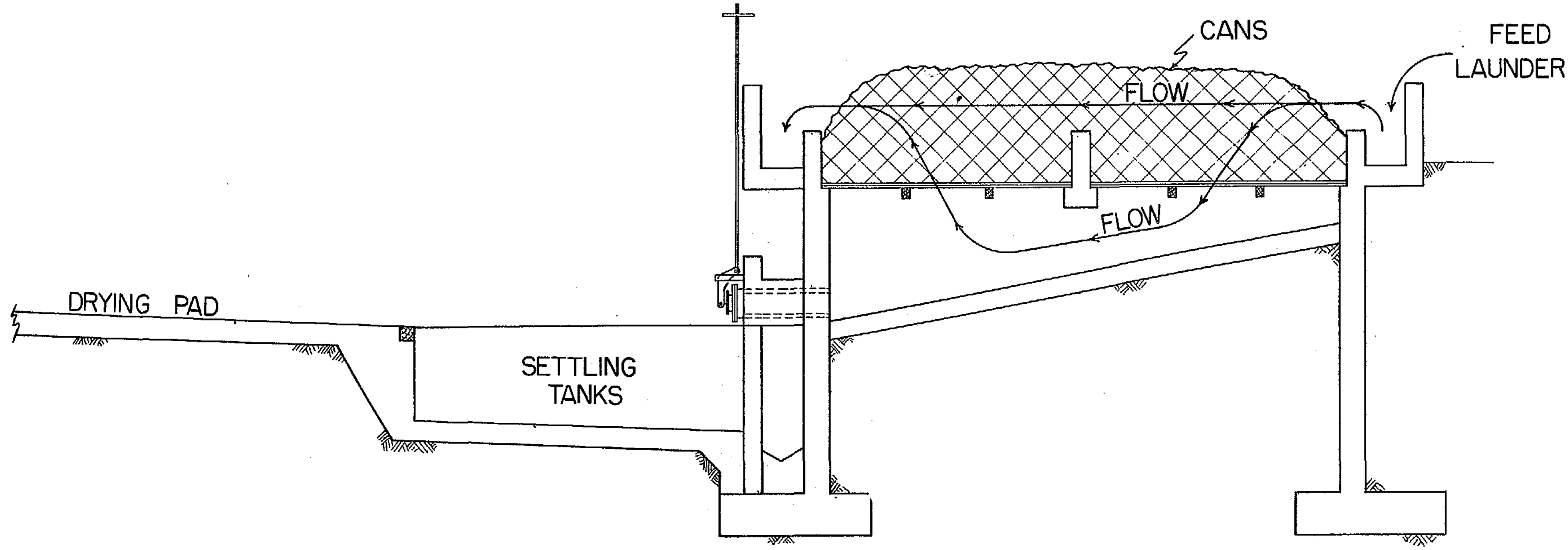


FIGURE 3

BY-PRODUCT MOLYBDENUM RECOVERY

AT SILVER BELL UNIT

AMERICAN SMELTING AND REFINING COMPANY

By

Clement K. Chase, Chief Metallurgist

Revised December 1966

BY-PRODUCT MOLYBDENUM RECOVERY AT SILVER BELL UNIT

AMERICAN SMELTING AND REFINING COMPANY

INTRODUCTION

The Silver Bell Unit of American Smelting and Refining Company is an open pit copper operation located forty miles northwest of Tucson in the Silver Bell Mountains.

Although Silver Bell is known primarily for copper production molybdenite is also produced as a by-product in the 11,500 ton-per-day flotation mill.

The Silver Bell ore comes from two pits approximately two miles apart. Copper mineralization is essentially chalcocite resulting from two to three-fold secondary enrichment in a highly altered zone. A typical porphyry copper deposit, Silver Bell ore is of medium hardness. The molybdenum occurs in the ratio of about one part molybdenum to seventy-five parts of copper. In the copper concentrate, which is the starting point for separate molybdenum recovery, the ratio is one part molybdenum to forty parts of copper.

Molybdenum mineralization is primary. Enrichment is not evident but it is notable that the molybdenite content is higher in the more siliceous, harder rocks. The mineral occurs as diversely oriented flakes of fairly constant grain size, mostly in fractures associated with quartz veins. Most of it appears unoxidized; secondary coatings on the molybdenite flakes are fortunately rare.

In May of 1962 a modification of the earlier molybdenite circuit was put into operation. This paper describes the new molybdenite plant.

The metallurgical design and layout of the circuit was by Mr. A. B. Romney and Mr. Russell Salter under the direction of Mr. Norman Weiss. Detail design was handled by the Central Engineering Office of American Smelting and Refining Company, Salt Lake City, Utah. Construction was by Western Knapp Engineering Company of San Francisco, California.

THE SILVER BELL MOLYBDENITE CIRCUIT

Sulfide Copper Circuit:

The starting point for the molybdenite circuit is the cleaned copper concentrate, but a few words on the copper circuit are in order since this has a bearing on subsequent results.

The ore is ground with lime in ball mills and classified to 15% on 65 mesh and 50% minus 200 mesh in an all-cyclone circuit. Copper flotation collector is primarily potassium hexyl xanthate (Z-10) plus occasional auxiliary use of a dithiophosphoric salt collector (AF-238), and the frother is a mixture of 75% hardwood creosote-25% steam distilled pine oil. The copper rougher is in open circuit, rougher tailings being the greater portion of final tailings. The rougher concentrate is classified, the sands reground, and the combined sands and slimes cleaned twice with lime to attain the usual shipping grade of about 30% copper. The first cleaner tailing is scavenged, the scavenger tailing joining the final tailing and the scavenger concentrate joining the rougher concentrate.

Dextrin Depression of the Molybdenite:

The circuit is shown schematically in Figure 1. Copper concentrate is withdrawn from a 100 foot diameter thickener to a 6 ft. x 6 ft. open conditioner tank at the rate of about 195 TPD. Dextrin (either Stadex 120 or Clinton 761) in 12% solution is added at the approximate rate of one and one-half pounds per ton of concentrate, and make-up water is added from the 50-foot diameter thickener further down the circuit. The pulp is diluted to about 20% solids with this water (which contains some dextrin), and lime is added to the level of 0.2 pounds free CaO per ton of solution.

Next this slurry is fed to the molybdenite depression section, the first stage of which is a 30 foot long Miami-type air cell. Most of the copper floats here together with a small proportion of the molybdenite. To the froth from the air cell are added a half pound of dextrin per ton and a small amount of lime, in a bank of six #24 Denver Cells. The froth from this bank of cells is a final copper concentrate low in molybdenum and goes directly to the copper filter plant. The rougher and cleaner dextrin tailings containing the molybdenum are combined and sent to a 50 foot diameter thickener.

Since the combination of dextrin and high alkalinity from the lime produces a virtually non-settling pulp, acid is used to settle this material. The acid added modifies the pH from 11 to 8. The source of the acid used, both sulfurous and sulfuric, is the scrubber slurry from the dust collection system on the five-hearth furnace described below. Settling is adequate at this point.

Heating Step:

The thickener underflow is pumped to an 8 ft. x 8 ft. Eimco drum filter mounted directly on top of the 18-foot diameter Bartlett-Snow-Pacific five-hearth furnace fired by natural gas. The filter cake drops through a slot onto the top, or drying hearth, and proceeds downward by action of the rabble arms.

The furnace operates more as a drier than a roaster. Evolution of sulfur dioxide is not our objective, although some does occur as a result of the fall of fines from hearth to hearth through the flame areas. Retention time in the furnace was about four hours during initial operations but this has been reduced to 1-1/2 hours by increasing the number and length of the teeth on the rabble arms in the three upper hearths. The shaft speed was also increased from 2/3 to 1 RPM. The effect of these adjustments was to increase through-put while still maintaining satisfactory heating.

Measurement of hearth temperatures is accomplished through thermocouples indicating on an 8-point strip chart recorder. Final product temperature is 575 to 625 degrees Fahrenheit and it is important to keep it in this range since under-heating fails to destroy the dextrin coating with poor recovery and over-heating calcines the material with resultant high lime requirement and poor flotation conditions.

The furnace feed is occasionally wet and sticky. Under these conditions lumps can form and move through the furnace unbroken. Testing proved that the interior of such lumps can be under-heated, so two rollers were chained 180 degrees apart to the rabble arms on the middle hearth. Each roller is two feet in diameter by one foot wide. Constructed of 3/8" steel plate, they weigh about 250 pounds apiece and crush the lumps thoroughly.

The dust in the furnace gases is removed in a Doyle stainless steel wet scrubber. Installed between the exhaust fan and the stack, this unit scrubs out virtually all of the dust in the stack gases and the resulting slurry is returned to the thickener ahead of the furnace for recovery of the molybdenum and copper values.

<u>Reflotation of the Molybdenite:</u>

Hot calcine from the furnace drops through a chute into a 4 ft. x 5 ft. repulper tank and is slurried with fresh water. A normal roast will overtreat some fine particles and an acid repulp results. Since acid conditions here tend to activate copper and iron minerals that we want to depress, lime is added to the repulper under automatic control to maintain the pH near 7.

Soda ash is also an effective alkali, but is more costly. Lime is fed from a branch line off the regular mill milk-of-lime loop to a supply tank in the molybdenite plant. This was necessary because the molybdenite plant is distant from the milk-of-lime loop and considerably lower in elevation.

Fuel oil is added to the repulper as a collector for molybdenite. Fuel oil and an alcohol frother are added at the head of the refloat section which consists of six #48 Agitair cells. These reagents may also be stage added along the cells. Emulsification and dispersion of the oil in the pulp is added by addition of 0.1% surfactant.

The original laboratory work showed refloat recoveries in the middle nineties. Plant operations have not been up to this level because of the effect of the circulating middling particles in the first, second, and third cleaner tailing streams. Effect of these return streams was not easily assessed in laboratory testing.

The effect of slime coating on dextrin depression and refloat of the molybdenite mineral is difficult to determine but we feel that they may be responsible in part for occasional sub-standard performances.

Cleaning Operation:

The refloat concentrate, containing minor amounts of copper and iron minerals, is cleaned twice in #36 Agitair cells without further reagent addition. The first cleaner tailing can be returned to the refloat or can be routed to the roaster or the final copper concentrate. The refloat concentrate, twice cleaned, is then ground in a 3 ft. x 4 ft. Marcy overflow ball mill. We now have this mill in open circuit after several years of struggle to keep a small cyclone in operation but better regrinding through classification and return of oversize to the regrind unit would be helpful at times. The reground concentrate is cleaned five more times in a countercurrent cleaning circuit, with sodium cyanide and frother added to the final cleaner. The final product usually averages 51% Mo and less than 0.5% copper. Iron and insoluble commonly run about 3.5% each.

The third cleaner tailing was originally returned to the refloat stage but it soon became evident that the copper-cyanide complex anion was activating copper and iron minerals. Further testing established that this intermediate tailing should be returned to the furnace. This has been done but results in a long recycle path for a portion of the molybdenite.

Filtering and Drying:

Froth from the final cleaning stage runs by gravity to one of three holding tanks where it can be cyanide-leached, if required, to lower the copper content. This is seldom necessary. It is fed from the holding tanks to a three foot diameter, two-disc Eimco leaf filter. Filtered concentrate is held on the floor below in a surge pile whence it is charged into an enclosed Abbé rotary vacuum drier in 1000 pound batches. Discharge is by flexible tube into 55-gallon drums fitted with removable tops. Dust loss at this point is minimal, even though moisture averages only a half percent.

The circuit described above has not changed in any important respect since initial operation in May of 1962 but small mechanical improvements have been made from time to time which have made operations easier.

Some operating data appear in Table I. Also, a bibliography of literature of interest is appended.

SUMMARY

A considerable proportion of the molybdenite in the Silver Bell porphyry ore floats with the finished copper concentrate. This is the starting point for the molybdenite circuit, which includes the following steps:

1. Two-stage dextrin depression of the molybdenite into a product representing one-third of the copper concentrate tonnage.

2. Thickening, filtering, and heating of this fraction.

3. Reflotation of the molybdenite by addition of lime, fuel oil, and frother.

4. Seven-stage flotation cleaning with the addition of sodium cyanide to attain specification grade of molybdenite.

The following conclusions result from our experience with this circuit:

1. The dextrin circuit performs efficiently, concentrating most of the molybdenite into a fraction representing one-third of the copper concentrate weight, a dextrin coating on the molybdenite surface probably being the mechanism of depression.

2. Heating is a positive process to effect oxidation of the collector coating on the copper and iron mineral surfaces and, most importantly, to destroy the dextrin coating on the molybdenite surfaces so that the molybdenite can be re-floated.

3. Very satisfactory copper suppression in the final molybdenite concentrate is achieved in this operation.

4. Production of a satisfactory concentrate from the Silver Bell igneous ores is generally independent of the copper mineralogy. Since parts of the ore body contain copper also as chalcopyrite although mineralization is primarily chalcocite, and the proportion of chalcopyrite may increase further in the future, the ability to recover molybdenite from both these copper minerals, and mixtures thereof, is important. Chalcopyrite in sedimentary ores at Silver Bell is a special problem for which circuit modification may soon be necessary.

5. The first three full years of operation of the new molybdenite circuit showed twice as much production as the average of the six preceding years. It should be noted, however, that the capacity of the mill was increased by 20% during the latter period.

TABLE I

OPERATING DATA - SILVER BELL MOLYBDENITE PLANT

Reagent Consumption - Typical Year

Reagent	Pounds Consumed per ton of Copper Concentrate Treated
Dextrin	2.0
Lime	15.9
Fuel Oil	0.7
Frother	0.1
Sodium Cyanide	1.5

Regrind Balls, used, scrap	0.2 #/ton concentrate treated
Natural Gas	555 Cu ft./ton Conct. treated
Kilowatt Hours	29.9 per ton Conct. treated

Percent Mo Distribution - Typical Year

Mill Feed	100.0%
Molybdenite Concentrate	58.7%
Copper Concentrate	24.7%
Final Tailings	16.6%

In-Circuit Tonnage and Assays, Typical Day

Circuit	Approx. TPD	Approx. % Cu	Approx. % Mo
Dextrin Cleaner Feed	195	30	0.5
Dextrin Tailings (to furnace)	85	20	1.0
Copper Concentrate (to Cu Smelter)	110	35	0.1
Refloat Feed	100	14	1.6
Refloat Concentrate	15	14	9.0
Refloat Tailings	85	14	0.2
First Cleaner Tailings*	8.3	17	1.0
Second Cleaner Tailings*	4.4	15	5.0
Third Cleaner Tailings*	2.0	7	15.0
Final Molybdenite Concentrate	1.3	0.4	51.0

*These products are in circulation.

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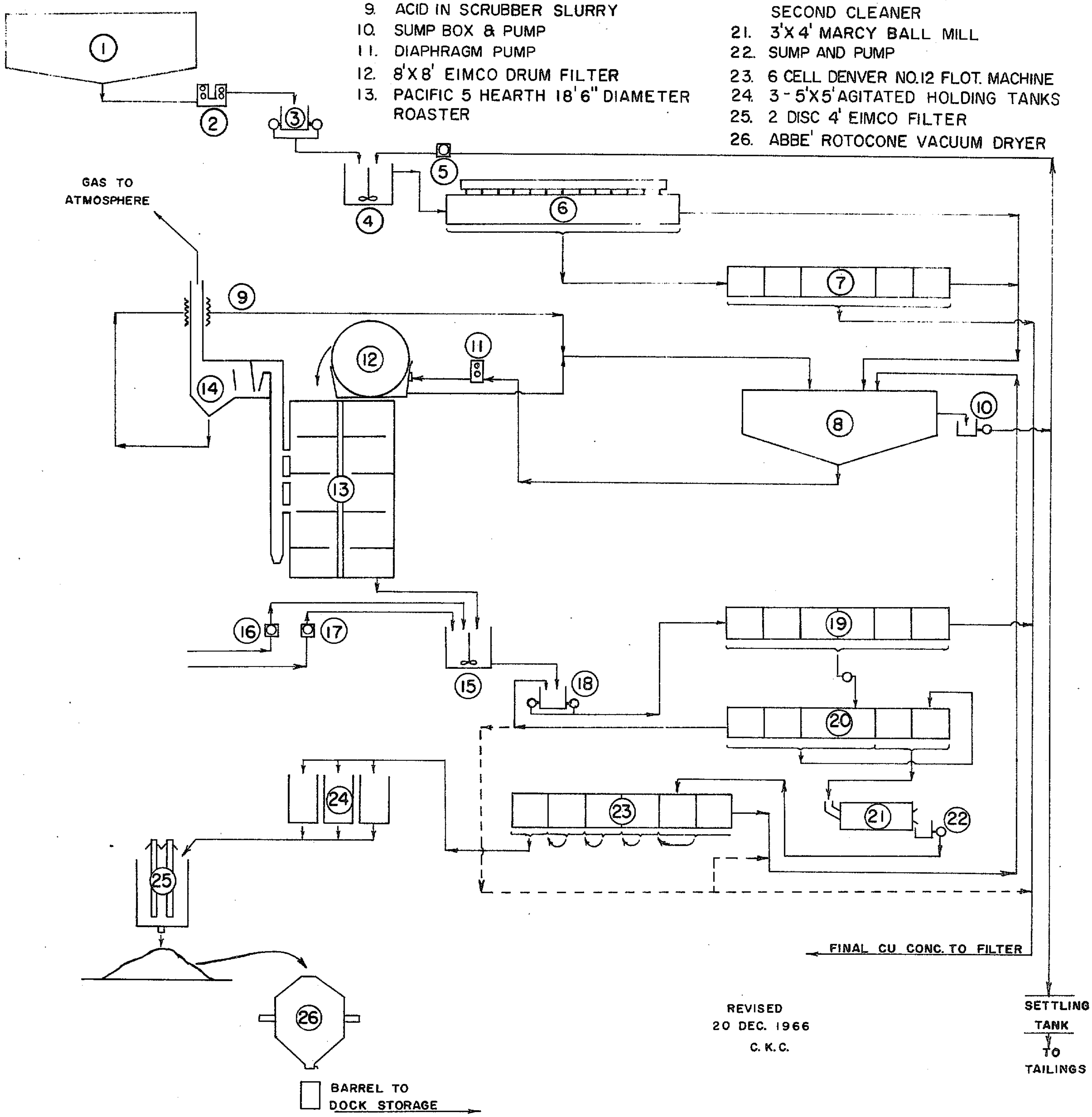
FIGURE I

CLEMENT K. CHASE

1. 100' DORR THICKENER
2. DIAPHRAGM PUMP - NO. 6 DUPLEX
3. PUMP BOX & PUMPS
4. 6'X 6' CONDITIONER
5. AUTOMATIC DENSITY CONTROLLER
6. 30' AIR LIFT FLOTATION MACHINE
7. 6 CELL NO. 24 DENVER FLOTATION MACHINE
8. 50' DORR THICKENER
9. ACID IN SCRUBBER SLURRY
10. SUMP BOX & PUMP
11. DIAPHRAGM PUMP
12. 8'X 8' EIMCO DRUM FILTER
13. PACIFIC 5 HEARTH 18'6" DIAMETER ROASTER
14. TURBULAIRE WET SCRUBBER 16-T
15. 5'X 4' CONDITIONER
16. AUTOMATIC LIME CONTROLLER
17. AUTOMATIC DENSITY CONTROLLER
18. PUMP BOX & PUMPS
19. 6 CELL NO. 48 FLOTATION MACHINE
20. 6 CELL NO. 36 AGITAIR FLOTATION MACHINE, FIRST & SECOND CLEANER
21. 3'X 4' MARCY BALL MILL
22. SUMP AND PUMP
23. 6 CELL DENVER NO.12 FLOT. MACHINE
24. 3 - 5'X 5' AGITATED HOLDING TANKS
25. 2 DISC 4' EIMCO FILTER
26. ABBE' ROTOCONE VACUUM DRYER



SILVER BELL MOLYBDENITE PLANT FLOW SHEET

CYCLONES AS PRIMARY CLASSIFIERS

AT

SILVER BELL

PRESENTED AT ANNUAL MEETING
ARIZONA SECTION AIME
DECEMBER 7,1959
TUSCON, ARIZONA

CYCLONES AS PRIMARY CLASSIFIERS AT SILVER BELL

By Russell Salter

Mill Superintendent, Silver Bell Unit American Smelting and Refining Company, Silver Bell, Arizona

OCTOBER 1959

The removal of the spiral classifiers from the primary grinding circuits of the Silver Bell concentrator and their substitution by cyclones climaxed our investigation for a more satisfactory means of primary classification. The work leading to this substitution was motivated by the high charges for labor and parts needed for maintenance and repair of the spiral units, the lengthy shut-downs incurred, and the nuisance of replacing the frequent broken shafts.

Our search of published literature revealed that experience with cyclones in primary single-stage grinding circuits was not extensive. The only local investigation reported at that time was that of Morenci 1/. The work of a group of South African gold mine operators reported by Dennehy 2/ among other, provided much evidence of the worth of the cyclones.

Some of the advantages of the cyclones reported or inferred were (1) sharper classifications - less misplaced material, (2) saving in floor space - important in new design - or to relieve crowded space, (3) saving in capital investment, (4) saving in maintenance cost and plant down time, (5) ability to shut down under full load, (6) ability to bring the circuit to balance rapidly, (7) elimination of cyclic surges, and (8) saving in power.

Whether or not less power would be used by cyclones would appear to depend on an individual installation and its relative pump head, inlet pressure needed, flow volume, etc. The other points seemed valid and attractive.

An additional advantage from a metallurgical point of view is the ability of the cyclone to produce a suitable overflow at higher density than the mechanical classifier and still provide a sand of proper dilution for return to the mill. Also, an improvement in the size relationship of gangue to mineral was expected to be advantageous in the flotation circuit.

Although these published reports indicated that successful cyclone separations could be made on primary circuit products, a tabulation by Hitzrot 3/ of major applications of various types of classifiers indicated that cyclones would handle separations only in the range of 100 mesh to 5 microns (as compared with 20 mesh to 100 mesh for mechanical classifiers), and the maximum top size given (for up to 30 inch cyclones) was only 14 to 20 mesh. The normal feed density range was given as 1 to 30 pct. solids - a far cry from a ball mill discharge. Morenci's experience, however, had shown that these limitations could be exceeded considerably.

The sobering consideration was excessive pump wear. Pumping the full ball mill discharge of siliceous, angular and coarse (2% plus 3/8", 20% plus 10 mesh) material against much of a head would ruin ordinary pumps. Since pump wear increases approximately as the cube of pump speed, the lowest permissible speeds were seen to be vital to hold wear to a minimum. We believe that the use of large capacity cyclones at low inlet pressures might permit acceptable pump life.

Experience at Morenci indicated that it was advisable to prevent entry of the larger particles into the pump. Use of a spiral classifier for the first stage of classification, sending the spiral overflow into the cyclone as the second stage was recommended.

With this background we converted one of the primary grinding circuits at Silver Bell into dual-stage classification with the original 78 inch duplex spiral classifier serving as the first stage and a bank of three 20 inch Krebs cyclones, fed by 10 x 10 inch Linatex pumps, as the second stage.

One-fourth of the rougher flotation cells - two banks of 12 No. 66 Fagergrens - was isolated with this test grinding circuit so that the effect of the cyclone classification on rougher flotation could be compared with "normal" flotation of spiral overflow. This isolation was possible since we have no return of middling to the rougher circuit. It was impossible to isolate the flow beyond the rougher circuit so the effect on the cleaning circuits and on regrinding was indeterminable.

The initial test program and some of the objectives of the program were first described in our 1957 article on the status of cyclones in Arizona. 3/ At the time that this initial presentation was prepared, the test program had been in progress less than five months. During the interval between then and the final removal of the spiral classifiers, many modifications were made in equipment and operating procedure. A chronological summary of the test program will be given later.

Our present primary classification is handled by banks of four 20 inch Krebs cyclones fed from a common header by a 10 x 10 inch Linatex horizontal slurry pump. Pump protection from tramp steel in the ball mill discharge is provided by hydraulic ball traps and large pump sumps. The pumps, which are equipped with manually adjusted variable pitch sheaves, operate at 360-390 rpm to suit circulating loads and draw from 22-23 ph, handling 1570 to 1670 gpm against 25 feet of total discharge head.

The cyclones are operated at an inlet pressure of 2-1/2 psi. The cone angle is 15°; the vortex finder diameter is nine inches and that of the orifice is three and one-quarter inches.

OPERATING AND MAINTENANCE DATA

The prime benefit of the conversion to cyclones has been in reduced operating and maintenance requirements. Mechanically the pump-cyclone units are more reliable than the spirals; there has been no failure with the cyclones whereas there had been frequent occasions of replacement of spiral components because of breakage or excessive wear. The cyclones require no special consideration when shutting down either normally or in case of power failure and there is no special technique needed to start after shutdowns.

Our maintenance costs for cyclone-pump units vary between 40 and 50 per cent of those for spirals. Plant lost-time due to classifier maintenance is considerably less. The biggest single maintenance item of the cyclone system is the pump which accounts for two-thirds of the total cost.

Pertinent maintenance data of the cyclone circuit can be indicated in days of useful life and tons handled but note that some parts were removed before necessary because of convenience. The impeller of the Linatex pump suction plate and glands all lasted about 120 days, equivalent to 235,000 tons of new feed or including circulating loads, 1,175,000 tons. Back plates lasted twice as long and casings and bearings have not yet shown excessive wear. The cyclone head liners have lasted 840 days and cone liners 365 days, handling tonnages of 1,640,000 and 714,000 tons of new feed respectively, or 8,200,000 and 3,570,000 tons including circulating loads. Vortex finders and the currently used apex orifices of Refrax, a bonded silicon carbide, have been very durable.

EFFECT OF CYCLONES ON METALLURGICAL RESULTS

The data of Table I were selected from regular monthly reports to illustrate the operating results during all-spiral and all-cyclone periods. The periods chosen had very nearly the same head assays of copper and non-sulfide copper.

The copper recoveries are about the same: 80.2 from cyclone overflow and 79.9 pct from spiral product. The higher grade of copper concentrate for the later period is a direct function of the fineness of the regrind product, which in turn was influenced by the finer rougher concentrates obtained from the cyclone sized feed.

More detailed comparative data is available from data taken during plant tests such as the data for March 1957, summarized in Table II and amplified in Figures 1, 2, and 3. Note that the increased revovery of copper in the minus 200 mesh fraction is offset by a reduced revovery of the copper in the plus 200 mesh fraction, so that the net recovery from the cyclone product is only slightly greater than that from the spiral prepared feed. The lower copper and sulfur distribution in the coarser size fractions, and the higher soluble, indicate that the copper in the large particles tends to be more locked with gangue - the more free chalcocite and chalcopyrite having been dropped into the underflow and subjected to additional grinding. Note also that the sulfur in the plus 65 mesh fractions of the flotation feed from the cyclone is insufficient to satisfy the iron in these fractions as sulfides. The excess iron is probably in the form of the oxides and silicates of lower specific gravity. The bar chart of Figure 1 and the curves of Figure 2 illustrate the difference in the copper and sulfur assays and distributions in the spiral and cyclone overflows. The major part of the shift in concentration is in the sand sizes; there is no undue enrichment of the slimes.

These observations agree with the original expectation that the cyclones would not improve selectivity between pyrite and chalcocite, nor directly improve the recovery of copper in the rougher circuit. On an ore in which the copper and the pyrite are intimately associated the cyclone would show metallurgical benefit because it would return even fine aggregate sulphide particles to the mill. The association of pyrite with chalcocite in the Silver Bell ores is not at all intimate, whereas the association of the copper minerals with the siliceous gangue is more intimate. The action of the cyclone is to exaggerate small differences in specific gravity, and therefore the cyclone tends to reject into its spigot product a larger amount of sulphides and a lesser amount of gangue than a mechanical classifier. The result at Silver Bell is that a lot of pyrite which from the metallurgical point of view has reached finished size, is returned to the ball mill instead of being finished in the overflow, while on the contrary, gangue which still contains copper escapes in the overflow when it might have been rejected in a mechanical classifier. The effect, therefore, is to produce a flotation pulp in which there is more middling of copper and gangue, and more pyrite ground considerably beyond the size of liberation. Since the pyrite is ground finer than necessary its recovery in the rougher concentrate would tend to be more favored. The higher recovery of iron from the cyclone overflow (see Figure 3) illustrates the operation of this principle.

Although the use of cyclones as primary classifiers at Silver Bell has not been the basis of metallurgical improvements, neither has it been detrimental. The decision to convert from spiral classifiers was based almost entirely on the anticipated savings in maintenance and down time, rather than on improved metallurgy.

PLANT TESTS

During the elapsed time between the initial test work on Section 4 and the final conversion of the last section, a span of about two years, comparisons were made among the following classifier types or combinations: Four 20 inch cyclones, two 30 inch cyclones, one 30 inch cyclone, four 20 inch cyclones plus the spiral classifier, three 20 inch cyclones plus spiral, and of course spirals alone. Various modifications were made to the cyclones, such as changing vortex finders and apex valves, varying the length of the cyclone cylinder, adding hydraulic water to the underflow, and varying the input pressure.

Testing was started on July 10, 1956. As previously mentioned, the first trials were with a two-stage scheme in which the overflow of the spiral classifier became the feed to the cyclone, thus eliminating danger of pump damage or excessive pump wear due to the coarser ore sizes and tramp steel. During the first several months, operation of the test section was somewhat erratic due to the frequent shutdowns for changes in pump speeds and classifier settings but the conclusion was soon reached that the cyclones would produce acceptable classification at as low as 3 psi inlet pressure and that the circuit was satisfactory from an operating viewpoint. Comparative data taken over a period of even operation proved that the test section was doing as well as the rest of the mill, reaching a slightly leaner rougher concentrate with slightly higher recovery.

Early test data indicated that increased flotation time, obtained by increasing the solids-liquid ratio, did not result in a lower copper tailing. Tests with densities ranging from 20 to 32 per cent showed that increase in tailing losses was experienced when the flotation feed density was over 24 per cent. As a result of these tests, the per cent solids was held to 24 rather than the 20 per cent used until then, thus providing a small increase in flotation time. This experience agreed with contemporary laboratory tests.

The last few months of 1956 were devoted to the accumulation of operating, maintenance and metallurgical data using cyclones to produce a finer grind than standard. It was felt that previous fine grinding tests were inconclusive due to excessive dilution of the flotation feed pulp (18 per cent solids), necessary for obtaining the finer overflow from the spiral classifiers. The cyclones were capable of producing a sharper classification at a higher density (26 to 28 per cent solids) than previously obtained, but the benefit was not as great as might be expected. During a period of 45 days, the tailing loss of copper was about 0.3 pounds less per ton of ore milled than the check sections, but the grinding rate was reduced from 84 to 67 tons per hour. The percentage on 65 mesh for the test section was 12.3 and the remainder of the mill was 21.3; the corresponding percentages through 200 mesh were 54.5 and 46.6 per cent, respectively.

Early in January 1957, a fourth 20 inch cyclone was added to the original three. We found, however, that there was no practical benefit gained by using this additional cyclone in the two-stage classification circuit.

In February, the ten inch Linatex pump was dismantled for inspection and repair. By this time, it had pumped spiral classifier overflow at 58 per cent solids for 180 days, handling over one million tons of solids, including circulating loads but not the coarse spiral sands. The impellar was found in good condition with about half of its useful life remaining. Minor repairs were necessary on the suction cover, and gland seals were replaced. No parts were replaced on the cyclones except for the expansion type apex valves.

In view of the fact that maintenance costs experienced with the cyclones were less than half of those with the spirals, we believed that a substantial saving might be realized by going to an all-cyclone classifier circuit. At this time, the spiral classifier of the test section was by-passed. The ten inch Linatex pump delivered the ball mill discharge at 60 per cent solids to the four cyclones, which overflowed the finished material at 33-35 per cent solids and underflowed the oversize at 78 per cent solids. The circulating load tonnage was maintained equal to that returned by the spirals (400-500 per cent of new feed).

After several months operation to confirm the dependability of the all-cyclone circuit, the spiral classifier was removed from the test section. This conversion was finished in October, 15 and one-half months after the start of plant testing.

A month later, No. 3 Section was equipped with a single D-30L, large inlet cyclone to be used in the transition from spiral classifiers to cyclones. This unit, mounted at the rear of the ball mill-classifier space, facilitated the removal of the spiral classifier and installation of the four 20 inch cyclones without interrupting production.

Data provided during the transition period permitted comparisons to be made among the three types: One 30 inch, four 20 inch and spirals. The single 30 inch cyclone was found to operate at a higher inlet pressure, and overflow at a higher density for a given size of classification than the four 20's. The overflow solids consistently showed more tramp oversize for a given weight on 65 mesh.

Later, No. 3 Section was equipped with two standard D-30L cyclones. As the results produced by these cyclones were so close to those of the other sections, the test was extended for a period of eight months, January to August 1958, at which time the section was converted to the four 20 inch cyclones, thus completing the mill conversion. Comparisons of size distributions from grinding-classifier circuits during this test period are given in Table III.

The two 30 inch cyclones produced a screen sizing equivalent to that of the 20's using slightly more horsepower to elevate the somewhat more dilute pulp to a higher point. The wearing parts of the 20 inch cyclones have lasted about twice as long as those of the 30's. The advantages of the 30 inch cyclones, less floor space and better ability to handle overloads of abnormally coarse ball mill discharge were insufficient reason to substitute them for the 20's which were more flexible, required less inlet pressure, and were more easily relined.

AUTOMATIC CONTROLS

Recently, No. 2 Grinding Section was equipped with automatic controls to keep the size distribution of the flotation feed approximately constant in the face of changing ore hardness. After investigation of three other systems of control, the present "Float and Density Control System" was found to be most applicable.

The systems rejected included a vacuum control, similar to that of Marmora 2/. It was found that our coarser overflow product, multiple cyclones, and in particular lower inlet pressure all contributed to the impracticality of this method at Silver Bell. Severe fluctuations of vacuum and unsatisfactory correlation with grind forced the abandonment of this method.

The second method of approach was to use the angle of the underflow spray as a measure of the amount of material in the cyclone underflow. Measuring the interruption of a light beam by photoelectric cells was attempted but was soon set aside because of equipment difficulties. Complete tests were never run.

Following the presentation of data by Littlewood here two years ago, a recording watt-hour meter was installed to follow the power variations at two points in the grinding circuit. The power trace on the cyclone feed pump motor was soon shown not to be a reliable parameter for circuit control purposes. The trace on the ball mill motor seemed to follow mill loading conditions and a system of recording and control was set up. It was soon found that changes in power consumption were often due to changes in outside factors having no relation to the relative hardness or sizing of the ore feed. For example, the starting or stopping of the primary crushing motor among others, was sufficient to displace the ball mill power consumption curve about as much as a normal change in ore. Although changes in dilution within the mill and of source the addition of balls were readily apparent on the power graph, normal changes in grind and amount of circulating load produced too small a power variation to be used as a control directive.

The system next tried, and still in use, is based on the density of the cyclone overflow slurry, using equipment similar to that described by Rachlin 6/. Any increase or decrease of density from a reference point is used to control the rate of feed to the circuit according to the sequence described and illustrated in Figure 4.

Although no direct control of dilution water is exercised by the instruments, excellent control is maintained on the density, and consequently on the size distribution of the finished product. Variations in feed rate from 60 to 105 tons/hour have been handled with ease. Graphical records of finished product density and ore feed tonnage have reduced maintenance and attendance to that of periodical inspection and checking. Manpower requirements are certainly less with this control system, although other factors have prevented a reduction in operating crew.

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TABLE I

COMPARISON OF DATA FROM ALL SPIRAL AND ALL CYCLONE PERIODS

MONTH	2/56	2/59
FEED	**SPIRAL**	**CYCLONE**
% Cu	0.91	0.90
% N.S. Cu	0.14	0.13
% Fe	3.0	2.4
TAILS		
% Cu	0.22	0.21
% S. Cu	0.10	0.10
CONCT.		
% Cu	26.66	31.28
% Fe	24.4	25.3
% Insol.	2.6	4.9
% Revov. Cu	79.9	80.2
" S. Cu	84.4	87.3
% Iron Rejected	77.9	75.7
Ratio of Conct.	36.8	43.3
Grind Rate TPH	81.7	80.9
Grind-KWH/Ton	8.2	8.3
Class. O'Flow +65	16.1	20.4
" " -200	47.9	48.0
Conct. - 325	82.0	94.0
Class. O'Flow % Solids	20	35
Flot. Feed % Solids	20	23
B.M.Disch. % Solids	72	75

TABLE II

COMPARISON OF DATA FROM SPIRAL AND CYCLONE ROUGHER FLOTATION SECTIONS

MARCH 1957

			SPIRAL	CYCLONE
% Solids O'Flow			22.8	31.9
% Solids Flot. Feed			22.8	24.2
T.P.H.			83.6	86.4
Ratio of Conct.			14.3	12.5
FLOTATION FEED				
	% Cu	+65	0.66	0.38
		-200	1.29	1.58
	% S	+65	2.43	0.65
		-200	1.89	2.72
	% Fe	+65	2.94	1.31
		-200	2.96	3.35
	% Insol	+65	89.1	93.2
Ro. Tails % Cu			1.10	1.18
Ro. Conc. % Cu			12.2	11.5
% CU RECOVERY				
		+65	54.9	42.3
	-65	+200	72.5	69.4
		-200	78.4	80.8
		Total	77.6	78.9
% FE REJECTION				
		+65	44.3	63.8
	-65	+200	34.9	29.9
		-200	59.1	52.0
		Total	50.6	44.3
% DIST. CU				
	Feed	+65 m.	11.4	6.2
	Tail	+65 m.	18.1	17.7
	Feed	-200 m.	58.7	67.2
	Tail	-200 m.	59.1	61.2

TABLE III

SIZE DISTRIBUTION FOR ALL MILL-CLASSIFIER CIRCUITS USING SPIRAL
FOUR 20 INCH CYCLONES AND TWO 30 INCH CYCLONES

CUMULATIVE PER CENT RETAINED

	BALL MILL FEED			BALL MILL DISCHARGE			CLASS SANDS			CLASS O'FLOW		
THRU ON	SPIRAL	4-20's	2-30's	SPIRAL	4-20's	2-30's	SPIRAL	4-20's	2-30's	SPIRAL	4-20's	2-30's
- 4	47.6	92.3	49.5	5.7	3.8	10.2	7.1	5.2	8.5			
4 8	65.3	61.4	66.3	10.4	9.0	18.0	12.6	11.4	17.8			
8 14	76.4	73.7	76.8	17.7	16.5	26.7	21.8	20.5	22.9			
14 28	82.4	81.7	82.6	32.2	29.9	39.0	42.1	36.9	46.3			
28 48	86.7	86.6	86.9	60.0	57.3	58.5	79.1	69.9	73.1	5.8	1.1	7.1
48 65	88.5	88.3	88.6	70.0	67.5	67.1	88.2	79.9	81.7	19.5	16.0	19.0
65 100	90.1	90.1	90.2	76.2	76.0	73.3	92.7	87.0	86.8	31.0	30.3	30.4
100 150	91.6	91.4	91.6	80.8	80.9	78.2	95.1	90.8	90.3	41.3	40.5	41.2
150 200	92.5	92.2	92.4	83.2	83.5	80.7	96.0	92.5	91.7	47.8	48.9	47.9
200	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
-200	7.5	7.8	7.6	16.8	16.5	19.3	4.0	7.5	8.3	52.2	51.1	52.1
% Solids				72.6	72.0	72.5	-	75.0	76.0	24.0	31.0	33.8
Feed Rate (tph)	89.8	99.8	93.9									
% Circulating Load							275	400	320			

SUMMARY

At the time of this writing, the Silver Bell Concentrator has been operating for a little over a year with cyclones serving as the only primary classifiers. The cyclones have performed their job satisfactorily, producing the desired flotation feed without undue problems of maintenance or operation.

Operating costs have been less than half of those experienced with the spiral classifiers formerly used and operation has been greatly simplified. Use of the cyclones has permitted a satisfactory method of automatic control of finished product size in the face of changing ore hardness.

The physical difference between flotation feed produced by the cyclones has not been detrimental in the rougher flotation circuit and is presumed to be of benefit to the scavenging and cleaning circuits.

In summary then, our conversion to cyclone classification has resulted in improved process control and has provided us with another tool in the job of lowering costs.

ACKNOWLEDGEMENTS

Credit for the successful operation described must be given to the operating and test personnel at Silver Bell and to the equipment manufacturers represented, for their products and their kind assistance.

The author thanks the management of the American Smelting and Refining Company for permission to give this paper on the use of cyclones in the grinding circuits at the Silver Bell Unit.

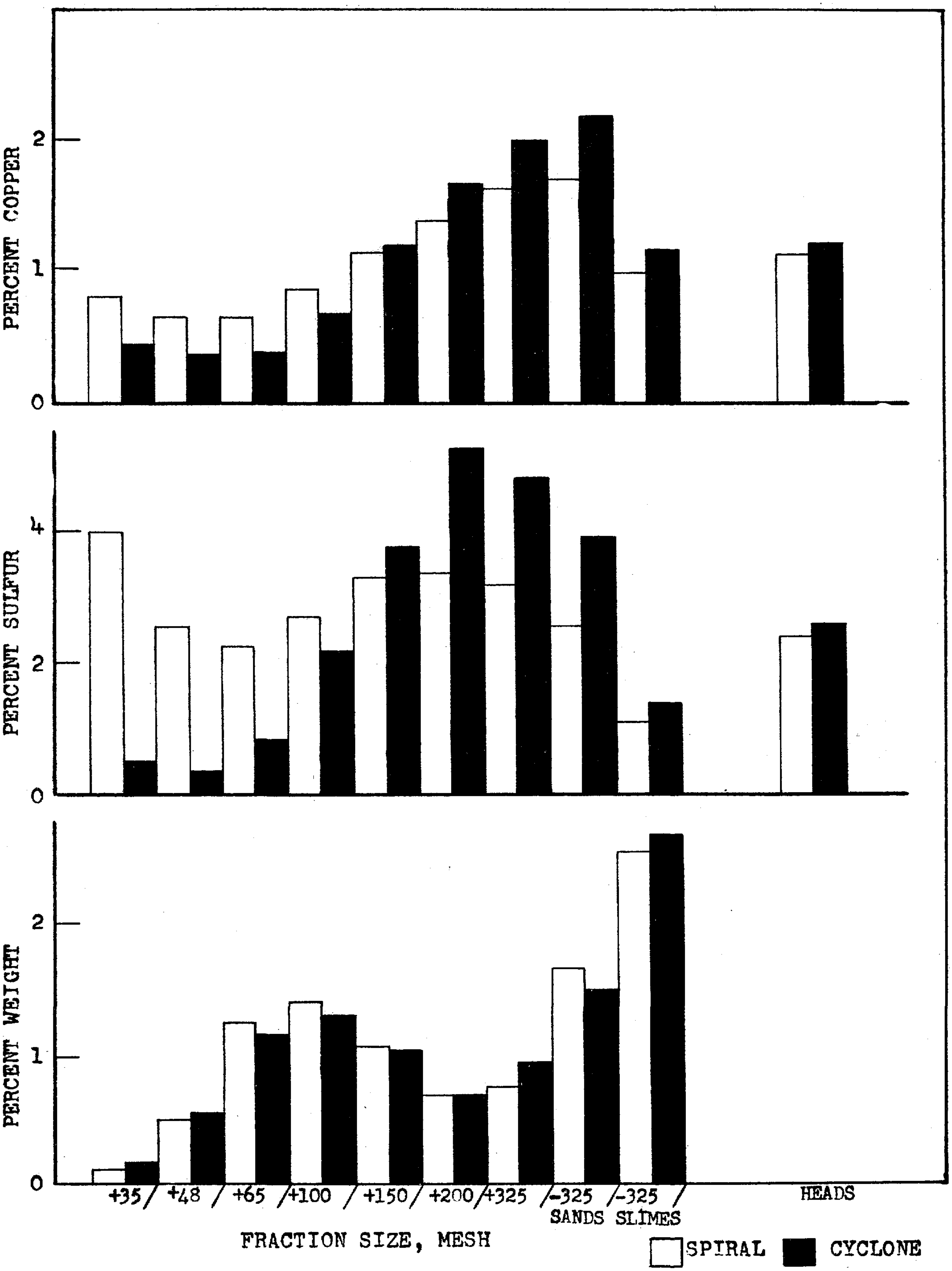

FIG 1 - COPPER AND SULFUR ANALYSES OF SPIRAL AND CYCLONE OVERFLOWS

SB 3/57

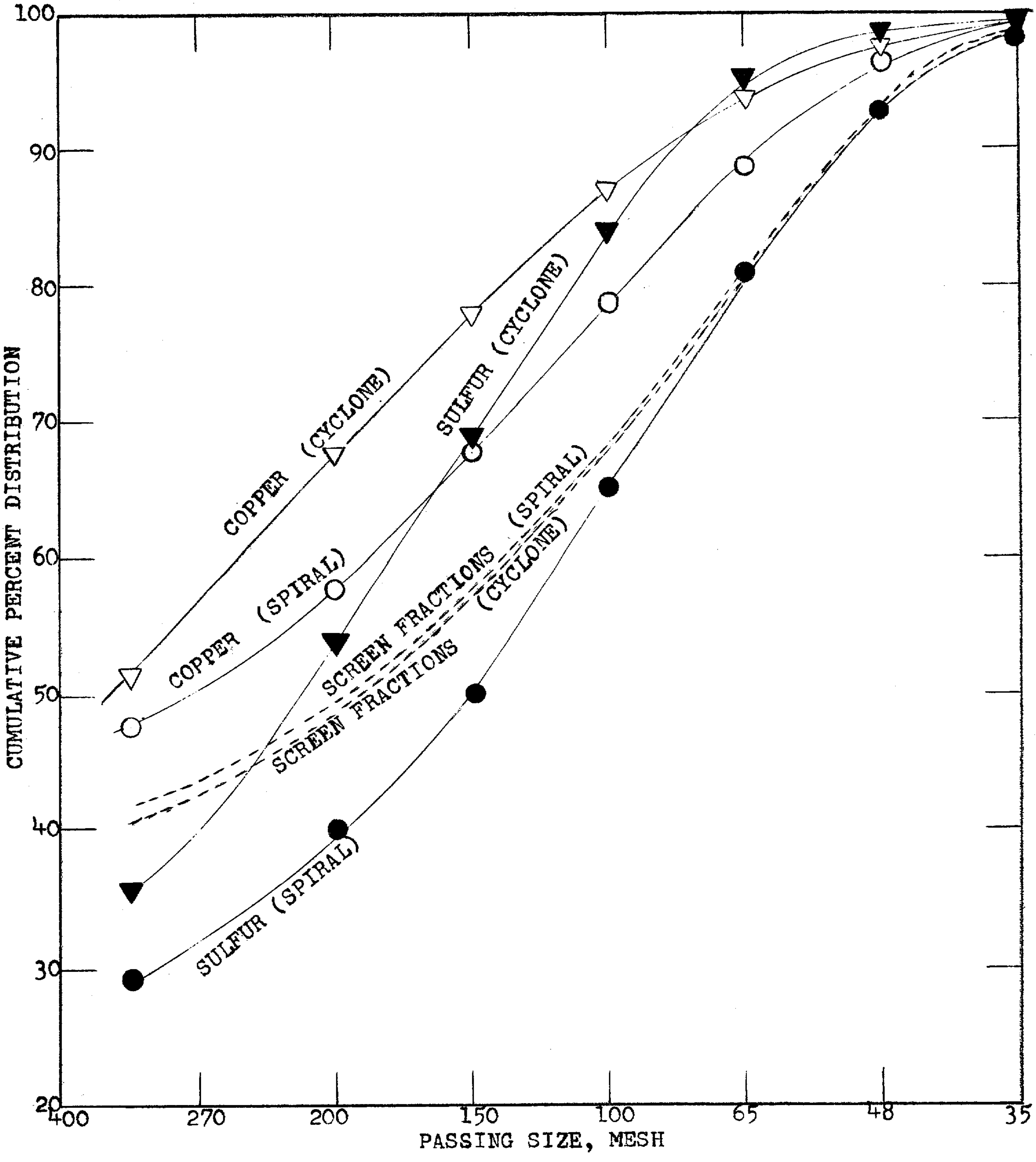

FIG 2 - CUMULATIVE PERCENT DISTRIBUTIONS IN CYCLONE AND SPIRAL OVERFLOWS.

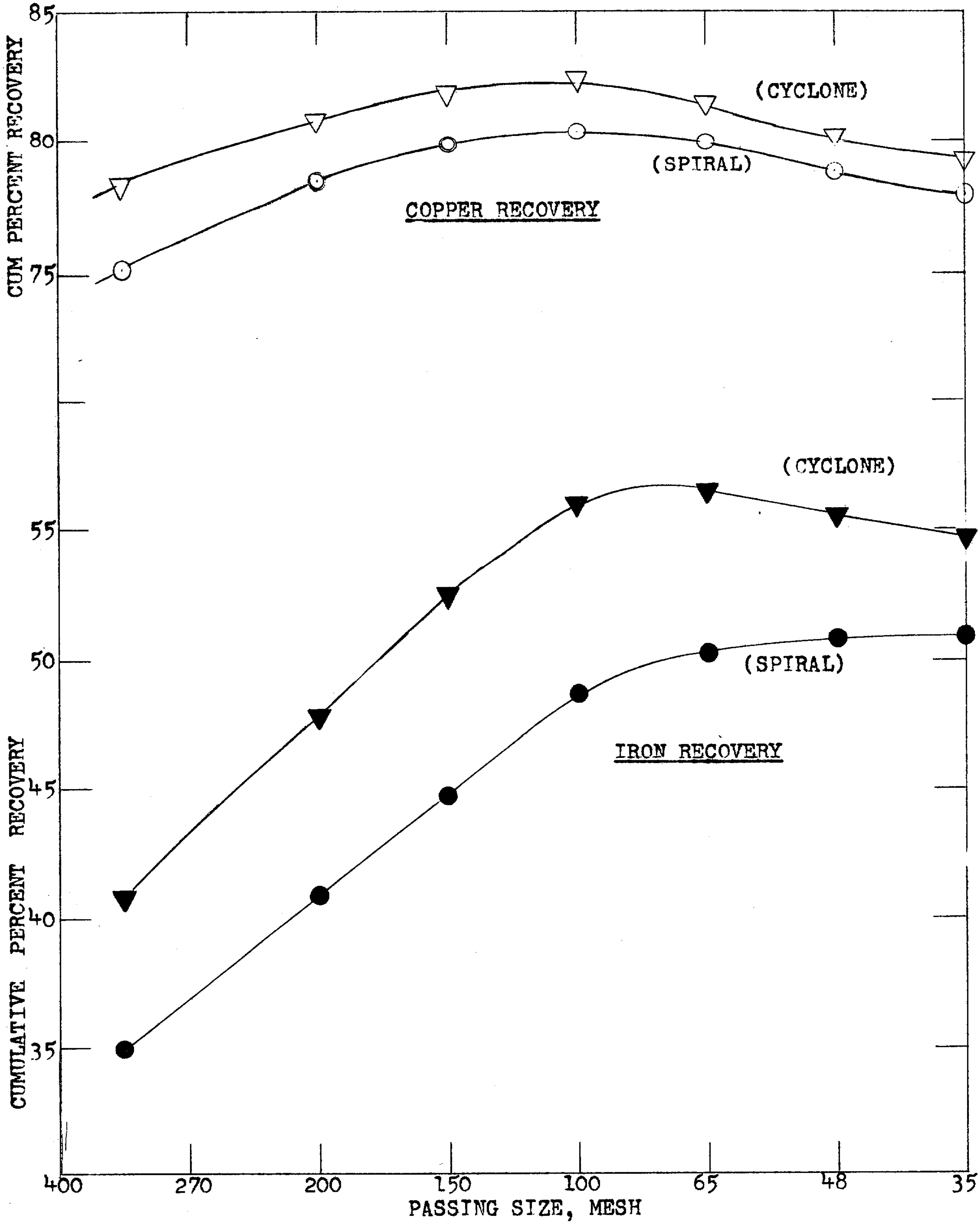

FIG 3 - CUMULATIVE RECOVERIES IN ROUGHER CONCENTRATES.

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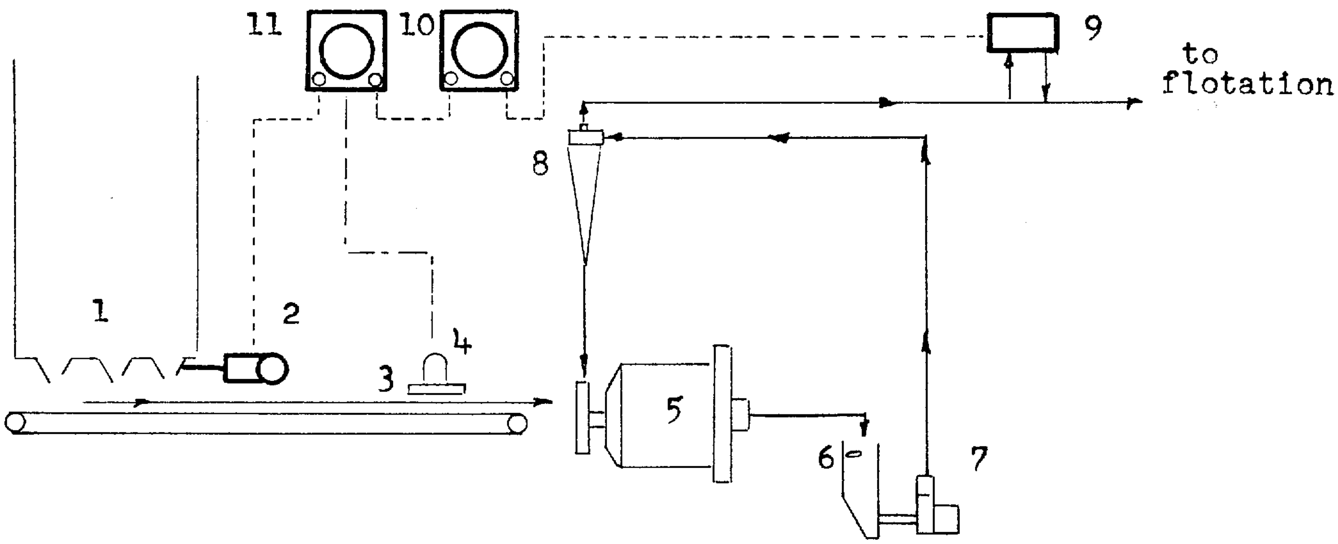

EQUIPMENT LIST

1 - Fine Ore Bin
2 - Actuator with Side-Mounted Moore Positioner (M-H)
3 - Conveyor Scale (F-M)
4 - Motion Transmitter, Electric (M-H)
5 - 10-1/2 x 12 - ft. Grate Ball Mill (A-C)
6 - Constant Level Float Valve in Pump Sump
7 - 10 x 10-in. Pump (Linatex)
8 - Four 20-in. cyclones (Krebs)
9 - Density Receiver and Differential Converter (M-H)
10 - Single Pen Recorder-Controller (M-H)
11 - Single Pen Pneumatic-Motion Receiver - Controller-Recorder (M-H)

PRINCIPLE OF OPERATION

Entry of softer ore to grinding circuit results in decrease of circulating load so that the volume of ball mill discharge decreases, thus requiring water addition by the float valve to maintain constant volume in pump sump (6). This results in lower pulp density in cyclone overflow causing displacer of density-receiver unit (9) to sink, thus, via the differential converter, signalling density recorder-controller (10) which moves index of tonnage recorder-controller (11) causing actuator (2) to release more ore. Tonnage is detected by conveyor scale (3) and transmitted by motion transmitter (4) to tonnage receiver recorder-controller. Increased tonnage restores circulating load which cancels sequence.

Hard ore results in higher circulating load, higher overflow density, and subsequent decrease in tonnage. Manual over-riding is possible at all times.

FIG. 4 - AUTOMATIC DENSITY CONTROL OF CYCLONE OVERFLOW.
Silver Bell 10/59