TUNGSTEN MINING

AT THE

GETCHELL MINE

A THESIS

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The Osgood Range rose nearly north and south and the contact of the stock with the sedimentary rocks dips generally 40 degrees to the east on
TUNGSTEN MINING AT THE GETCHELL MINE

I. Introduction

The Getchell Mine is located in northeastern Humboldt County, Nevada, on the eastern slope of the Osgood Mountains in the Potosi Mining District. Golconda is the post office address and is 26 miles southwest of the Getchell Mine while Red House is the nearest rail point and is 16 miles south of the mine. By automobile the Getchell Mine may be reached by turning off U. S. Highway 40 two miles east of Golconda onto Nevada State Highway 16 and driving a distance of 24 miles, the first fourteen miles on paved road and the remaining ten miles on an improved dirt road.

In January 1951 the Getchell Mine turned from a producing gold mine treating oxide and sulphide gold ores to the mining and milling of tungsten ores occurring in the district. The tungsten mineral is principally scheelite (calcium tungstate) which occurs in the tactite deposits formed by an intrusive stock of granodiorite that has invaded sedimentary rocks of Triassic age. The granodiorite stock is roughly in the shape of a figure eight about six miles in length, and some two miles wide, and narrowing at the center to less than 300 yards. See Plate I for the geologic map of the Osgood Range showing the location of the principal tactite deposits with the names of the open pits and underground mines on each known deposit in the district. The highest peaks in the range rise to a height of some 3,400 feet above the valley floor on either side of the range.

The Osgood Range runs nearly north and south and the contact of the stock with the sedimentary rocks dips generally 40 degrees to the east on
PLATE I
Osgood Range Intrusive - Potosi Mining District
Scale 1" - 4000'

Wm. Newman
the eastern slope. The contact is nearly vertical on the western slope as exposed in the open pit known as the Alpine Pit. The contact is also conformable to the attitude of the sedimentary series. The tactite deposits were formed around the periphery of the granodiorite stock by the metamorphism of the limestone adjacent to the stock. Where the conditions were favorable the limestone was altered to tactite. A favorable condition for the formation of tactite seems to include the proper temperature and pressure of the invading granodiorite and the proper chemical composition of the limestone or other sedimentary rock. As the invading stock is nearly homogeneous as regards chemical composition and can therefore be considered a constant then the only remaining variable is the limestone. The fact that tactite does not occur as a continuous deposit around the periphery of the granodiorite stock but only in isolated deposits seem to bear out the fact that the limestone is the variable. The economic tactite deposits occur in the Osgood Range where the limestone is not highly siliceous. In addition, another very favorable condition is an irregularity in the granodiorite - limestone contact where an abrupt change of strike of the contact may project the limestone into the granodiorite. This condition is considered favorable enough from the history of the known ore bodies that when found as a surface indication warrants further investigation by diamond drilling or cross-cutting or by drifting as the case may be best suited to topographic conditions.

In other places along the periphery of the stock where the conditions were not favorable the limestone was altered to a wollastonite and makes a frozen contact with the granodiorite stock. The argillite that is interbedded with the limestone is altered to a shale or hornfels.

A fault zone which is the result of normal faulting is present along the northeastern slope of the Osgood Range. This fault zone averages about
thirty feet in width and is exposed by the open pit and underground mining for a distance of one mile. This fault, known as the Getchell fault, is later than the granodiorite intrusion and contains the gold ore deposits which are separate and unrelated to the tungsten deposits of tactite.

The tactite ore bodies contain typical metamorphic minerals and are chiefly garnet. Other minerals in order of prominence of occurrence are calcite, quartz, epidote, chalcopyrite, pyrite, powellite, scheelite, and molybdenite. The garnets are varying in size and are usually in crystal form. The calcite is crystalline and never clear but a milky color. The quartz is usually amorphous but is sometimes in clear crystals when found in seams or vugs. One known quartz crystal about six inches long has been found in the underground ore from the Kirby mine with the typical quartz termination on both ends. The epidote is crystalline as is the chalcopyrite, pyrite, and molybdenite. Scheelite crystals have been found in high grade pockets or seams in both the underground and open pit mines. Most scheelite in the tactite is in small crystals and under the Mineralight appears a blue white color. Powellite (calcium molybdate and calcium tungstate) is usually in fine particles and, depending upon the ratio of scheelite to molybdenum, lamps from an almost cream color, when the molybdenum is less, to nearly a canary yellow when there are equal amounts of scheelite and molybdenum.

The production from the various tactite deposits located around the periphery of the granodiorite stock of the Osgood Range from December 1950 to and including March 1955 is given in the table on the next page.
1. Alpine Pit 43,700 EWT
2. Moly Pit 12,650 EWT
3. Kirby Pit 34,550 EWT
4. Pacific Pit 208,100 EWT
5. Tip Top Pit 32,300 EWT
6. Top Row Pit 2,820 EWT
7. Riley Extension Pit 27,810 EWT
8. Granite Creek Mine 164,980 WT
9. Riley Extension Mine 144,580 WT
10. Moly Mine 5,400 WT
11. Pacific Mine 2,300 WT

**Total** 679,190 EWT

The tonnages for the open pits are estimated wet tons taken from the daily truck tally. The tonnages for the underground mines are weighed wet tons as all underground ore goes over the truck scales at the mill before going to the primary crusher.

The tonnages are broken down into a total of 361,930 estimated wet tons for the open pit mining. Also during this period the underground mines produced the balance of 317,260 wet tons.

**II Development of the Ore Deposits**

All of the underground mines and the open pits have been developed from tactite outcrops. Open pit mining is usually begun on the outcrop and as the deposit is exposed then work is begun to develop the ore at depth either by cross-cutting, drifting, raises, shafts, or winzes. The diamond drill is also used to locate ore deposits or prove structure in order to facilitate the development of the ore body for mining. Areas along
the contact where there is little or no tactite outcrop but where there is a considerable change in strike of the contact are prospected either by diamond drill or from underground workings. The Granite Creek mine is a deposit of this type and the ore has been found to be sixty feet wide in the underground stope where the strike of the granodiorite contact makes a right angle swing around the limestone.

The problem of beginning an open pit operation on a tactite deposit in this district is fairly easy to solve. On the east slope of the Osgood Range where the principal tactite deposits are found the sedimentary rocks lie to the east or the down hill side of the ore along the contact. The granodiorite is above the tactite or to the west and as a result the weathering and subsequent decomposition of the granodiorite has made an overburden of good sand on the hillsides which makes for easy road construction and very inexpensive as very little drilling and blasting is required. All D-8 Caterpillar crawler type tractors here are equipped with angle blades that are particularly well suited to road building on side hills.

A rough road on a ten per cent grade is first put in to a tactite deposit in order to move a 3/4 yd shovel to the ore to begin stripping. Benches are usually held to about fifteen feet in depth for this size shovel in order to keep the bank clean of loose rock at all times. When the loose overburden is removed and solid limestone, that overlays the tactite deposits forming the hanging wall, is exposed it is drilled by a wagon drill and blasted and the waste again removed from the ore. The tactite is usually solid enough so that it will also require drilling and blasting. The ore is then hauled to the mill where an entire day's haul may be run over an automatic sampler to obtain a better sample than is
possible by hand sampling or by lamping at night.

This process of benching by open pit mining will continue on the ore body until an ore - waste ratio of 1 part ore to 10 parts waste is reached, then the ore can be more economically mined by underground methods. The grade of the ore also has some consideration but generally the rule is to hold the open pit operations under the ten to one ratio.

The underground mining is carried on by standard methods of shrinkage stoping. There is a slight exception to the standard method when the ore body dips less than 40 degrees; then a slusher is employed to pull the ore from the stopes to the chutes. The ore bodies that are extensive enough are developed by drifts, cross-cuts, and raises into blocks roughly 100 feet in length and 150 feet in height on the dip of the ore. Chutes are spaced from 25 to 30 feet on the haulage level and are connected together in the stopes with a pillar between each chute. Manway raises at each end of the ore blocks provide entry to the stopes and service for air and water lines and supplies.

When driving the haulage drift beneath an underground tactite deposit that dips less than 40 degrees it is the practice to drive the drift wide enough to permit a double track, one of which will be for the ore train to pass directly under the flat chutes. The other track will be for a slusher hoist mounted on an old mine car chassis to be moved from chute to chute. This method allows one slusher to do the work of several slashers and each working chute has its own scraper.

The sedimentary rock in the form of limestone that constitutes the hanging wall is very hard and is ideal for shrinkage stoping. It is also an economic as well as geologic hanging wall as there are no values that project into the limestone. The only place that the sedimentary rock is
troublesome is where a water course is encountered and it can be dangerous unless proper pillars are provided. The granodiorite that forms the footwall is also economic as well as geologic. The limestone hanging wall and the granodiorite footwall have a practical advantage in that both are very readily distinguished from the garnet brown of the tactite ore deposits. These walls are fairly regular, and have a tendency to break cleanly. Even when drilling, the cuttings from the return drill water show the miner when he has drilled out of the ore zone.

III Open Pit Mining.

In the open pit mining of the tactite ore deposits around the granodiorite stock the wagon drill has aided in the solving of a very difficult drilling job in both ore and waste. One Ingersoll-Rand FM-2 and a later model FM-3 wagon drill both equipped with Ingersoll-Rand DA35 drifters chucked for 1 1/8" round lug steel do all the drilling in the open pits. Four changes of steel are used to drill all holes to a depth of 21 feet wherever possible. The first steel or starter is 6' 6", the second steel is 11' 6", the third steel is 16' 6", and the fourth steel is 21' 0". The slide on the wagon drill is eight feet long and has the advantage of permitting steel of one change that may be broken to be made up into the next smaller change usually with only one forging of a thread or shank depending on which end is broken and without regard to exact measurement of steel change.

Detachable bits of the Ingersoll-Rand Company are used with the 118 type thread and are four point bits with tungsten carbide inserts. The starter steel has a 2 3/8" bit, the second a 2 1/4" bit, the third a 2" bit, and the fourth has a used 2" bit. There is no problem of one bit not following another even if they were manufactured with less gauge
difference as there seems to be very little gauge loss of the bits. In fact one bit could probably be used to drill the entire hole if it were changed from steel to steel. The reason for starting with such a large size bit is to allow the hose on the pneumatic powder loader to pass into the hole in order that the bag type powder can be placed as near the bottom of the hole as possible for the flat holes. On vertical holes the hole can be started with a 2 1/4" bit and followed by an older 2 1/4" bit then followed by a 2" bit on the third and using an older 2" bit for the finisher or fourth steel as the bag powder is poured in the drill holes without the use of the pneumatic loader.

These Ingersoll-Rand type 118 tungsten carbide bits as used on the wagon drills in the open pits have proven very satisfactory. A record of the drilling was made until the excellent results were established and then the records were abandoned in favor of more important records that were required on the underground drilling which will be given in full in a later section. It is a matter of record that the type 118 carbide bit used on the wagon drill is capable of drilling upwards of 2,000 feet of hole per bit under the most adverse conditions of hard wollastonite lime of the hanging wall and of the tactite ore deposits. This results in an average cost of approximately 14 per foot of hole drilled. As the steel cost is also nearly negligible then the total drilling costs resolve into a labor cost and the cost of compressing the drilling air by portable compressors.

In drilling the ore and waste rock in the open pits where the 1 1/2 yd diesel power shovels are operating the benching is carried on in twenty foot benches. As a general rule horizontal holes are drilled in preference to vertical holes as the drilling is done more nearly at right angles to the bedding rather than with the bedding. Also when a new bench
is made the roads to the old bench above are cut out and it would require considerable extra work to get the drilling equipment up on the old bench for the vertical holes. In order to drill a minimum of holes, the flat holes are spaced at intervals of from twelve to fifteen feet. Each hole then has from 500 to 600 tons of ground on it and with proper springing each hole can then be loaded with enough powder to break the ground with a minimum of secondary blasting. A ratio of from 1/4 to 1/2 pounds of 40 per cent bag powder to a ton of ore or waste is sufficient to produce excellent primary breaking. If the drilling has been done in relatively soft rock, such as near the surface where it may be weathered, then 1/4 pound of bag powder to a ton of rock may be used. If the formation is solid rock and has very few slips or seams then the 1/2 pound per ton ration would be used. The driller can usually judge the character of the ground while drilling and can determine the proper ratio for blasting.

The success of the drilling and blasting operation depends upon the proper springing of the drill holes. If the holes are in very fractured ground and cannot be properly sprung then extra holes have to be drilled in order that the proper ratio of powder may be put into the block of ground to be broken. Springing is done with 60 per cent semigelatin powder and is usually with the 1 1/3" x 8" stick. Stick sizes of 1 1/2" x 8" and 2" x 8" are also available in case the hole is free after the first spring in order to save time where the sticks are loaded one at a time in the flat holes. An 1 1/2" stick is equal to about 2 sticks of the 1 1/3" size and a 2" stick is equal to about 4 sticks of the 1 1/3" size. The first spring is usually with about four or five 1 1/3" x 8" sticks and as the flat holes are inclined slightly down, in order to keep the pit floor level, water can be used for stemming. The
holes are usually sprung by the helper on the wagon drill as the drill is advanced along the face and therefore the first spring is usually completed when the drilling is done. This also allows the rock to cool sufficiently before a second spring. The use of water for stemming shortens this cooling period somewhat but in any case one should allow at least an hour cooling period in solid rock formations for safety. The second spring will consist of from 20 to 25 sticks and by measurement on the loading stick if this does not provide a satisfactory chamber or pocket a third spring of 80 to 100 sticks may be required. These figures are for the average formations encountered in this area and there have been occasions in solid rock where the third spring has consisted of a case of powder or approximately 160 sticks of the 60 percent 1 1/3" x 3" stick powder.

After the cooling period the holes are loaded with a 40 percent bag powder. The loading is done with a pneumatic loader with a two bag capacity. The loader has a pressure regulator attached to allow only a maximum pressure of 25 pounds into the loader and only one 12 1/2 pound bag is loaded at one time. The loader is well grounded by wetting the ground around the legs of the loader. Springing or loading of the drill holes is never done on days when a thunderstorm is in the vicinity because of the danger of lightning. The loader has a 30 foot rubber hose from the loader to a 15 foot brass tube of 1" outside diameter and 7/8" inside diameter which is put in the drill hole as near the chamber in the bottom of the hole as possible. The use of the larger starter bit permits the rubber hose to also pass inside the drill hole.

The bag powder is primed with several sticks of 60 percent powder using an electric cap with a 24 foot copper wire lead. Two sizes of empty cartridges are on hand to be filled with loose, dry decomposed granite sand
for use as stemming bags. The electric leads are connected in series and are in turn connected to a 500 foot duplex blasting wire lead to a 50 cap plunger type blasting box. As each hole is loaded and stemmed the lead wires are tested for broken wires with a blasting galvanometer and the circuit is again tested after all leads are connected in series and before connecting to the 500 foot duplex blasting wire. A 6 volt wet cell battery on the pickup truck has been occasionally used for firing the holes but has resulted in misses. The plunger type blasting box kept in good condition, free from dust and moisture, is the most satisfactory method of firing the drill holes in the open pits.

Secondary blasting is greatly reduced by the installation of a 42" x 48" jaw crusher in the mill for primary crushing. Any boulders that will pass through the 1 1/2 yd. shovel bucket can be crushed in this crusher. As the pits are located some distance from the mill and the trucks are on at least an hour cycle there is ample time for mudcapping or plastering the large boulders. A 60 per cent semi-gelatin powder is used for plastering. The powder is removed from the wrappers and pressed on the boulders to be plastered preferably in a hollow or on a seam or crack near the center of the boulder. Here again the large size cartridges of powder save time in unwrapping and placing on the boulders. A four foot fused cap is placed in the powder charge and the powder is then covered by loose sand leaving the fuse exposed. The fuse is split with a knife to insure quick lighting or spitting with a twelve inch hot wire lighter. Even if there are only two fuses to be spit a hot wire lighter is used for safety.

The equipment for loading and hauling ore and waste in the open pits consists of a 1 1/2 yd. Northwest diesel powered shovel and usually
four Kenworth 150 - 200 hp diesel trucks with 14 cu. yd. boxes for each shovel if on ore haul and three trucks if stripping. The tail gates of the truck beds have been removed and a rock gate installed to permit hauling large boulders, especially waste, without secondary breakage.

A model D - 8 Caterpillar tractor equipped with an angle blade is employed in the pit for clean up of spill when the shovel is loading the trucks and for road building and benching when a new bench is begun. A model 12 Caterpillar motor patrol is constantly maintaining all haul roads from the open pits and the underground mines to the mill.

All shovel operators are issued a Minelight and as the back of the shovel provides a dark room they become fairly good at determining the ore. The operators have been on the job since the beginning of the tungsten program and so are very familiar with the ore deposit.

The ore trucks are loaded in the pit with as near an average of 15 tons per load as possible. This usually amounts to six buckets of broken ore with the 1 1/2 yd. shovel and the trucks are weighed once in a while to check the weights. The condition of the roads and the tire wear have contributed to this amount as an average load.

The average cost of hauling the ore is 24¢ per ton mile from the various open pits located on the mine property. At first this cost seems very high but it must be remembered that, as the pits are located from 8 to 10 miles from the mill, the shovel is standing idle after loading the ore trucks for an hour to an hour and a half until the trucks return for another load. Each truck can be loaded in less than five minutes so much of the shovel time is idle time and accounts for a high loading cost of 43¢ per ton where under ordinary circumstances a truck could be under the shovel at all times and a shoveling cost of from 10¢ to 15¢ could be obtained. Also it must be remembered that the road building by the D - 8
Caterpillar and the road maintenance by the motor patrol are reflected in this haulage cost.

When the 24¢ per ton-mile figure is broken down it will be found that the shoveling or loading cost is about 26 per cent of this total figure or a little over 6¢ per ton-mile. The actual haulage is about 65 per cent of the total or about 16¢ per ton-mile. The road maintenance accounts for the remaining 9 per cent or about 2¢ per ton-mile.

The stripping costs can only be estimated because the dozer pushes an undetermined volume of waste when stripping a new bench. The best estimate that can be made is from the truck load count and eliminating the dozer costs while stripping. This results in a cost of an estimated 40¢ per yd., or approximately 20¢ per ton. This figure is high but it must be remembered that much of the stripping is done under adverse conditions especially when starting new benches on the steep sidehill as the trucks have to do much backing into the shovel before a bench is wide enough for them to turn around.

IV Underground Mining.

In the underground mine a variety of rock drilling equipment, principally of Ingersoll-Rand manufacture, is in use. There is now a program underway to make some changes in the drilling equipment. These changes were proven economically necessary by an analysis of data on the performance and costs of drill bits, drill steel and powder, the results of which are tabulated in a later section of this paper. The tendency in the underground drilling now is to go to a smaller size rock drill machine using smaller drill steel and smaller rock drill bits in order to obtain an improved cost of rock drilling.
In the early stages of development of the underground tungsten properties, the column mounted Ingersoll-Rand DA-35 drifters with automatic feed were used to drive the drifts and cross-cuts. As the number of drifts and cross-cuts to be driven increased an Ingersoll-Rand Model DJB 2 two machine boom type jumbo mounting two DA-30 drifters using 1 1/8" round steel with Timken type H thread. Two changes were used with a 45" starter with an MCD 1 3/4" Timken carbide insert detachable bit and a second steel of 75" length with an MCD 1 5/8" carbide insert detachable bit. This jumbo is being replaced to a certain extent by the use of the new type J38A jackdrill with a 36" air feed leg by Ingersoll-Rand. This machine is very satisfactory for drilling in drifts and cross-cuts unless the rock is very hard then the column and bar or the jumbo is again used. The J38A jackdrill uses a 7/8" hexagon steel and has a starter of 30" overall with the Timken type F thread with an FCA 1 5/8" Timken carbide insert detachable bit. The second is a 54" overall steel with an FCA 1 1/2" Timken carbide bit. A third is 78" overall steel with an old 1 1/2" bit or a new FAC 1 3/8 carbide insert bit. As in the drilling in the open pit there is no problem of gauge loss and an old bit will follow a new one of the same size with ease.

Some of the large R58 stopers and the smaller R48 stopers are still in service in some stopes and raises but will be replaced with the smaller R38A stoper as quickly as possible. The R38A stoper consumes 93 cfm at 90 lbs compared to 170 cfm for the R58 and 132 cfm for the R48 stoper. This represents a definite saving in compressed air and permits more smaller machines to work on the same volume of compressed air. The ratio permits approximately three smaller machines to operate where two of the larger machines formerly operated. Another factor is that the smaller machines are operated by one miner where formerly each large machine had
a miner and a helper. This is a definite advantage as production can be stepped up with the same number of men employed.

The steel used in the R43 and R58 stopers is the 1" quarter octagon with Timken type H thread. The first steel is 30" overall with a NC
1 3/4" carbide insert bit, a second steel of 54" overall length with a NC 1 5/3" carbide insert bit, and a third steel 73" long with a MC 1 1/2"
carbide insert bit. The R36A stoper uses the same steel as the J36A jack-drill as both have the same type fronthead. The stoper uses the pear shaped collared shank as does the jackdrill but with the steel puller. This type of jackhammer front head without an anvil block seems to eliminate an oscillation effect of the anvil block which seems to be a contributing factor in breaking the carbide insert bit around the skirt or wringing as it is called. The larger proportion of bits wrung in the larger sizes used on the R58 and R43 stopers have been reduced in test work on another property by changing the front head and thus eliminating this oscillation effect which reduces bit breakage. This was a serious failure on the early types of tungsten carbide insert bits with thin side walls. Later carbide bits are manufactured with a heavier wing that does not have a fast breakaway but supplies a reinforcement down the skirt of the bit to the shoulder.

Compressors of the Ingersoll-Rand type XCB and Imperial type X with capacities of 1200 cfm to 1600 cfm at 100 lbs psi supply the compressed air for the underground mines.

In the Granite Creek mine which is the oldest of the tungsten mines in the district the regular ore deposit was mined by driving a cross-cut to the ore and running a drift along the strike on what is known as the 300 foot level. See Plate II for a plan map and Plate III
PLATE II-I
COMPOSITE MAP
GRANITE CREEK MINE
Scale 1" - 100'
Wm. Newman
for section of the Granite Creek Mine. Chutes with a radial gate and a straight gate were installed every 30 feet along the drift and connected in the stops leaving pillars between. The stops was served through a manway raise at either end of the stop and as the stop was more than 200 feet in length a cribbed raise was carried through the broken ore near the middle of the stop. This cribbed raise worked very well as it seemed flexible enough to give with the broken ore but care was taken not to pull the chutes too heavy on either side of this raise in order to leave some support around it and not allow it to get out of line.

Ore deposits recently developed at Granite Creek are being mined as non-contiguous bodies and are seldom over 50 feet in length and 10 to 12 feet in width. The manway raises were sometimes run in the waste in order to be able to work the stop properly. All mining is done by shrinkage stoping and has been a very economical method as no timber is required with the exception of the chutes. Pillars were left at intervals in the stops as a safety measure and extra pillars were left where the hanging wall limestone was fractured by slips or water courses.

In 1951 a 1700 foot drift was driven on the 500 foot level at the entrance of the Granite Creek canyon to intersect the ore body at a depth of 260 feet below the 300 foot level. Two divided raises were started at each end of the ore body and soon one was abandoned at the 400 foot level in order to concentrate on one raise through to the 300 foot level to provide ventilation for driving the second raise and for the stops that were being started from the 400 foot level to the 300 foot level.

In order to get some added ventilation to the first raise a diamond drill was set up on the 300 foot level and put down on the line of the raise. This hole was put down in the AX (2 1/4") size and then reamed to the BX (2 7/8" size) which provided good ventilation in the face of the raise.
An intermediate level was driven off the raise to connect the two raises along the strike of the ore 92 feet above the 500 foot level. The ore was fairly narrow at this point and open chutes were put in at 25 foot intervals and this intermediate level became a scram drift with the ore being slushed toward the center where a third raise had been driven up from the 500 foot level. Regular shrinkage stoping then was possible from this level to the 300 foot level.

In the Riley Extension mine the ore bodies on the 200 foot level were divided into the east vein which is the north extension of the U. S. Vanadium - Riley Mine ore deposit and the west vein which is an ore body occurring on the north side of the Getchell Mine property line. See Plate IV for plan map and Plate V for section of the Riley Extension mine. Both the east and west veins are mined by shrinkage stopes but the east vein requires a slusher - scrapper arrangement to aid the flow of broken ore to the chute as it dips less than 40 degrees and will not flow by gravity. A flat chute is built on the level at 25 to 30 foot intervals and the drift is driven for a double track. Each stope being worked has a 36" Pacific Slushmaster scraper with Pacific sheave anchors for placing the tail block in the face of the stope. A 10 hp electric hoist with two drums is mounted on an old car chassis and rolled from chute to chute where needed. Anchor bolts with turnbuckles are provided at each set-up to securely anchor the slusher as it tends to be top heavy and care must be taken while moving. Broken ore can then be slushed out of the stopes into the cars as required to provide a working face in the stopes for drilling.

In 1952 a two and three quarters compartment shaft was sunk a hundred feet north of the property line between the U. S. Vanadium property and on the Getchell side to a depth of 183 feet with a station at
PLATE IV
SECTION B - B
RILEY EXTENSION MINE
SCALE 1" - 100'
WM. NEWMAN

- gd. Granodiorite
- tc. Tactite
- hf. Hornfels
- Is. Limestone
152 feet below the collar. Cross-cuts were driven from the station to both the east and west veins and raises put up to the 200 foot level. Intermediate levels were driven off these raises and became scram drifts as open chutes were put in at 25 foot intervals along the scram drifts and stoping operations began. Broken ore was pulled from the scram drifts by 15 hp electric slushers mounted on turntables. This permitted slushing in either direction along the scram drifts to the original raises from the 400 foot level. The broken ore was dropped to the level where it was pulled from a regular chute with a radial gate and a sand gate and then trammed to the station and put in the pocket on the shaft station for hoisting to the surface.

The pocket on the station was a flat bottom bin cut out of solid rock which had a 10 hp slusher with a 42" Eimco folding scraper loading the 3 ton skip. The advantage of a pocket of this type was that the shaft didn’t have to be sunk to so great a depth as would have been required to provide a pocket of sufficient storage to feed by gravity to the skip. Also it permitted handling most any boulder that the Eimco Model 12B and Model 21 mucking machines could load. Further advantages were a definite saving in secondary breaking and no lost time in loading the skip because of stuck chutes.

The broken ore is hoisted to the surface and dumped directly from the skip, into another scraper arrangement for loading the diesel truck with ore for haulage to the mill. The underground ore at both the Granite Creek and Riley Extension mines is loaded by this method of slushing into the truck rather than dumping into a bin then pulling the broken ore from the bin into the truck. At the Granite Creek mine a 25 hp slusher pulls a 42" Pacific Slushmaster scraper and at the Riley Extension mine a 30 hp slusher also pulls a 42" Pacific Slushmaster scraper. The length of the Slusher bin
depends on the amount of cable the drums will hold but in both cases the bin is about 130 feet in length from the back to the slusher. The depth depends upon the desired storage and at Granite Creek where the broken ore is dumped from mine cars the track is about 16 feet above the bottom of the bin. At the Riley Extension mine the skip dumping point is also about 16 feet above the bin bottom. A ramp is built out of 8 x 8 timbers and 3 x 12 lagging to permit the diesel truck to back under the slusher which is mounted above the truck. The bottom of the bin has 8 x 8 ties set across the width of the bin and securely anchored. Old railroad rails of about 90 lb. weight are then securely set on the ties about two feet apart and running the full length of the bottom to the dumping chute into the truck. As the broken ore fills in between the rails and becomes solidly packed a smooth bottom is formed. If the flow of ore from the mine is particularly heavy then a slusher operator is put on to load the trucks, otherwise the truck drivers load their own truck. A truck can be loaded with an average of from 15 to 16 tons in less than five minutes with this type scraper arrangement.

One advantage of this type slusher for loading trucks over the bin is a greater storage of broken ore is possible for about the same construction cost. Also larger boulders can be handled and loaded into the truck for the large crusher to break which again is a definite saving in secondary breaking. Loading times are about equal but there is a safety feature in the slusher arrangement that cannot be overlooked when comparing its advantages to a bin with a chute for loading. Many men have been injured, some seriously, trying to bar down a plugged chute in spite of all safety precautions.
A two compartment inclined shaft is now being sunk from the 400 foot level of the Riley Extension mine to develop more ore at depth. This inclined shaft has advanced about 150 feet from the collar and is inclined about a minus 32 degrees to the east. From the information supplied from the diamond drilling in the immediate vicinity of the inclined shaft and from the indications found in sinking the shaft to its present depth it appears that the shaft will be in the tactite zone to the 600 foot level and possibly to the 800 foot level.

In order to speed up the mucking cycle of sinking the inclined shaft a 20 hp electric hoist is mounted above the level on the incline. This hoist pulls a 42" Joy scraper of medium welded construction with short harness at 200 fpm up the incline over a hopper and dumps directly into a mine car on the level. This arrangement permits removal of about 50 tons of broken ore from the shaft in about 2 1/2 hours with very little hand mucking. The blade of the scraper was cut on top about 3" from the edge and bent downward about 30 degrees then rewelded to permit a digging action while moving up the incline. This was the only alteration that was necessary and no extra weights were added to the scraper.

In the underground mine as the development work progressed to a point where the stoping could begin and more men and machines were employed it became apparent that the drilling costs were slightly higher than desired. Consequently a series of records were kept by the author that would show the pertinent information from the daily drilling. The data recorded was tabulated in monthly form and resulted in a satisfactory record which did not require an elaborate system and could be done very quickly at the end of each month. Examples of these results are given in table form and a discussion of each table follows below.
When the Getchell Mine converted from mining and milling gold to tungsten late in 1950 and early in 1951 one of the problems to be considered was whether to enlarge the drill steel repair shop or to have the work done by the piece by a commercial shop already in operation. The drill steel and rock drill bit problem in the gold mining was practically nonexistent as the ore and waste is very soft and for the most part could be excavated by a power shovel in the open pits without any drilling or blasting. The underground mine was in the process of doing some development work on the north end of the gold ore zone and the amount of drilling was small. Ingersoll-Rand steel rock drill bits with the type 1 thread were used in the gold mine for both the open pit and the underground. The steel repair shop was equipped with a small electric furnace for forging and tempering the steel bits and the drill steel threads and shanks. Also a grinder for sharpening the steel bits and a small tank for tempering the drill steel. One man did the steel repairs and sharpened the bits when it was necessary and that was only two or three times a month.

In order to get tungsten mining started as soon as possible, and because of the time required to equip a steel repair shop and the difficulty in securing experienced steel repairmen, it was decided to give a contract for steel repairs to a commercial firm on a trial basis. Several tool shops were contacted and the Fourth Street Tool Shop of Reno, Nevada offered the most attractive price by the piece and was most available from the standpoint of shipping facilities. The piece price was submitted by the thread and by the shank, either lug shank or stoper shank. This was satisfactory as used steel was sent in for repairs which sometimes only required a thread or a shank. By spending an average of three days a week the Fourth Street Tool Shop has been able to fill our drill steel requirements over the past four years.
Table I on Drill steel for the year 1954 is the result of a record kept on the monthly drilling done and resolved into a cost per foot of hole drilled. The costs are given for the new steel in terms of cost per foot of hole drilled as is the threading and shanking of all new and used steel. It will be noticed that a record on the drilling of the 7/8" hexagonal steel with the Timken type F thread was not begun until May when more of the drilling footage began to be accounted for by the smaller sizes of stopers such as the Ingersoll-Hand R38 and the JR38 jackleg. In May a separation was made on the steel being made up with the F series thread and the 1" quarter octagon and the 1 1/8" round steel being made up for the M series bits with the Timken type H thread as used on the DA30 drifters on the jumbo and the R48 and R58 stopers.

In studying Table I one will notice there is no definite trend from month to month because of the steel on inventory in the mine that is carried over from one month into the next. However the total average figures at the end of the year gives a figure that appears often during the year showing several average months. The months that are above the average can be traced to several causes but mostly are months of high labor turnover. Breaking in new men to run the drill machines, especially the jacklegs, often resulted in much steel breakage and in higher steel cost and threading and shanking costs. An example of this can be found when a miner using a jackleg where the steel change is not critical will start a hole with a second steel instead of the shorter starter steel. As the starter is a 30" overall 7/8" hexagonal steel and the second is a 54" overall 7/8" hexagonal steel this places a greater stress on the focal point of the steel which happens to be the collar of the shank at the machine and the result is much broken steel at the collar. This practice will also
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be reflected to a certain extent in the costs of the tungsten carbide bits coinciding to these months of high steel costs.

From Table I it will also be noted that the new steel cost for the smaller drilling machines are near the costs of the larger drilling machines in the eight month averages. However the cost of threading and shanking for the small machine is twice the cost of threading and shanking for the larger drilling machines. As there is nearly twice the footage of hole drilled for the small steel as compared to the larger steel it would still indicate a large amount of steel breakage in the small steel and the consequent re-threading and reshanking on the used steel. If a steel is broken in the longer changes it is sent into the tool shop to be cut down to the next smaller size change and threaded or shanked as the case may be or both if required.

When the tungsten mining program was begun early in 1951 the Ingersoll-Rand Company had just put on the market a tungsten carbide insert rock drill bit with a type 1 thread. This was the thread in use at that time on the Getchell property for the steel bits and was the first bit of this type used for rock drilling. The bit was satisfactory for perhaps 50 feet of drilling after which time the threads failed. Welding the bit to the drill rod was tried but that practice met with little success.

About the same time the Ingersoll-Rand Company also came out with a tungsten carbide insert rock drill bit with a type 118 thread especially suited for a 1 1/8" round steel and larger for wagon drill use. The fact that this same type rock bit is still in use today on both wagon drills is proof of the success of this particular type of rock drill bit for this particular drilling machine. The results have been phenomenally good compared to the results on any manufacturer's rock drill bits in use for underground drilling machines. A record was kept on these wagon drill bits
until it was determined that they were able to drill well over 2,000 feet per bit. A cost of about $0.010 per foot of hole drilled was thus established and was so satisfactory that no problem existed and no further records have been kept since that time.

The real problem to improve the bit cost of drilling was with the bits in use in the underground mine. A method of recording the drilling footages against the cost of the new bits issued to give a cost per foot of hole drilled was a record that could be made up quickly each month from data already on hand. An accurate account of the new bits issued from the warehouse to each underground mine was available. The daily drilling footages were available from the time cards and were entered daily in table form for easy totaling at the end of the month. The total cost of the new bits issued divided by the number of new bits issued resulted in an average cost per new bit issued. The total footage drilled during the month divided by the number of new bits issued determined the average footage drilled for each new bit issued. In Table II this is the result of both the F and the M series combined. Table III is an eight month record of the F series bit and Table IV is for the same eight month period for the M series bit. The average cost per new bit issued divided by the average footage drilled per new bit issued resulted in the average cost per foot of hole drilled.

A study of the results of Tables II, III, and IV have revealed some interesting facts to be considered. First and foremost Table III has shown that the F series bit can drill in this particular formation for three cents per foot less than the M series bit. This one point was well worth considering all the factors involved because a definite saving could be effected by a change over to the smaller types of rock drilling machines. The drill steel for the smaller type machines were nearly average as seen from Table I and some of the difficulties of steel breakage has been pointed
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Table III: Timken Carbide Bits
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<tr>
<th>Month</th>
<th>Total M Drilled</th>
<th>New M Series Bits Issued</th>
<th>Total M Bits Issued</th>
<th>Avg. Cost per new M bit</th>
<th>Avg. Feet per new M bit</th>
<th>Avg. Cost per foot M hole drilled</th>
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<td>128 114 4</td>
<td>246</td>
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<td>99 120 68</td>
<td>287</td>
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<td>17,582</td>
<td>47 48 36</td>
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<td>108</td>
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<td>158</td>
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<td>May</td>
<td>21,033</td>
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<td>126</td>
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out and may be practically eliminated by proper supervision. The drill steel therefore being nearly equal for the large and small drill machines then other factors in favor of the small machines were considered. The drilling capacity of each underground mine, where the compressor output was fixed, could be increased by the use of the 38A jackleg and the 38A stoper over the R28 and R5 stopers and the DA30 drifter as the smaller machines consume less compressed air. The resultant was that three small machines could be used in place of two of the larger machines. This definitely would contribute to an increase in production as more working faces could be worked at one time. Also the smaller machines seemed to have a little faster rate of penetration over the larger machines, 12 inches per minute as compared to 10 inches per minute. In one underground mine where the small machines were first put on trial the powder consumption was found to be 2.0 lbs. per ton of ore broken over an eight month period. In another underground mine where the larger machines were still in use the powder consumption was 3.0 lbs. per ton of ore broken over the same eight month period. This would seem to indicate a saving in powder may also be effected by a complete change over to the smaller drilling machines using smaller drill steel and bits. Still another factor to be considered in favor of the smaller rock drills was that repair parts were somewhat less expensive than parts for the larger machines. Last of the items in favor of the small drills was the ease of handling and the appeal and effect on the men using the equipment.

With all the factors involved and the evidence developed by the tungsten carbide bit records members of the staff discussed the possibility of gradually retiring the larger rock drill machines in favor of the smaller machines. The change over program was approved and put into operation early this year. The first of the large machines have been taken out
of service and overhauled to put in good working order and offered for sale. The spare parts in stock will be used for the overhaul and any left over will be exchanged for parts for the smaller machines or sold outright.

In conjunction with the records kept on the performance and costs of the Timken carbide insert bit and the subsequent change over to the smaller drill machines using the 7/8" hexagonal steel, a one month test was conducted on the use of the Joy Intraset drill steel. This is an alloy steel manufactured in Canada and has an integral chisel bit. The steel is made up at the factory in predetermined lengths and can be ordered in any desired lengths for the jackleg or the stoper. The drill steel tested here was also a 7/8" hexagonal steel with 4 1/4" shank and a pear shaped collar. Two JR38A jacklegs driving a drift in ore, in the lime hanging wall, and in the granodiorite footwall were given eight sets of Joy Intraset drill steel for the test. The sets included eight starters 2' 6" in length under the collar and eight seconds 5' 0" under the collar. The test was conducted under normal conditions with ordinary supervision. The chisel bits were sharpened by being held by hand against the grinding wheel and the gauge and bevel was maintained as close as possible with gauges supplied. A grinder to hold the steel in the proper position for sharpening has since been purchased for further test work. The eight sets drilled a total of 4,995 feet of hole and the test ended with 2 starters and 3 seconds that were available for extra footage. One second steel was stuck and the balance of the steel was lost due to breakage, mostly at the collar. None of the steel was lost due to premature insert failure. The test conducted over less than a month's time developed a cost of $0.052 per foot of hole drilled. This is a cost well worth further investigation because it includes both the drill steel and the carbide insert bit cost.
in one. This result is nearly half the best total cost of the 7/8" hexagonal drill steel from Table I plus the total cost of the F series Timken carbide bits from Table III. At least the figures already obtained by the one month's test warrant further test work with the possibility of a greater saving in the total drilling costs.

Before concluding this section on the use of the tungsten carbide insert bit a word about the method of sharpening the bits may be of some interest. For the first year the carbide bits were in service they were sharpened by holding the bit in the hand against the grinding wheel. This was a difficult operation and not very satisfactory according to the theory of correct sharpening. An Ingersoll-Rand JA-3 air driven grinder was purchased and adapted for sharpening the carbide bits. This included the proper collets for sharpening the Timken type F and M series bits, with the F thread and H thread respectively, and the Ingersoll-Rand type 118 thread for the wagon drill bits. Two grinding wheels are mounted in the JA-3 grinder, one for sharpening with a K face or Vee face and the other for gauging with an A face or a flat face. Both stones are of the GC 80 L11 VR class in the Carborundum Company classification and can be either 60 or 80 grit. The K face grinding wheel for sharpening is a 1 1/4" x 1 1/4" x 12" wheel and the A face wheel for gauging is a 1" x 1 1/4" x 12" wheel size. The grinding wheels are both rotated at about 3400 rpm and a stream of water is played on the grinding wheel to eliminate the silicon dust.

If the carbide bits are not allowed to become too dull it only takes about five minutes per bit to sharpen and gauge. Carrying racks holding eight bits are numbered and issued to the miners each shift. More bits are issued than required in order that all bits may be used and not dull any one set. Much less of the carbide insert and the bit body is lost to
grinding if the bit is not drilled too dull. An eighth of an inch wide 
flat edge of the cutting edge of the insert is considered excessive drilling. 
From test work conducted in various working faces in the mine it was determined that ten to twelve holes were sufficient drilling for one set of bits and sharp bits should be put on the drill steel to hold excessive bit wear to a minimum.

The conclusions drawn from the results of the records of the drill steel of Table I and of the Timken carbide bits of Tables II, III, and IV have shown certain advantages in favor of the smaller types of drilling machines. These advantages can be summed up briefly as:

1. A direct saving $3 per foot of hole drilled.
2. Less compressed air consumed.
3. Increased rate of penetration.
4. Less powder consumption per ton ore broken.
5. Less expensive parts for repairs.
6. Ease of handling.

V Diamond Drilling.

The results of a three year diamond drilling program with an accompanying table of costs may be of some interest before this paper is concluded. The use of the surface and underground drilling machines have been mentioned briefly for the usual prospecting and development work and for determining geologic structure.

The diamond drill used for surface drilling is a Chicago Pneumatic No. 8 gasoline driven drill rated at 1000 feet of EX fittings for 7/8" core. The drills in use underground include a Joy HS-15 air drill with a vane type motor and rated at 500 feet capacity with EX fittings for 7/8" core. Also a Joy No. 12 air drill with a piston type motor and is rated
<table>
<thead>
<tr>
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<th>1952</th>
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<td>340</td>
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<td>Average per drill</td>
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<tr>
<td>drill shift</td>
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<tr>
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<tr>
<td>Setting &amp; blank bit</td>
<td>0.39</td>
<td>0.36</td>
<td>0.36</td>
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<td>0.36</td>
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<tr>
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at 750 feet capacity with EX fittings for 7/8" core.

One novel use of the underground drill here was to improve the ventilation in a long raise being run to connect one level to another. The diamond drill was carefully set up on the 300 foot level of the Upper Granite Creek workings and an AX hole of 2 1/4" diameter was put down a distance of 260 feet on the dip of the ore to the 500 foot level of the Lower Granite Creek workings. The hole was then reamed to an EX hole of 2 7/8" diameter. The two compartment raise was then begun from the 500 foot level using this drill hole as a pilot and providing some ventilation.

One of the most troublesome and most expensive problems encountered while drilling surface holes in the vicinity of the Riley Extension mine was about 200 feet of loose sand as an overburden. Drilling and cementing had little effect as the cement would cave with the walls of loose sand. While the problem still exists, holes were put down by first putting a 6" churn drill hole down as far as possible up to 100 feet and casing it with 5" casing. Then the diamond drill rig is set up over this vertical hole and a 3 1/2" fishtail bit is used inside 4" pipe to put the 4" pipe down another 20 feet if possible. Next a 2 1/2" fishtail bit was used inside the EX casing to get it down as far as possible and so on until the AX and EX casing is in to solid formation which could be expected from 200 to 210 feet.

Table V shows the costs per foot of hole drilled for the machine rental, materials, labor, diamonds, and setting and blank bit charges from a three year diamond drilling program. The drill machines were on a rental basis from the Mitchell Diamond Drill Company of San Francisco. Mr. Mitchell supplied experienced drillers and attended to the details of keeping the drills supplied with materials such as casing, diamond bits, etc. The Getchell Mine furnished helpers for these drills and two of these men have
stayed on and are now running company-owned diamond drills.

VI. Conclusion.

The tactite deposits of the Osgood Range were formed by the granodiorite invading the sedimentary rocks of the Triassic age. The deposits have been known for many years and were mined during World War II. The various deposits carrying the heavier copper stain were prospected in the seventies, probably for copper.

The government program for stockpiling strategic minerals in 1950 made it economically feasible to mine the Potosi District tungsten ores and treat them in the Getchell Mill, with minor changes in the gold circuits. The attractive unit price offered for the tungsten made it possible for the Getchell Mine Inc. to investigate and develop all known deposits of tactite in the district occurring around the periphery of the granodiorite stock that forms the northern end of the Osgood Range. The result of this development program has exposed many times the tonnage of economic ore than was ever reported in any of the publications that have been written on the possible ore reserves of the district. The tonnage of ore mined from the various tactite deposits belonging to the Getchell Mine Inc. since the beginning of this tungsten program late in 1950 totals 679,190 estimated wet tons. Of this 361,930 estimated wet tons has been produced by open pit mining and the balance of 317,260 wet tons was produced by underground mining.

When developing a tactite ore body occurring around the periphery
of the granodiorite stock the best indication of a deposit, of course, is a surface outcrop. In the absence of a surface outcrop an indication of a possible ore deposit may be found in the trace of the granodiorite – lime contact. An abrupt change of strike that may project the lime beds into the granodiorite is considered favorable enough to warrant further investigation by diamond drilling or underground drifts or cross-cuts if not too extensive. Also favorable lime beds seem to be a contributing factor in the formation of a tactite deposit.

Open pit mining is done by benching the ore deposit on the steep hillsides. A ratio of 1 ton ore to 10 tons waste is the limit of the stripping that can be done on an ore body before it is mined by underground methods. The wagon drills do all the rock drilling in the open pits. The wagon drill holes are sprung and loaded with bag powder and produces very good fragmentation of the tactite rock. Haulage costs of 24 $ per ton-mile includes a shoveling or loading cost of 6 $ per ton-mile, the actual haulage cost of 16 $ per ton-mile, and the road maintenance accounts for the remaining 2 $ per ton-mile for the average haul of 8 to 10 miles from the open pits to the mill.

Underground mining is done by shrinkage stoping and where the deposit dips less than 40 degrees a slusher is used in the stops to move the ore to the chutes. Slushers are used in scram drifts on sub-levels and in the pocket on the shaft station to load the skip. A slusher is also used to load the mine ore into a truck for haulage to the mill.

A study of the underground rock drilling equipment produced results that were favorable enough to warrant a complete change over of drilling equipment. Large rock drilling machines are being replaced by smaller drilling machines for a direct saving in rock drill bit costs of 3 $ per foot of hole drilled. Other advantages of the change over to smaller machines
are: 1) less compressed air consumed therefore allowing more working faces and hence added production, 2) an increased rate of penetration with the smaller bits and also results in an increased production, 3) less powder consumption per ton ore broken, 4) less expensive drill machine repair parts, and 5) easier handling of the smaller machines. These advantages were sufficient for the Getchell Mine to offer for sale all their large stopers and replace them with small stopers of the Ingersoll - Rand 38A model.

The diamond drill has proven itself a useful tool in the prospecting and development of the tactite ore bodies. A four year program has resulted in an overall cost of $8.12 per foot of hole drilled. Of this cost $2.19 per foot was for machine rental, $0.83 per foot for materials, $3.68 per foot for labor, $0.76 per foot for diamond cost, and $0.37 per foot for setting and blank bit charges. The balance of $0.29 per foot was for machine purchase, and truck rental and purchase.
VII Acknowledgment.

The writer wishes to express his appreciation to Mr. Roy A. Hardy, Consulting Engineer, and to Mr. Royce A. Hardy, General Superintendent, of the Getchell Mine Inc., for their help and permission to use the items of information included in this paper.