1200 0150

Gaylord

Ralph

309

ITEM 213

HOLT, INC.

Mining & Mineral Exploration Consultants

199 N. Stone - Suite 200

Tucson, Arizona

PROGRESS REPORT

EXPLORATION PROGRAM

CENTRAL COMSTOCK MINES

VIRGINIA CITY MINING DISTRICT

STOREY COUNTY, NEVADA

DATA BY: Dana D. Norman
Dana D. Norman, Sr. Geologist

DATE: August 21, 1969

FOR: Siskon Corporation

APPROVED BY:

Robert E. Holt
President

or

REGISTERED GEOLOGIST
CERTIFICATE NO.
7231
DONLON
LOBIONDO
Date Signed 8-22-69
Arizona U. S. A.

Donlon LoBiondo
Chief Geologist

PROGRESS REPORT

EXPLORATION PROGRAM

CENTRAL COMSTOCK MINES

VIRGINIA CITY MINING DISTRICT

STOREY COUNTY, NEVADA

INTRODUCTION

This report has been prepared to update the information presented in my progress report of June 5, 1969, and to re-evaluate the possible ore reserve picture as originally described in my report dated April 7, 1969.

The initial drilling program has been completed and, in general, has tended to verify what was previously known or surmised concerning the mineral deposition in the Comstock properties. A considerable amount of additional drilling will be necessary to verify the existence of an ore-body and make plans for mining, as well as to get a better idea of exactly what the dollar value of any ore might be. The text of this report describes the drill results to date, how such results may affect mining or ore reserves, and future exploration.

A total of 1866 feet of drilling was completed from the time the program commenced on May 6, 1969, until it was terminated on July 18, 1969. This drilling consisted of 1089 feet of rotary or non-core drilling and 777 feet of core drilling. Seven holes were drilled, varying in depth from 187 feet to 380 feet. Most of them did not reach the proposed depth because of extremely bad drilling conditions and/or old underground mine workings. Core recovery varied from 41 percent to 68 percent in the cored portions of individual holes. Two of the holes were inclined at -45°.

All of the drilling within the mineralized portion of the lode substantiates the belief that ore-grade material is erratically distributed within quartzose zones, which are apparently lenticular in shape and, in all likelihood, pinch and swell along the strike of the vein system as well as along the dip.

Assay values were about what was indicated from underground sample data, although ranging both higher and lower than expected, and showing much more non-ore grade material than was hoped existed.

DRILLING

Following is a brief description of each hole drilled, and included are the holes that were described in the progress report dated June 5, 1969. Dollar values of mineralization were calculated using prices of $42/oz. gold and $1.75/oz. silver (latest silver price $1.56).

DRILL HOLE CCM-1:

Coordinates: 12820 N., 6670 E; collar elevation: 6260 feet; total depth: 187 feet; inclination: vertical, drilled rotary to 135 feet; deepened by coring to 153 feet with difficulty and poor recovery. The drill rods were extended to a depth of 187 feet without encountering anything solid and without recovery. The hole was presumed to be in an old working and was stopped.

All drilling was in highly fractured ground, much of which could have penetrated old caved or filled mine workings. Core recovery was 55 percent and consisted of fragments of quartz, altered, mineralized andesite, and clay. Much limonite stain was present. In zones of higher grade values, very fine-grained, disseminated silver sulfide mineralization was noted.

Traces of finely crystalline pyrite were noted in the core, and a speck of free gold was noted in panning.

Assays:

0-60 feet - No ore grade values. Assays varied from $0.80 to $4.11 per ton.

60-80 feet = 20 feet @ $21.45

80-90 feet = 10 feet @ $2.89

90-105 feet = 15 feet @ $5.89

105-120 feet = 15 feet @ $15.86

120-131 feet = No recovery

131-145 feet = 14 feet @ $9.13

DRILL HOLE CCM-2:

Coordinates: 21135 N., 6320 E.; collar elevation: 6310 feet; total depth: 288 feet; inclination: vertical; drilled rotary to 110 feet; deepened by coring to 288 feet, a portion of which apparently passed through old mine workings where drilling was difficult and recovery poor. The hole was stopped after reaching the 365 foot underground level. Core recovery averaged 61 percent.

This hole was drilled to verify the existence of indicated ore-grade mineralization on L-237 and to test for possible extension of such material to L-155 and above, and below to L-365.

Drilling was in overburden to a depth of at least 20 feet. Drilling from 20 feet to 192 feet consisted of mineralized, altered andesite and quartz, with some finely disseminated pyrite. Much of this drilling apparently passed through old mine workings. Drilling from 192 feet to 288 feet was in pyritized, silicified andesite with moderate quartz veining, some of which was ore-grade. The latter zone was moderately to intensely fractured, but no mine workings were encountered. Trace amounts of silver sulfide dissemination were noted in the zones of highest values.

Assays:

0-180 feet = 180 feet - No ore-grade material. Maximum value $5.30.

180-185 feet = 5 feet @ $11.53

185-215 feet = 30 feet - No ore-grade material. Maximum value $1.75.

215-235 feet = 20 feet @ $12.48

235-240 feet = 5 feet @ $1.86

240-288 feet = No assays available.

This hole apparently passed through the zone of strongest quartz mineralization in the east vein and thence into the footwall in pyritized, silicified andesite, with one interesting zone from 215 feet to 235 feet. The fact that much of the drilling to 192 feet passed through old workings with poor assay results may indicate that most of the ore-grade material was removed and the workings filled with waste, or caved.

DRILL HOLE CCM-3:

Coordinates: 12090 N., 6075 E.; collar elevation: 6320 feet; total depth: 206 feet; inclination: vertical; drilled rotary to 80 feet; deepened by coring to 206 feet. The hole was stopped after encountering extremely bad ground with coincident high costs and poor recovery. Average core recovery was 41 percent.

The hole was drilled to verify the existence of indicated ore-grade mineralization between L-155 and L-237 and to test the zone from L-155 to the surface. Drilling was stopped just below L-237 so that the zone to L-365 could not be tested.

All drilling was in strong quartz mineralization with a few short sections of altered andesite. Much of the quartz may represent highly silicified andesite. Limonite stain was prominent. Finely disseminated silver minerals were noted in zones of highest assay values. Wood fragments and poor recovery indicate that the hole passed, in part, through old workings. Recovery during last 10 feet consisted of pebble-sized quartz fragments.

Assays:

0-30 feet = 30 feet @ $2.42

30-50 feet = 20 feet @ $5.26

50-80 feet = 30 feet @ $9.92

80-150 feet = 70 feet - no ore-grade material

150-165 feet = 15 feet @ $15.36

165-206 feet = 41 feet - no ore-grade material

DRILL HOLE CCM-4:

Coordinates: 12375 N., 6220 E.; collar elevation: 6230 feet; total depth: 210 feet; inclination: vertical; drilled 47 feet rotary and deepened to bottom with coring rig. Core recovery averaged 68 percent.

This hole was drilled to verify the existence of ore-grade mineralization from the bottom of the present pit to L-365.

Drilling to 30 feet was in overburden of pit slough. Drilling from 30 feet to bottom was in massive, limonite-stained quartz, except for a zone from 116-152 feet drilled in argillized andesite. Trace amounts of finely disseminated silver minerals were noted in the better grade zones. The hole was stopped 10 feet short of the proposed depth because of poor recovery and blocking of the bit. It may have been in an old working.

Assays:

0-30 feet = 30 feet overburden - no ore-grade material
30-45 feet = 15 feet @ $7.29
45-110 feet = 65 feet @ $15.61
110-140 feet = 30 feet - no ore-grade material
140-170 feet = 30 feet @ $22.51
170-190 feet = 20 feet - no ore-grade material
190-210 feet = 20 feet @ $6.67

DRILL HOLE CCM-5:

Coordinates: 11700 N., 6360 E.; collar elevation: 6310 feet; total depth: 220 feet; inclination: vertical; drilled entire length rotary and stopped after reaching water and failure to encounter any significant mineralization.

This hole was drilled to test for the possible southerly extension of the east vein.

Recovery to 105 feet consisted of mixed fragments of altered andesite and quartz with much clay, soil, and limonite stain. A few fragments of diorite were present in the upper portion. This entire zone could be in overburden. From 105-220 feet, drilling was in moderately to strongly silicified, moderately argillized, highly pyritized andesite. Pyrite content averaged about 15 percent.

Assays:

Gold values did not exceed a trace and the highest silver assay for a 5-foot sample was 0.53 oz.

From the results of this hole, it might be concluded that the east vein does not extend this far south or that it may lie east of the zone penetrated by the hole.

DRILL HOLE CCM-6:

Coordinates: 11780 N., 6110 E.; collar elevation: 6350 feet; total depth: 375 feet; inclination: -45°; bearing: N.73° W.; drilled rotary to 60 feet; deepened to bottom with coring rig. Core recovery averaged 68 percent.

Drilling to 110 feet appeared to be in overburden. From 110-195 feet, drilling encountered weathered, oxidized andesite. Drilling from 195-354 feet was in massive quartz and and silicified, weakly pyritized andesite. From 354-375 feet, rock type consisted of silicified, argillized andesite with finely disseminated pyrite, about 15 percent by volume. The hole was stopped after extending 19 feet beyond massive quartz mineralization and beyond the projection of the footwall of the lode. However, footwall diorite was not encountered.

This hole was drilled to test the relatively unexplored area between indicated ore-grade mineralization 200 feet to the north, on L-155 and L-237, and the south end of the property. Although a significant width of massive quartz was encountered, the drill results from this hole suggest ore-grade material does not extend south of that indicated by underground sampling.

Assays:

Assay values were relatively uniform throughout most of the lower, quartzose portion of the hole. These values averaged about $1.25/ton; and although there were several higher assay values, there was no ore-grade material encountered.

DRILL HOLE CCM-7

Coordinates: 11700 N., 6360 E.; collar elevation: 6310 feet; total depth: 380 feet; inclination: -45°; bearing: N. 73° W.; drilled with non-coring bit entire length.

This hole was drilled to test for mineralization in an unexplored zone, believed to be barren, but in which underground geologic mapping suggested the possibility of quartz mineralization.

Drilling to 60 feet was in overburden. The remainder of drilling was in silicified, pyritized andesite which was heavily oxidized and limonite-stained near the surface. The rock became progressively fresher with depth, and pyrite content was estimated at 3-4 percent at the bottom of the hole.

Assays:

No ore-grade assays were obtained from the samples of this hole; in fact the highest value obtained was $0.89/ton.

RESUME OF DRILL RESULTS

If the drill hole assay data is studied from a purely statistical basis, a depressing picture is presented. Four drill holes (CCM-1, 2, 3, and 4) penetrated into the presumed ore zone. These holes had a combined footage of 891 feet of which 264 feet, or 29.6 percent, was in ore-grade material averaging $13.83 per ton, as per the following table.

Hole No.	Total Depth	Ore Thickness	Ore Grade
CCM-1	187'	64'	$13.80
CCM-2	288'	25'	12.30
CCM-3	206'	45'	11.73
CCM-4	210'	130'	14.87
Totals	891'	264'	Average $13.83

If these drill results are believed to be indicative of the entire mineralized zone, then it would have to be assumed that profitable extraction of the mineralization could not be realized. Several factors, however, strongly suggest that these results do not indicate the true ratio of ore to waste in the potential ore zone. One factor is the poor core recovery; considerable portions of unrecovered material could have contained ore values. Of more significance, is the fact that the holes were drilled vertically into a zone of mineralization where it is believed that the ore is localized in nearly vertical, lenticular shaped veins or bodies. Vertical drilling could easily miss ore zones and also tells nothing about the width of mineralization. Furthermore, underground assay maps, if considered reliable, indicate much more ore-grade mineralization than was revealed in the drilling;

and since such mineralization is erratically distributed, it could easily have been missed in the drilling to date.

PROPOSED DRILLING:

A program of inclined drill holes is believed to be the only method of properly evaluating the ore potential of the deposit without actually rehabilitating the old mine workings and sampling or drilling underground, which would be very costly. Such drilling should be done at angles of -45° from the east wall of the vein and would have to extend along the entire length of the indicated ore zone at predetermined intervals.

To test the extent of mineralization and reasonably verify the existence of an orebody, it is proposed that a minimum of 18 inclined holes be drilled from the east wall of the lode. Although only 14 proposed holes are shown on the accompanying drill hole map and cross sections, 4 additional holes are included in the proposal in case of lost holes or the necessity of a few additional holes where they might be indicated. These holes would be located along east-west lines, spaced 200 feet apart along a length of 1200 feet. Two to three holes would be necessary on each line depending on the indicated vein width and the depth to the limit of mineralization. They would have depths ranging from about 200 feet to 400 feet and probably average 275 feet. In several instances, vertical holes might have to be substituted for inclined holes at the north end of the property, where roads and buildings would prevent construction of drill sites.

This proposed drilling would have a combined total footage of about 5000 feet, of which approximately half could be done with a non-coring bit through the hanging wall andesite. A cost estimate on such work is as follows:

Core Drilling - 2500 feet @ $25/ft.	$62,500
Non-core drilling - 2500 feet @ $8/ft.	20,000
Assaying	2,500
Supervision and Miscellaneous	15,000
Total	$100,000

If two drill rigs running on two shifts per day were available, the time required for such work could vary from four to six months.

An alternate method of drilling could be used: that of a large rotary or oil well type rig employing air. This type drilling might prove more successful in certain aspects, but it might be more costly since it would be limited to the drilling of vertical holes, which in turn would require a greater number of holes to secure data that can be gained from inclined drilling.

In studying the accompanying map and cross sections, it is readily obvious that the westernmost, shallower holes should be drilled first. In view of the fact that ore-grade material must be encountered immediately or at very shallow depth to make the deposit economic, the lack of ore-grade material in these holes would preclude the possibility of an orebody, and the remainder of the holes would not have to be drilled. This, of course, might reduce the estimated cost of drilling by about one-half.

<u>ORE RESERVES:</u>

In my report of April 7, 1969, I estimated that potential reserves could amount to approximately 1,774,000 tons, if such tonnage were to include the northern part of the assumed ore zone, which would necessitate removal of the old Fourth Ward Schoolhouse. I also estimated that approximately 1,366,000 tons would be available if the pit limits were kept away from the school. These original tonnage estimates were based on the assumption that all of the ore-grade mineralization indicated on the underground assay maps was in place and minable. The tonnage figures were arrived at by calculating by planimeter the areas of indicated ore-grade mineralization on each level and applying a proper thickness above and below the level to arrive at a volumetric calculation. This figure was divided by a factor of 13 cu. ft. per ton to arrive at a tonnage estimate to L-365.

In a more recent calculation, intended to check the former, potential reserves were estimated by using a different method. The areas of potential ore indicated on cross sections 1A through 7A (accompanying this report) were measured by planimeter. The potential ore areas of each two adjacent cross sections were averaged and were multiplied by the distance between them along the strike of the mineralized zone. This volumetric calculation was then divided by 13 cu. ft. per ton, producing a resultant estimated tonnage potential of approximately 1,600,000 tons. This checks reasonably well with the original calculation of 1,774,000 tons and is considered to be more accurate.

Drilling results now indicate that, within the potential ore zone, there is likely to be a considerable amount of non-ore grade material, as well as mined-out zones that were not completely back filled with material that was presumed to offer reserves of sufficient value to support open pit mining. Therefore, it is readily apparent that the maximum potential ore reserve estimate must be reduced by some factor. There is, however, no way of determining at this time precisely what that factor should be.

In view of the fact that insufficient drilling has been done, I am assuming that a practical factor might be 30 percent. Further assuming that there could be 1,600,000 tons of material within the zone of potential reserves and applying the reduction factor, we arrive at an estimated potential tonnage of 1,120,000 tons.

The recoverable gold-silver value has been placed at about $9 per ton. This value is based upon calculations made by W. D. Thompson and Irving Gray during their evaluation of the property in 1968, in which they estimated the ore value at $10.08, and on calculations by H. B. Chessher, Jr., based on Hazen's figures for 1950, which indicate ore value at present market prices of $10.29 per ton. A mill recovery of 90 percent was assumed.

The results of additional drilling may indicate that the ore reserves actually present, if any, will be lower than these estimates; but they may be of higher grade.

ECONOMIC CONSIDERATIONS:

In view of the limited amount of new information gained from the drilling program and the impossibility of making an accurate ore reserve estimate, there is little that can be added to my previous comments in reports or to the economic analysis submitted by H. B. Chessher, Jr., on February 2, 1969. However, several other considerations should certainly be recognized now.

First of all, it appears that potential ore tonnages may be substantially lower than the 2 million tons originally estimated. Such tonnage might be of a higher grade than thought, but to maintain such a grade will necessitate close ore control in any pit operation.

A calculation made of the assumed waste to ore ratios from the cross sections indicates that the ratio will be about 7/1 rather than the 4/1 ratio assumed in Chessher's analysis.

When considering the reduced tonnage that presently appears available, the capital investment and extraction costs, plus the cost of proving the existence of an orebody, the advisability of a joint venture seems even more obvious.

A joint venture might also prove advantageous in another respect. The presumed orebody that Union Pacific has reportedly developed is approximately 3000 feet south of the mineralization on the Siskon ground. It is also at an elevation that might allow the driving of a haulage adit beneath the Siskon ground and extracting the presumed ore by bulk underground mining. There are many obvious problems to be solved if underground mining is contemplated, but if at all economical, one that should not be overlooked in view of the many problems also present when considering an open pit operation.

DRILL HOLE CCM-3

Rotary Cuttings

Depth	Description
0-5' 5-10' 10-15' 15-20'	Quartz fragments containing blebs of limonite (after pyrite?) and limonite coatings on fracture surfaces. Small amount of magnetite. 95 percent quartz. Quartz is amorphous to crystalline. Small amount ferromags associated with quartz.
20-25' 25-30' 30-35' 35-40' 40-45'	Same as above, but finer particles with more magnetite ± 5 percent. Wood fragments at 40'. No sulfides. No Ag minerals observed.
45-50' 50-55' 55-60' 60-65' 65-70' 70-75' 75-80'	Same as from 0-20'. Trace amounts of very fine-grained metallic mineral, possibly Ag sulfide, disseminated in some fragments of crystalline quartz.

DRILL HOLE CCM-3

Core Depth	Recovery	Description
80.0'-90.0'	6.0'	Quartz. Crystalline, tan coloration. Limonite (goethite) along fractures. Badly broken. Limonite inclusions after pyrite. Manganese stain on fracture surfaces.
90.0'-91.9'	1.0'	
91.9'-102.0'	----	
102.0'-108.4'	6.0'	Andesite. Highly altered. Oxidized. Feldspars argillized. Mafics to limonite. Thin veinlets of introduced quartz.
108.4'-112.0'	1.0'	
112.0'-120.5'	6.5'	Quartz. Same as from 80.0'-102.0'. Some later quartz veining. Oxidized. No sulfides. Highly broken, recovery fragments only.
120.5'-127.5'	3.5'	
127.5'-135.1'	3.8'	Andesite as from 102.0'-112.0'.
135.1'-142.0'	4.0'	Quartz-andesite fragments. Maximum size about 1". Oxidized. No sulfides.
142.0'-147.0'	----	No recovery.
147.0'-150.0'	0.1'	Crystalline quartz fragments. Largest 1 1/2". Silver minerals seen under scope. Oxidized. May be old working.
150.0'-152.0'	----	
152.0'-157.0'	0.1'	
157.0'-161.5'	0.4'	
161.5'-166.0'	0.4'	
166.0'-167.5'	----	
167.5'-170.6'	0.3'	

DRILL HOLE CCM-3

Core Depth	Recovery	Description
170.6'-174.0'	2.6'	Andesite. Argillized, oxidized. Mafics to limonite. Quartz veining.
174.0'-181.1'	6.0'	Quartz, massive crystalline. White mottled with tan. Limonite-manganese on fractures. No sulfides. Highly broken.
181.0'-194.8'	9.0'	
194.8'-200.5'	1.5'	Quartz. Same as above, but intensely broken. Fragments 1 1/2". Could be old working. Hole stopped because of broken ground, coincident high costs to proceed, and poor recovery.
200.5'-206.0'	----	

DRILL HOLE CCM-4

Rotary Cuttings

Depth	Description
0-5'	Quartz fragments with fragments of altered
5-10'	andesite and fresh, fine-grained diorite.
10-15'	Fine-grained disseminated magnetite: Much
15-20'	limonite stain. Believed to be overburden
20-25'	or slough. 15 percent quartz. No sulfides.
25-30'	No recovery.
30-35'	Coarse quartz fragments. Limonite stain on
35-40'	fracture surfaces. Some quartz fragments
40-45'	show finely disseminated grey metallic
45-47'	mineral. Ag sulfide? 90 percent quartz.
	No pyrite observed.

DRILL HOLE CCM-4

Core Depth	Recovery	Description
47.0'-54.5'	7.0'	Quartz, heavy limonite. Oxidized. No sulfides. Badly broken. Quartz a brownish color. Trace amounts dark grey, soft metallic mineral, presumed to be Ag sulfide, disseminated in thin veinlets of later quartz. Limonite-stained clay (faults?) erratically distributed. Occasional streaks and blebs of black quartz.
54.5'-61.5'	6.5'	
61.5'-70.0'	7.0'	
70.0'-80.5'	7.3'	Same as above, but with manganese-limonite coating on fractures and within quartz. Small amount of unoxidized pyrite present. Some finely crystalline magnetite. No Ag sulfides observed. Much black coloration apparently due both to manganese and magnetite content, erratically distributed. Broken.
80.5'-86.7'	6.0'	
86.7'-95.5'	6.8'	
95.5'-104.0'	7.0'	
104.0'-105.5'	1.5'	Quartz, white to buff colored. Massive, crystalline. Much limonite on fractures. Intensely broken. No Ag minerals observed.
105.5'-115.0'	7.0'	

DRILL HOLE CCM-4

Core Depth	Recovery	Description
116.0'-129.0'	8.5'	Andesite, intensely argillized. Much limonite after pyrite dissemination and hairline veinlets. Badly broken. Intense crushing from 129.0'-152.0'.
129.0'-141.0'	9.0'	
141.0'-152.0'	7.0'	
152.0'-163.0'	9.5'	Quartz, buff colored, massive crystalline. Intensely broken. Much limonite. Inclusions of altered, limonite stained andesite. Trace amount of Ag sulfides associated with veinlets of later stage quartz.
163.0'-168.0'	3.0'	
168.0'-174.0'	5.0'	Quartz, massive crystalline, white to slight buff colored. Badly broken, but not as much as 152.0'-168.0'. Less limonite stain on fractures. Some manganese stain. No Ag minerals observed.
174.0'-188.0'	3.0'	Same as 168.0'-174.0'. Intensely broken with very poor recovery.
188.0'-194.0'	5.0'	Quartz, same as 168.0'-174.0'. Badly broken. Inclusions of silicified andesite, oxidized. Poor recovery. Hole stopped because of poor recovery, blocking of core, and nearness to projected depth of 220 feet. No Ag minerals observed.
194.0'-210.0' T.D.	5.0'	

DRILL HOLE CCM-5

Rotary Cuttings

Depth	Description
0-5' 10-15' 20-25' 30-35' 40-45' 50-55' 60-65' 70-75' 80-100'	Overburden. Mixed fragments of altered andesite, fine-grained diorite, and quartz. Much limonite stain. Clay and soil. Disseminated magnetite in quartz and in rock fragments. Lower portion of this zone could be altered, silicified andesite, but impossible to be sure. 75 percent quartz. Small amount very fine-grained pyrite seen in panning lower part of this zone.
100-105'	Same as above with pyritized andesite fragments.
105-215'	Moderately to strongly silicified, highly pyritized andesite. Medium gray color. Pyrite content 10 - 20 percent by volume. Some argillization of feldspars. No oxidation. No Ag minerals observed. Pyrite content locally to 25 percent. No magnetite, no mafics.

DRILL HOLE CCM-6

Core Depth	Recovery	Description
0'-60'	Rock Bit	Overburden. Fragments of altered, oxidized andesite, clay, soil.
60.0'-81.0'	9.0'	Overburden. Unconsolidated fragments of altered, oxidized andesite, clay, sand. Occasional fragment of diorite.
81.0'-94.0'	8.5'	
94.0'-110.0'	8.0'	
110.0'-125.0'	9.0'	Unconsolidated fragments of oxidized, weathered andesite. Limonite stained. Crumbly.
125.0'-140.0'	5.0'	
140.0'-142.0'	1.2'	Oxidized, weathered, fine-grained andesite. Much limonite, badly broken. Thin quartz veinlets, randomly oriented. Considerable limonite pseudomorphs after finely disseminated pyrite.
142.0'-163.0'	6.5'	
163.0'-171.0'	3.7'	
171.0'-180.0'	3.0'	Fault gouge.
180.0'-190.0'	3.6'	Andesite fragments same as 140.0'-171.0'.
190.0'-195.0'	3.5'	Andesite, highly weathered, oxidized. Fragments of massive vein quartz. No sulfides or Ag minerals observed.

DRILL HOLE CCM-6

Core Depth	Recovery	Description
195.0'-205.0'	8.5'	Quartz. Massive crystalline. Light buff colored. Badly broken. Veinlets of later stage, clear crystalline quartz. Much limonite stain, mostly on fracture surfaces. Inclusions of pyritized, altered andesite. Small amount of finely disseminated pyrite in some quartz mineralization. No Ag minerals observed. Portion of material is highly silicified andesite.
205.0'-218.0'	9.0'	
218.0'-227.0'	8.5'	
227.0'-234.0'	5.0'	
234.0'-238.0'	3.0'	Silicified andesite. Finely crystalline pyrite dissemination ±1 percent by volume. Rock brecciated and recemented with quartz and intensely silicified. No Ag minerals observed. Limonite on fracture surfaces. Practically all mafics, feldspars destroyed. Some chlorite remaining.
238.0'-247.3'	8.5'	
247.3'-249.0'	1.7'	
249.0'-256.0'	6.5'	Quartz, massive. Light grey to tan; grey where unoxidized. Moderately to badly broken. Limonite on fracture surfaces and in oxidized zones of quartz. Disseminated, fine-grained pyrite ±2 percent. No Ag minerals observed. Oxidized zones probably represent incompletely silicified andesite. Very short runs.
256.0'-265.0'	6.8'	
265.0'-274.0'	8.0'	
274.0'-288.0'	9.0'	
288.0'-298.0'	8.5'	

DRILL HOLE CCM-6

Core Depth	Recovery	Description
298.0'-304.0'	5.0'	Crushed zone, (fault?). Fragments of quartz and silicified andesite. Much limonite.
304.0'-306.0'	2.0'	Quartz, white to tan. Oxidized with considerable limonite on fractures and within quartz. Finely disseminated pyrite in unoxidized zones, ±3 percent by volume. Moderate to strong fracturing. No Ag minerals observed. Some portions intensely fractured and crushed. Small amount of included andesite.
306.0'-316.0'	10.0'	
316.0'-324.5'	8.4'	
324.5'-330.5'	6.0'	
330.5'-342.0'	8.8'	
342.0'-352.0'	9.2'	
352.0'-354.0'	2.0'	
354.0'-360.0'	6.0'	Andesite (?). Medium grey, fine-grained, highly silicified, and argillized with finely disseminated pyrite. 10-20 percent by volume. Rock soft and, in part, intensely crushed. Unoxidized. No Ag minerals. Some chlorite.
360.0'-370.0'	9.0'	
370.0'-375.0'	4.0'	
T.D.		

Hole stopped after extending 19 feet beyond massive quartz mineralization and beyond projection of the footwall of the lode. However, footwall diorite was not encountered.

DRILL HOLE CCM-7

Rotary Depth	Description
0-10'	Overburden. Limonite stained soil, clay,
10-20'	and sand. Small fragments of quartz and
20-30'	weathered andesite. Some fresh pyrite in
30-40'	fragments of unoxidized quartz or silicified
40-50'	andesite.
50-60'	
60-70'	Andesite. Partially weathered, limonite
70-80'	stained, silicified. Some fragments of
80-90'	fresh, silicified, pyritized material. Dis-
90-100'	seminated magnetite present. Some chlorite
100-110'	and also argillized fragments. Difficult
110-120'	to ascertain exact position of overburden -
120-130'	andesite contact. Fragments 70 percent
130-140'	quartz. No Ag minerals observed. Rock progres-
140-150'	sively less oxidized with depth. At 200' rock
150-160'	contains about 20 percent oxidized material.
160-240'	Pyrite content 3-4 percent.
240-320'	
320-380'	

XEROX

AMERICAN CYANAMID COMPANY

MINERAL DRESSING LABORATORY

INVESTIGATION NO. 30.
PROBLEM NO. 747.

September 12, 1950

PROGRESS REPORT - PART I

Cyanidation and Flotation Tests on a Sample of Low Grade Gold-Silver Ore Submitted by Central Comstock Mines Corporation, Virginia City, Nevada

Introduction:

At the suggestion of our Mr. O. R. Brown, two samples of ore were submitted by the subject company for metallurgical testing. The first sample had a net weight of approximately 500 pounds and was received at the Mineral Dressing Laboratory in good condition on November 16, 1949. The second sample weighed approximately 350 pounds, and was received in good condition on March 8, 1950.

Present Operations:

The subject company is currently operating an all slime cyanide plant on flotation tailings from the former Arizona Comstock operations.

Object of Investigation:

Part I

a. To determine the response of this ore to standard flotation and cyanidation procedures.

b. To identify the refractory silver mineral or minerals in the primary colloid fraction of Comstock ore.

Part II

To attempt the development of an economical process for recovering values from the primary colloid fraction of Comstock ore;

CRW - JAMcA
F.W.F.

primary colloid being defined as the minus 30 micron portion of ore crushed to minus 1 inch.

Summary and Conclusions:

1. A representative head sample of the 500-pound sample received on November 16, 1949 assayed 0.04 oz. gold and 2.07 oz. silver per ton.

2. Optimum gold and silver recoveries were obtained on ore ground to approximately minus 35 - 48 mesh. Finer grinding gave somewhat lower tailings but did not appear to be economically desirable.

3. Direct cyanidation gave tailings that assayed only .002 oz. gold and 0.38 oz. silver per ton, corresponding to an extraction of 95 % of the gold and 79 % of the silver. Owing to the presence of soluble salts, especially ferrous sulfate, reagent consumption amounted to ±40 pounds of CaO and from 1.2 - 1.5 pounds NaCN equivalent per ton. Preaeration of pulps at a high pH was required in order to keep the consumption of cyanide below 1.5 pounds per ton. We were advised, however, that it has been found impractical to cyanide crude Comstock ore directly because of the high proportion of slimes which have poor settling and filtering characteristics, and cause excessive consumption of reagents.

4. Two procedures for rejecting the objectionable slime fraction were found to give fair recoveries of gold and silver. They were:

   (a) Bulk flotation, with cyanidation of the combined bulk flotation concentrate and sand fraction of flotation tailings.

   (b) Flotation of the slimes, with cyanidation of the combined slime flotation concentrate and original sands.

   Both procedures gave final tails that assayed approximately 0.004 oz. gold and 0.7 oz. silver, which corresponded to a recovery of approximately 90 % of the gold and 65 % of the silver. Cyanide and lime consumptions were reduced to approximately 15 pounds CaO and 0.5 pounds NaCN equivalent per ton of original ore.

CRW - JAMcA
Sw.

5. In most cases, gold recoveries could be considered fairly satisfactory, but silver recoveries were low in all tests. Major silver losses occurred in the finest, minus 10-micron sized fraction of the slimes. Evidence obtained by infrasizing, heavy liquid testing, and microscopic examination of slime tailings indicated that the principal refractory silver mineral in this finest sized fraction of Comstock ore was probably argento-jarosite.

6. Further work on total Comstock ore using standardized procedures was discontinued at the advice of our Mr. O. R. Brown, who requested that the work at Stamford be confined to a research investigation on the primary colloid fraction, as previously outlined under the heading "Object of Investigation". Results of this latter investigation will be submitted separately at a later date as Part II of this report.

Other Investigations:

This investigation was conducted concurrently with, but independently of, an investigation made at Central Comstock Mines laboratory by our field engineers, Mr. O. R. Brown and Mr. E. C. Herkenhoff, working in conjunction with the subject company's Mr. H. L. Hazen. This latter work will be the subject of a separate report by Mr. E. C. Herkenhoff.

CRW-Jones
JWF.

Xerox

Virginia City- Nevada
November 1, 1950

HBC-Reno-Nevada

The following discussion is concerned with the the report dated September 12, 1950, submitted by the Laboratory of the American Cyanamid Company , and titled - Part I .

We sent that laboratory a sample carefully cut from the two ton sample of the dumps we took for our official sample. Our laboratory work has been reported under the name"Series No. 27."

Head assay

Reference		Au $	Ag $	Total $
ACC - 4 --	Head assay ----	1.40	1.83	3.23
27 - 3 ---	Head assay -----	1.74	1.71	3.45

"ACC"- 4 means page 4 of the report mentioned above from the American Cyanamid Company

" 27 - 3 " means page 3 of the Comments by H.L.Hazen on Test Series No. 27 .

In this discussion I have converted all assays to dollars, with gold figured at $35 and silver at 90¢ per oz.

American Cyanamid treated the entire sample in the tests covered by this report. We first removed the primary slime and worked on the balance of the ore, which I have named the " Mill Heads ".

Comparison of results obtained by
American Cyanamid and by Central Comstock

Reference		Gold	Silver	Total
ACC - 2	Recovery from-original ore -------------	90 %	65 %	75.83 %
27 - 53	Recovery from Mill Heads	95 %	84 %	90.00 %
27 - 49	Gold and silver in mill heads per ton of ore	96.7 %	79.40 %	89.92 %
	Recovery from original ore indicated in Test Series No. 27 --------------	91.8 %	66.7 %	80.92 %

It would appear from the preceeding that the overall recovery was increased 5 % because we discarded the primary slime before treatment.

The ACC report , page 2 , states that the consumption of lime amounted to 15 lbs per ton , even when the primary slime was discarded along with the secondary slime after flotation.

Our tests gave an overall consumption of 6.5 lbs of lime per ton of mill heads , equal to 5.85 lbs per ton of original ore. (27 - 64) . I am sure this decrease was entirely due to the additional wash given the ore by removing the primary slime before even the first flotation or grinding to 40 mesh .

Then there would be the additional saving in flotation reagents by reason of not floating the primary slime.

The ACC tests obtained a chemical consumption of only 0.5 lbs NaCN (page 2) as compared to our consumption of 1.65 lbs per ton of mill heads (page 64 of series 27) . ACC did this by blowing air through the pulp after making alkaline with lime but before adding cyanide. This would be a most important saving in operating cost and would cut almost 16¢ per ton from the costs I have reported to date. The excess air, in an alkaline pulp, oxidized the ferrous sulphate up to ferric sulphate. Ferrous iron consumes cyanide but ferric iron doesn't. (4 hrs.)

The report gives quite a detailed analysis of the sample (ACC - 4) and it shows sulphate sulphur present to the extent of 0.82 %

In their work ACC tried all the flotation reagents they could think of that might possibly increase the recover (ACC- 7) They ran the pH from very acid to very alkaline. Nothing helped.

The loss of silver occurs in the veryfinest sizes (ACC- 66) . It is most important, in treatment of this ore , that great attention be paid to the handling so that as little slime as possible is produced.

HIH

ESTIMATED
INVESTMENT AND COST SUMMARY

1/29/69

For Counter Current Cyanide Plant at CCM (Gold & Silver ore)

Gross Possible Production: 2,000,000 tons of ore @ $11.10 per Ton @ 85% recovery = $18,870,000 ←

	(175,000 TPY) 500 TPD	(262,500 TPY) 750 TPD	(350,000 TPY) 1,000 TPD
Capital Investment:			
Mill	1,500,000	2,250,000	3,000,000
Mining equipment	250,000	375,000	500,000
Property aquisition	175,000	175,000	175,000
Preparation of pit, tails dam, etc.	250,000	375,000	500,000
Subtotal	2,175,000	3,175,000	4,175,000
10% Contingency	217,500	317,500	417,500
Total	2,392,500	3,492,500	4,592,500
Daily gross bullion recovery @ $9.35 per ton milled	4,675	7,012.50	9,350
Yearly gross bullion recovery @ 350 milling days per year	1,636,250	2,454,375	3,272,500
Daily operating costs:			
Mining (4 tons waste to 1 ton ore), 5 tons @ $0.35 per ton = $1.75 per ton milled	875	1,312.50	1,750
Milling @ $3.00 per ton	1,500	2,250	3,000
Taxes & Insurance @ $0.30 per ton	150	225	300
Bullion marketing cost	50	75	100
Subtotal	2,575	3,862.50	5,150
10% Contingency	257.50	386.25	515
Total	2,832.50	4,248.75	5,665
Yearly operating cost @ 350 milling days per year	991,375	1,487,062.50	1,982,750
Yearly gross bullion recovery	1,636,250	2,454,375	3,272,500
Less Yearly operating cost	991,375	1,487,062.50	1,982,750
Net Yearly Profit (excluding amoritization, depreciation, income tax, depletion, etc.)	644,875	967,312.50	1,289,750
Years to exhaust 2,000,000 ton ore reserve	11.43	7.62	5.71

PRESENT WORTH OR PRESENT VALUE OF FUTURE INCOME USING HOSKOLD FORMULA AS FOLLOWS:

$$V_p = \frac{A}{\frac{r}{R^n - 1} + r'}$$

A = Annual earnings
r = "Safe" interest rate on reinvested capital redemption (Use 6%)
r' = Risk rate of return on invested capital (Use 20%)
n = Life of operation
R = Amount of $1 with one year's interest
Vp = Present worth or present value of future income

500 TPD, 175,000 TPY, 11.43 years

$$V_p = \frac{\$542{,}122.50}{\frac{0.06}{1.06^{11.43} - 1} + 0.20} = \frac{542{,}122.50}{0.2634} = \underline{\underline{\$2{,}058{,}172}} \leftarrow$$

750 TPD, 262,500 TPY, 7.62 years

$$V_p = \frac{\$843{,}796.25}{\frac{0.06}{1.06^{7.62} - 1} + 0.20} = \frac{843{,}796.25}{0.3073} = \underline{\underline{\$2{,}745{,}839}} \leftarrow$$

1,000 TPD, 350,000 TPY, 5.71 years

$$V_p = \frac{\$1{,}168{,}045}{\frac{0.06}{1.06^{5.71} - 1} + 0.20} = \frac{1{,}168{,}045}{0.3519} = \underline{\underline{\$3{,}319{,}253}} \leftarrow$$

2/2/69

PRESENT WORTH OR PRESENT VALUE OF FUTURE INCOME USING STRAIGHT DISCOUNT METHOD AS FOLLOWS WITH 20% RETURN OF CAPITAL INVESTMENT PER YEAR:

A = Annual Earnings

r' = Risk rate of return on invested capital (Use 20%)

Vp = Present worth or present value of future income

n = Life of operation

$$V_p = \frac{A\left[(1+r')^n - 1\right]}{(1+r')^n\, r'}$$

500 TPD, 175,000 TPY, 11.43 years

$$V_p = \frac{542{,}123\left[1.20^{11.43} - 1\right]}{1.20^{11.43} \times 0.20} = \frac{542{,}123 \times 7.036}{1.607}$$

$$V_p = \$2{,}373{,}601$$

750 TPD, 262,500 TPY, 7.62 years

$$V_p = \frac{843{,}796\left[1.20^{7.62} - 1\right]}{1.20^{7.62} \times 0.20} = \frac{843{,}796 \times 3.014}{0.8028}$$

$$V_p = \$3{,}167{,}913$$

1,000 TPD, 350,000 TPY, 5.71 years

$$V_p = \frac{1{,}168{,}045\left[1.20^{5.71} - 1\right]}{1.20^{5.71} \times 0.20} = \frac{1{,}168{,}045 \times 1.832}{0.5664}$$

$$V_p = \$3{,}778{,}013$$

Estimation of Future Annual Earnings

	500 TPD	750 TPD	1,000 TPD
	175,000 Tons/Year	262,500 Tons/Year	350,000 Tons/Year
	11.43 Years	7.62 Years	5.71 Years
Net Yearly Profit (excluding income tax, depreciation & depletion)	$ 644,875.00	$ 967,312.50	$ 1,289,750.00
Less - Depletion (50% of Net Income before depletion) - Schedule C	227,609.38	270,867.19	267,093.75
	417,265.62	696,445.31	1,022,656.25
Less - Depreciation - Schedule A	189,656.25	425,578.13	755,562.50
Taxable Income	227,609.37	270,867.18	267,093.75
Less - Income Taxes - Schedule B	102,752.50	123,516.25	121,705.00
Net Profit	124,856.87	147,350.93	145,388.75
Add back depletion & depreciation			
Depletion	227,609.38	270,867.19	267,093.75
Depreciation	189,656.25	425,578.13	755,562.50
Annual Earnings	$ 542,122.50	$ 843,796.25	$ 1,168,045.00

Schedule B
Income Tax

Income Tax on 500 TPD Milling Operation:

0.48 X $ 227,609.37	=	109,252.50
Less 0.26 X 1st $25,000	=	6,500.00
Total		$ 102,752.50

Income Tax on 750 TPD Milling Operation:

0.48 X $ 270,867.18	=	130,016.25
Less 0.26 X 1st $25,000	=	6,500.00
		$ 123,516.25

Income Tax on 1,000 TPD Milling Operation:

0.48 X $ 267,093.75	=	128,205.00
Less 0.26 X 1st $25,000	=	6,500.00
		$ 121,705.00

Schedule A
Depreciation

	500 TPD	750 TPD	1,000 TPD
Capital Investment	$2,392,500.00	$3,492,500.00	$4,592,500.00
Less property acquistion	175,000.00	175,000.00	175,000.00
	2,217,500.00	3,317,500.00	4,417,500.00
Less Salvage	50,000.00	75,000.00	100,000.00
Depreciation Basis	2,167,500.00	3,242,500.00	4,317,500.00

Yearly Depreciation: (using unit of production method)

500 TPD Mill

$$\frac{\$2,167,500}{2,000,000 \text{ tons}} = \$1.08375 \text{ per ton}$$

175,000 tons X $1.08375 = $189,656.25

750 TPD Mill

$$\frac{\$3,242,500}{2,000,000 \text{ tons}} = \$1.62125 \text{ per ton}$$

262,500 ~~175,000~~ tons X $1.62125 = $425,578.13

1,000 TPD Mill

$$\frac{\$4,317,500}{2,000,000 \text{ tons}} = \$2.15875 \text{ per ton}$$

350,000 tons X $2.15875 = $755,562.50

Schedule C
Depletion

	500 TPD 175,000 Tons/yr. 11.43 yrs.	750 TPD 262,500 Tons/yr. 7.62 yrs.	1,000 TPD 350,000 Tons/yr. 5.71 yrs.
Gross Income	1,636,250	2,454,375	3,272,500
Deductions (Operating cost)	(991,375)	(1,487,062.50)	(1,982,750)
(Depreciation)	(189,656.25)	(425,578.13)	(755,562.50)
Net Income (before depletion allowance)	455,218.75	541,734.37	534,187.50
15% of Gross Income	245,437.50	368,156.25	490,875
50% of Net Income (before depletion allowance)	227,609.38	270,867.19	267,093.75
Depletion Allowance (Each year)	227,609.38	270,867.19	267,093.75