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L. J. MANNING & ASSOCIATES LTD.

Report For

BLACKDOME EXPLORATION LTD.

ON

THE 1981 DEVELOPMENT

OF THEIR

BLACK DOME NO. 1 VEIN

BY

L.J. MANNING AND ASSOCIATES LTD.

Dated

MARCH 1982

L.J. Manning, P.Eng. (Mining)

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Ass. Rept 11046

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L. J. MANNING & ASSOCIATES LTD.

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April 2, 1982

President and Directors
Blackdome Exploration Ltd.
904, 675 West Hastings Street
Vancouver, B.C.
V6B 1N2

Dear Sirs:

As described in the attached report, your company has, in 1981, completed 3,101 feet of underground development plus 500 feet of surface stripping on the Black Dome No. 1 vein. This work has exposed 1,024 feet of vein within 1,145.7 feet of strike length of the structure. This work has indicated the presence of Possible Mill Feed Reserves of 53,171 tons grading, in troy ounces per ton, 0.65 gold and 5.8 silver or 0.78 gold equivalent (Au Eq. = gold + silver/45). This reserve has a minimum mining width of 8.64 feet and is exposed in three areas of continuous mineralization aggregating 448 feet of strike length at the 6430 level. Stoping lengths at 6430 are, from north to south, 113 feet, 180 feet, and 150 feet. Please see Studies 9 and 17F in the appendix and Drawing U002 for details on the foregoing estimates.

Metallurgical testing of a sample from these reserves indicates that recoveries of 95% gold and 87% silver or 93% of the gold equivalent may be expected from conventional flotation of the ore.

Yours truly,

L.J. Manning, P.Eng. (Mining)

LJM/ww

Enclosure

DEVELOPED RESERVES

The reported Mill Feed Reserves have been estimated by cutting, diluting and expanding a "developed" reserve of 29,721 tons with a width of 7.15 feet grading, in troy ounces per ton, 1.16 gold and 8.0 silver for a gold equivalent of 1.33. This developed reserve has been estimated by extending the sampled area-grades a distance of 40 feet from the plane of sampling.

In estimating "Probable Mill Feed", assays were cut by doubling the arithmetic averages of material in the vicinity of any spot high and applying it to the sample area of the spot high. The schedule in Study No. 12C has been applied to arrive at minimum stoping dilution. Development dilution has then been added as estimated from Study No. 7A and is equal to an additional 2.345%.

Possible Mill Feed tonnage has been estimated by increasing the depth of floors only, of all developed blocks from 40 feet to 1/2 of the continuously mineralized length of each zone, plus extending the 1230 backs to the apparent limits of mineralization as determined by "straight lining" between surface and underground exposed limits.

These reserves are contained in three blocks with an aggregate length, at the 6430 level, of 448 feet; thus 44% of the drifted vein is mineralized. Continuously mineralized lengths at the 6400 level are, from north to south, 113 feet, 180 feet, and 150 feet. Vertical continuity to surface has been established in the two southern blocks by raising and closely spaced diamond drilling of these reserves (Study No. 17F, Drawing U002).

3,800 tons grading 0.80 gold and 4.4 silver or 0.89 gold equivalent are presently stockpiled on the pad at the portal (Study No. 19D).

GROSS VALUE OF RESERVES

With gold at US $350 per ounce, such reserves have a gross value before mining, concentrating, shipping and smelting of Can. $332 per ton or Can. $17,700,000 of which $1,625,000 is in the 3,800 ton stockpile.

The foregoing reserves have been obtained from systematic chip sampling of faces available after every blast. Muck samples have been used to establish the grades of the stockpiles.

METALLURGICAL TESTING

As discussed in Study No. 20, metallurgical testing of material from a 550 pound composite made from equal quantities taken from 38 individual drift rounds, indicates a probable recovery of 95% of the gold and 87% of the silver. From heads grading, in troy ounces per ton, 0.417 gold and 4.9 silver, a concentrate was produced which graded 14 gold and 142 silver, thus indicating a concentration ratio of about 36:1. Because of the high proportion of silica (about 70%) and the low proportion of undesirable impurities, the concentrate should be very saleable and should incur few penalties. To obtain the foregoing results, a grind of 85% - 200 mesh was found necessary. Of the tailings, only about 30% consisted of sand suitable for mine hydraulic backfill.

COMPARISONS OF 1981 DEVELOPED RESERVES WITH PREVIOUS DRILL INDICATED RESERVES

Reserves developed in 1981 are compared with reserves indicated by previous diamond drilling in Study No. 18D with variations summarized in Study No. 18E. Both drill indicated polygons and developed reserve blocks are shown on Drawing No. U002. Study No. 18Dii)b) shows that Mill Feed reserves are 4.3% higher grade than ore grade polygons in the vicinity. Tonnage is 9.6% less, but this is only because blocks 1239 and 1248 have not been increased above the back to 1/2 their continually mineralized shoot length. The vertical area of the Mill Feed reserves is 22% less than that of the polygon reserve and the width is 16% greater.

It thus appears that the uncut, undiluted drill indicated reserves by Richardson Carlyle may be considered as equal to the developed Mill Feed reserves.

It is presently felt that an examination of all the drill intersections will reveal similar effects to polygons beyond the vicinity of the drift and that unless geologic conditions change, the drill indicated reserves may be used without cutting and diluting.

On considering drift intersections with material encountered by diamond drill holes near the drift, it is noted that of 7 holes within 5 meters of the plane of sampling of the drift (74, 69 F.W., 2, 13, 37, 39, and 44) only two, 13 and 44, were of reserve grade, and are in developed ore shoots. No. 39 grades about 1/2 the reserve requirement and is in an ore shoot. All others are in developed waste zones. Of five additional holes within 10 meters of the plane of sampling (72, 69 H.W., 57, 11 H.W., and 11 F.W.), only one, 11 H.W., is of reserve grade and is not correlatable with a reserve shoot. Hole 69 H.W., though not of reserve grade, was of sufficient interest that the drift was turned into the hanging wall to check for any parallel structure such as that found to contain the 1248 ore shoot.

MINE DESIGN

Disposition of Drift Indicated Reserves (Study No. 1 and Map U001 A and B)

A perusal of Study No. 1A indicates that of 100 holes assignable to the No. 1 vein, 93 hit vein material, and 7 were scouting for adjacent veins or the south extension. From the 93 holes, 108 intersections with vein material are noted.

A perusal of Study No. 1B indicates that:

a. of these 108 intersections, 63 or 58% graded a minimum of 3 grams per tonne, or 0.1 troy ounce per ton, of gold equivalent (Au Eq. = Au + Ag/35), and were therefore included in the Richardson Carlyle reserves (Appendix 1).

b. the content of metal of economic interest, as expressed by equivalent ounces, appears to be concentrated as follows: 33% over 1.0 ounce; 28% between 0.3 and 1.0 ounce; and 27.5% from 0.1 to 0.3 ounces.

c. tonnage grading 0.3 ounces and over includes 61% of the value in 69,523 tons or 23% of the reserve, and grades 0.928 equivalent ounces per ton.

d. the remaining tonnage, grading less than 0.3 ounces, includes 39% of the value in 237,975 tons or 77% of the reserve and grades 0.174 equivalent ounces per ton.

With gold evaluated at US $350 per troy ounce and with a Can./US exchange ratio of 0.82, material in c. above has a gross value of Can. $396 per ton while material in d. has a gross value of Can. $74 per ton. Material in c. would allow economic consideration of a small plant of about 200 tons per day, whereas material in d. would require a larger plant of about 800 tons per day.

A level interval of 40 meters (130 feet), had been selected by Richardson Carlyle for plotting anticipated vein geology. This level appears reasonable to the writer when considering mining the higher grade shoots and is shown on Drawing No. U001 A and B for 6300 and 6170 levels.

"Studied" Mineable Reserve

Study No. 2 considers mining all material with a minimum grade of 0.26 ounces per ton. As most of this material and most of the drilling is shown on Drawing No. U001A, only the tonnage shown on this sheet was studied for feasibility. As seen in Study No. 2 this "Studied" reserve contains 31% of the total reserve tons and 62% of the economic metal.

Application of Development Experience to Drill Indicated Reserves

The factors derived in Study No. 3 have been superceded by those derived in Study No. 18E,D,ii)b) where 1981 developed "Possible Mill Feed" has 9.4% less tons containing 5.5% less metal than the polygons above and below the drift. As the mill feed reserves have not included increasing reserves above the drift beyond 40 feet to 1/2 the strike length in the case of blocks F and G (Map U002), "Possible Mill Feed" may be said to equal Richardson Carlyle "High Grade" reserves without applying cutting and dilution factors. Study No. 4 should therefore be adjusted as follows:

1. A factor of 1 should be substituted for those factors derived in Study No. 3, i.e. Study No. 4 = Study No. 5.

2. No dilution should be applied.

3. Mill recoveries of 95% of the gold and 87% of the silver should be substituted for the recoveries of 97% and 85% used in Study Nos. 4 and 5 (see Study No. 20C).

4. Unit metal values and exchange rates for Can./US $ are probably satisfactory for life of mine considerations.

5. $3 million has been spent to end of 1981 so that $12 million should be considered as ongoing capital requirements.

Net value of both Study Nos. 4 and 5 then become Can. $8,000,000. Mine life of the studied reserve at 200 tons per day becomes reduced from 20 months to 1.54 years or 18.5 months. Operations would be much better if the usual three years could be put in sight by further diamond drilling, i.e. double the tonnage presently drill indicated.

Economics of Mining 1230 Stope

Study No. 6 has been superceded by Study No. 15 which increases the operating profitability of the 1230 stope from $4.1 million to $7.6 million after including considerations for the surface stripping results.

Study No. 7 allows order of magnitude commentary on crew, camp, equipment and supply requirements for extracting and treating the drill indicated ore as shown on Drawing No. U001A. No changes should be made to this until full feasibility studies are completed.

Study No. 8 was completed to note the effects of running plants above and below designed capacity as well as the adjustment to cut off grades with changes in metal price. As for Study No. 7, no changes should be made to these results at this time.

Disposition of Developed Reserves and Subsequent Mining Methods

On completion of the analysis of the 1981 Program, Drawing No. U002 was compiled and permitted the following observations which enforce use of mining methods contemplated in Study No. 7. 1981 developed reserves are contained in shoots within the north striking No. 1 vein which dips about 62° west. The shoots plunge to the northwest at angles from 39° to 53°. Because of these attitudes, the weakness of the walls, and the fact that the shoots tend to strike across the vein and migrate into the walls, consideration of cut and fill mining methods is recommended. As the ore requires fine grinding, little fill will be available from the tailings. It is recommended that fill be supplied by crushing of the existing dacite talus followed by quarrying of the dacite dome located at the head of 1237 raise. It is suggested that this be seasonal work using a portable crushing plant. Possibly the addition of cemented tailings sand fill to the crushed dacite would result in finished floors suitable for slushers or LHD's to work on with reduced loss of gold bearing fines.

It is presently contemplated that the 3rd or 6170 level be the track haulage level which would allow dumping directly into the coarse ore bin. The 6400 level has been driven at track grade so that track haulage may be considered on

:l if tonnage coupled with distance between shoots justifies such ion.

footwall haulage with sill and drawpoints to mill holes at haulage ı has been contemplated. Therefore raise development has been directly sill back with footwall haulage bypasses being driven in the vicinity hoots.

ɜ

ıo. 10B shows that Atomic Absorption assaying, as practised laterally in / Barrier Reef staff, was reliable up to 5 gm/tn (0.15 oz/t). Above the leached sand should be tested as it is probable that finer pulveriz- ıuld expose more of the gold to the aqua regia leach solution with uent results equivalent to fire assay results.

ɩATION COSTS 1981

No. 21C shows a contractor would have had to receive $470.55 per foot to ımpetitive with company performance. Study No. 21A shows that field costs ɜld labour + (field labour x 1.30 = supplies) + (field labour x 0.49 = ɾal) = field labour x 2.79. Administration or Vancouver office costs = d. labour x 0.13 so that total costs = field labour x 2.92. These ratios obtained while using basic hourly labour costs of $11.00, lead hands 00, mechanics $15.00, and with casual labour @ $70.00 per 10 hour day. ɜrs on contract were paid at a footage rate of $42.00 for lateral work and .00 for raising. Explosives and slickers were free.

dy No. 21B shows that at year end $333,000 had been applied to principal ng on total capital requirements of $580,000. Interest expense of $31,000 ; required for use of $380,000 worth of generator, compressors, trucks, and ɔops which are being purchased on a monthly basis. $240,000 worth of this ɹipment is with Guaranty Trust.

RECOMMENDED EXPLORATION PROGRAM FOR 1982

1. High gold content is nearly always associated with high quartz areas, but areas of high quartz don't necessarily have high precious metal content. Fractures are generally filled with limonite so that, at first glance, all rock appears the same in the vein or main shear area. Precious metals appear to be concentrated with little gradational material into walls. The portable gold analyzer being developed for North American Markets by EG & G Instruments of Toronto should be tested as its characteristics appear suitable for our deposit. It is a back packed gamma ray spectrometer and was originally developed in South Africa. It seems to be only useful where gold grades are relatively high. This instrument should permit the marking up of gold occurrences in the back so that the local controls within the shear may be identified. This has not been possible to date. Reports of tests of the instrument by companies in eastern Canada and eastern and western USA should be available soon through EG & G. One of the on-site Jarcoscoops could be used as a mobile platform.

   It is recommended that as soon as water is available, a crew be employed to wash and detail map the back in areas of interest. This to be followed by testing of the EG & G instrument by painting outlines of mineralized gold shoots on the back as delineated by the instrument, followed by sampling and fire assaying of these shoots.

   . This program should be instigated as soon as possible by the geologist who then would have "hands on" experience to enable correct interpretation of diamond drill core returns.

   EG & G Instruments may be reached by telephone at (416) 663-6230 in Downsview, Ontario.

2. Sample backs near 1248 to fill in area missed when gouge hanging wall fell in during crosscutting of the 1248 block for rib sampling purposes.

3. Continue advancing the drift to the south to examine the mineralization indicated by diamond drilling to lie within 500 feet of the present south face.

4. Advance drift on both veins in vicinity of DDH 11.

5. Open up the 130 feet of vein that was missed when the south face was driven on a false hanging wall stringer.

6. Raise on the north or 1248 stope and on any other shoots encountered to establish vertical continuity and ventilation.

7. Please note, that in lieu of drifting, short underground diamond drill holes may be used to delineate the unexplored zones of 4. and 5. above.

8. Extend surface stripping and sampling from trench 3 south past the 0.54 ounce intersections in trench 6 until overburden depth is impractical.

9. Provide diamond drill road to south end past BD 68 and 1110 south. Strip and sample vein in this area until overburden or steepness prevents such work being practical.

10. Strip and sample east vein where practical downhill from 1140 toward 1190.

11. The writer is not aware of the results of the geophysical tests made earlier this fall, but, if positive results were obtained, such surveys could be conducted as precedents to surface stripping.

12. The high grade ridge zone developed in 1981 is associated with a broad surface geochemical anomaly. The survey grids should be oriented to "mine north and south" and surveys should be extended to cover the south extension and the entire air strip and giant vein area from 1230 to 1400. This should be integrated with suitable geophysical surveys and where practical followed by surface stripping and sampling of the veins.

Reasons for Recommendations

1. The purpose of recommendation No. 1 is to develop a field technique which:

    a. provides immediate directional guidance to drift miners, dozer operators, and core loggers.

    b. provides a sound basis for finding and mapping gold mineralization controls.

    It is contemplated that resulting samples would be fire assayed in a commercial laboratory. It is quite conceivable that, with increased pulverizing time, the Blackdome Laboratory could be economically used.

    It is estimated that operating the lab would cost approximately $3,500 per month for labour reagents and power plus rental of the $30,000 lab and portion of the $20,000 power plant and water supply system. At $10,000 per month, 800 samples represents a breakeven point with present commercial lab rates of $12.50/sample. The operation of the lab would provide a much more satisfactory rate of return of results. Occasional checks of pulp splits in a commercial lab could provide adequate control for future financing reliability requirements. 800 samples requires an exploration rate from 3 to 4 drift faces or 40 feet/day of surface stripping.

2. Recommendation Nos. 2 through 8 have the prime purpose of establishing Possible Mill Feed reserves sufficient to cover payback of a 200 ton plant.

3. Recommendation Nos. 9 through 12 have the purpose of placing sufficient reserves for three years into the drill indicated category.

CERTIFICATE OF QUALIFICATIONS

I, Luard J. Manning, P.Eng. (Mining), of 945 Belvedere Drive, North Vancouver, B.C. certify as follows:

1. That I graduated from the University of British Columbia in 1951 as a Bachelor of Applied Science with a degree in Mining Engineering.

2. That I have been a member of the Association of Professional Engineers of Ontario since 1959 and a member of the Association of Professional Engineers of British Columbia since April, 1966.

3. That I have been engaged in the mining industry for over 30 years.

4. That I am, at present, the principal in the firm of L.J. Manning & Associates Ltd. of Vancouver.

5. That the studies for Blackdome Exploration Ltd. dated March 1982 were based on data made available by Blackdome Exploration Ltd. and data gathered while the property was developed under direction of the writer in 1981.

6. That I presently hold an option on 10,000 Blackdome Class A shares.

Dated at Vancouver, B.C. this 31st day of March, 1982.

L.J. Manning, P.Eng. (Mining)

ADDENDUM

ORE RESERVES

Mr. A.F. Reeve, President of Blackdome Exploration Ltd., requested the authors to make an independent uncut ore reserve calculation for the Blackdome No. 1 Vein and related mineralization.

In the calculation of reserves, it was decided to use the Polygon Method in the plane of the vein (Figure 11). The standard Polygon Method had to be modified because the drilling data are open at depth. Minimum grade was taken to be 3.4 gms/tonne "gold" with "gold" being (gms/tonne Au + 1/35 x gms/tonne Ag) across a minimum of 1.5 m. After this arbitrary assumption, which seemed reasonable to the authors, the gold and silver were calculated separately (Table 1). A specific gravity of 2.6 was used to calculate tonnages. Both diamond drill core assays and trench assays were used in the reserve calculations, but the reserves are considered to be Drill Indicated Reserves.

The assays were not cut in the calculations, and no dilution has been allowed for except to dilute to a minimum mining width of 1.5 m. It is pointed out that dilution will take place because the gold and silver do not occur regularly all across the vein. The prediction of the extent of the resulting dilution during mining will not be made more exact by additional drilling, but must be determined by drifting and raising underground.

Drill Indicated Reserves in the No. 1 Vein total 313,622 tonnes averaging 10.02 gms/tonne Au and 99.8 gms/tonne Ag (345,705 short tons of 0.29 oz/ton Au and 2.91 oz/ton Ag) across an average horizontal width of 2.32 metres. Included in the above, the South Block from Section 1180 to Section 1247 has 200,206 tonnes averaging 12.48 gms/tonne Au and 94.9 gms/tonne Ag (220,965 short tons of 0.36 oz/ton Au and 2.77 oz/ton Ag) across an average width of 2.52 metres. Less well defined Drill Indicated Reserves calculated from intersections near the No. 1 Vein total 15,218 tonnes averaging 18.79 gms/tonne Au and 182.8 gms/tonne Ag (16,774 short tons of 0.55 oz/ton Au and 5.33 oz/ton Ag) across an average horizontal width of 2.06 metres.

The total Drill Indicated Reserves in the No. 1 Vein plus the less well defined Drill Indicated Reserves total 328,839 tonnes averaging 10.43 gms/tonne Au and 103.62 gms/tonne Ag (362,479 short tons of 0.3 oz/ton Au and 3.0 oz/ton Ag) across an average horizontal width of 2.30 metres.

All polygons appear to designate material in steep veins (52°-80°) except that designated as item 67 on Section 1265.

In all cases, these reserves are open and would be increased by additional work.

P.W. Richardson

TABLE 1 - NO. 1 VEIN ORE RESERVE CALCULATION

ITEM	SECTION NO.	HOLE NO.	X-SECTION DIP	X-SECTION WIDTH MIN.	X-SECTION WIDTH HORIZ.	LONG SECT. VERT. AREA	VOLUME	@S.G.=2.6 TONNES	Au g/ton	Ag g/ton	GRAMS Au	GRAMS Ag	GRAMS **Au EQUIV.
1	1180	92	65°	1.85	2.04	1788.71	3651.20	9493.13	4.58	4.70	43,478.52	44,617.70	4.7
2	1184	86	52°	1.56	2.00	1187.09	2350.04	6110.11	31.13	31.70	190,207.91	193,690.67	31.8
3	1192	63	58°	1.50	1.77	1004.84	1777.33	4621.05	88.36	224.10	408,316.33	1,035,578.19	93.3
	1-3 incl.				(1.95)	3980.64	778.58	20,224.30	(31.74)	(62.99)	642,002.76	1,273,886.56	33.140
	*				6.40			22,293	.9258	1.837			0.966
4	1200	52	68°	1.50	1.62	822.58	1330.77	3460.01	7.84	11.64	27,126.45	40,274.47	8.1
5	1200	53	58°	1.84	2.17	1156.45	2509.14	6523.75	6.95	46.85	45,340.09	305,637.86	8.0
6	1204	4	67°	5.98	6.50	403.22	2619.49	6810.68	27.64	97.60	188,247.21	664,722.43	29.8
7	1204	5	60°	1.50	1.73	911.29	1578.40	4103.84	7.19	47.21	29,506.62	193,742.36	8.2
8	1204	6	67°	3.27	3.55	480.64	1707.42	4439.30	12.73	90.10	56,512.35	399,981.33	14.7
9	1208	25	65°	1.50	1.66	1527.42	2527.98	6572.75	1.55	73.64	10,187.77	484,017.59	3.2
10	1208	25	65°	1.50	1.66	1527.42	2527.98	6572.75	4.63	3.02	30,431.85	19,849.72	4.7
11	1210	TR 6	57°	1.58	1.88	482.26	908.55	2362.22	17.49	50.40	41,315.22	119,055.87	18.6
12	1210	76	63°	1.50	1.68	1432.25	2411.18	6269.06	3.48	32.13	21,816.34	201,424.96	4.2
13	1213	22	60°	1.73	2.00	1377.42	2751.57	7154.10	3.97	56.15	28,401.79	401,702.90	5.2
14	1214	21	65°	2.22	2.45	372.58	912.63	2372.85	3.58	22.71	8494.80	53,887.41	4.1
15	1215	TR22	65°	3.20	3.53	348.39	1230.10	3198.26	1.79	25.19	5724.88	80,564.08	2.3
16	1217	20	65°	1.50	1.66	777.42	1286.68	3345.37	8.98	29.47	30,041.45	98,588.16	9.6
17	1218	15	61°	2.36	2.70	514.51	1388.31	3609.61	5.42	7.34	19,564.07	26,494.51	5.6
18	1221	69	53°	1.50	1.88	1125.80	2114.48	5497.65	5.07	3.84	27,873.10	21,110.98	5.2
19*	1221	89*	51°	4.55	5.85	2416.12	14,145.79	36,779.06	7.68	0.75	282,463.20	27,584.30	7.7
20	1225	TR 1	76°	2.47	2.55	172.58	439.32	1142.24	106.29	209.15	121,408.49	238,899.10	110.9
21	1225	1	76°	1.70	1.75	538.71	943.84	2453.99	68.74	142.63	168,687.43	350,012.92	71.9
22	1230	TR 2	64°	2.35	2.61	261.29	683.17	1776.25	9.60	222.89	17,051.99	395,908.11	14.6
23	1230	12	64°	1.93	2.15	779.03	1672.83	4349.35	8.98	143.09	39,057.19	622,348.91	12.2
24	1230	13	64°	2.76	3.07	1130.64	3471.95	9027.07	26.07	978.75	235,335.60	8,835,240.26	47.8
25	1230	61	66°	1.50	1.64	567.74	932.20	2423.73	2.96	199.80	7174.24	484,260.92	7.4
26	1232	38	61°	4.09	4.68	870.97	4072.93	10,589.63	7.77	32.93	82,281.39	348,716.36	8.5
27	1232	57	61°	2.47	2.82	811.29	2291.15	5956.99	1.39	85.67	8280.22	510,335.68	3.3
28	1237	7	60°	1.53	1.77	148.39	262.16	681.61	57.69	412.08	39,322.34	280,879.68	66.8
29	1237	9	60°	1.50	1.73	561.29	972.18	2527.68	22.33	60.22	56,442.99	152,216.61	23.7
30	1239	39	53°	2.46	3.08	1109.67	3418.07	8886.97	4.10	12.00	36,436.57	106,643.63	4.4
*19	1221	89	51°	2.00	2.57	2416.12	6217.93	16,166.62	13.65	0.75	220,674.36	12,124.96	

TABLE 1 - NO. 1 VEIN ORE RESERVE CALCULATION (Continued)

			X-SECTION			LONG SECT.					GRAMS		
	SECTION	HOLE		WIDTH		VERT.		@S.G.=2.6	Au	Ag			**Au
ITEM	NO.	NO.	DIP	MIN.	HORIZ.	AREA	VOLUME	TONNES	g/ton	g/ton	Au	Ag	EQUIV.
31	1244	TR 4	60°	3.71	4.28	274.19	1174.61	3053.99	2.98	97.32	9100.90	297,214.74	5.1
32	1244	HW11	64°	1.50	1.67	1053.22	1757.72	4570.08	11.29	10.73	51,596.17	49,036.93	11.5
33	1244	FW11	64°	1.50	1.67	1053.22	1757.72	4570.08	4.98	5.21	22,758.98	23,810.10	5.1
34	1247	44	62°	1.94	2.20	1558.06	3423.35	8900.70	12.14	213.50	108,054.54	1,900,300.18	16.9
	4-34 incl.				(2.61)	26,566.06	69,223.69	179,981.64	(10.31)	(98.54)	1,856,036.20	17,734,463.05	12.[illegible]
	*				8.54			198,394	.3007	2.874			0.3046
	1-34 incl.				(2.52)	30,546.70	77,002.28	200,205.94	(12.48)	(94.94)	2,498,038.96	19,008,349.61	14.587
	*				8.26			220,965	.3639	2.769			0.4255
35	1255	40	64°	3.13	3.48	1040.32	3622.86	9419.43	4.15	243.08	39,090.62	2,289,674.11	9.6
36	1260	78	67°	1.50	1.63	1100.00	1792.49	4660.49	0.16	168.52	745.68	785,385.10	3.9
37	1265	83	60°	1.50	1.73	1869.35	3237.81	8418.30	6.45	26.40	54,298.06	222,243.22	7.0
38	1265	83	60°	1.50	1.73	1869.35	3237.81	8418.30	5.10	34.32	42,933.35	288,916.19	5.9
39	1275	TR19	69°	1.98	2.12	458.06	971.48	2525.86	6.17	273.70	15,584.55	691,327.71	12.3
40	1292	34	75°	1.50	1.55	1370.96	2128.98	5535.36	2.78	47.93	15,388.29	265,309.65	3.8
41	1292	98	79°	1.50	1.53	1725.80	2637.15	6856.59	3.79	5.22	25,986.49	35,791.43	3.9
42	1303	8	58°	1.92	2.26	548.39	1241.57	3228.08	26.40	230.74	85,221.18	744,846.03	31.5
	35-42 incl.				(1.89)	9982.23	18,870.16	49,062.41	(5.69)	(108.50)	279,248.23	5,323,493.45	8.101
	*				6.20			54,081	.1660	3.165			0.236
	1-42 incl.				(2.37)	40,528.93	95,872.44	249,268.35	(11.14)	(118.97)	2,777,287.19	29,655,336.51	13.786
	*				7.76			274,769	.3250	3.470			0.402
43	1328	28	71°	1.50	1.59	638.71	1013.27	2634.50	5.31	8.23	13,989.20	21,681.94	5.5
44	1328	28	71°	1.50	1.59	638.71	1013.27	2634.50	4.37	8.16	11,512.77	21,497.52	4.6
45	1328	29	71°	1.50	1.59	1061.29	1683.66	4377.52	1.41	39.35	6172.31	172,255.60	2.3
46	1334	SSTR	75°	1.51	1.56	509.68	796.77	2071.59	3.43	Tr.	7105.56	0.00	3.4
47	1336	SSTR	74.5°	1.67	1.73	412.90	715.57	1860.48	11.65	58.28	21,674.55	108,428.57	12.9
48	1337	24	74°	3.86	4.02	1317.74	5291.46	13,757.79	3.46	14.98	47,601.96	206,091.73	3.8
49	1338	SSTR	74.5°	2.49	2.58	70.97	183.38	476.80	9.22	40.11	4396.10	19,124.48	10.1
50	1338	23	75.5°	1.56	1.61	372.58	600.35	1560.90	7.20	24.00	11,238.50	37,461.67	7.7
51	1339	SSTR	74.5°	1.94	2.01	119.35	240.28	624.72	12.00	61.70	7496.67	38,545.36	13.4
52	1340	SSTR	75°	2.36	2.44	120.97	295.56	768.46	15.43	85.70	11,857.28	65,856.72	17.3
53	1340	SS 5	75°	1.50	1.50	762.90	1184.72	3080.27	5.31	31.96	16,356.22	98,445.35	6.0

TABLE 1 - NO. 1 VEIN ORE RESERVE CALCULATION (Continued)

ITEM	SECTION NO.	HOLE NO.	X-SECTION DIP	X-SECTION WIDTH MIN.	X-SECTION WIDTH HORIZ.	LONG SECT. VERT. AREA	VOLUME	@S.G.=2.6 TONNES	Au g/ton	Ag g/ton	GRAMS Au	GRAMS Ag	**Au EQUIV.
54	1341	SSTR	74.5°	2.20	2.28	162.90	371.91	966.96	9.26	54.85	8954.01	53,037.52	10.5
55	1342	SSTR	75°	1.50	1.55	146.77	227.92	592.60	5.85	48.77	3466.88	28,900.87	6.9
56	1343	SSTR	74°	1.62	1.69	122.58	206.58	537.11	6.86	65.13	3684.60	34,982.22	8.3
57	1344	SSTR	75°	2.65	2.74	187.10	513.31	1334.59	6.51	68.56	8688.21	91,499.78	8.0
58	1345	SSTR	74.7°	2.82	2.92	133.87	391.39	1017.60	10.28	61.70	10,460.94	62,786.00	11.7
59	1346	SSTR	73.75°	1.50	1.56	64.52	100.81	262.10	6.50	6.50	1703.64	1703.64	[illegible]
60	1346	SS 3	73.75°	1.50	1.56	1175.80	1837.09	4776.44	7.90	43.97	37,733.87	210,020.04	8.9
61	1346	90	80°	2.78	2.82	1585.48	4475.63	11,636.64	4.84	10.47	56,321.32	121,835.58	5.1
62	1346	94	80°	1.80	1.83	1256.45	2296.50	5970.90	8.05	3.00	48,065.72	17,912.69	8.1
63	1349	SSTR	74.5°	1.94	2.01	651.61	1311.83	3410.77	7.83	66.18	26,706.32	225,724.69	9.3
	43-63 incl.				(2.15)	11,512.88	24,751.24	64,353.24	(5.67)	(25.45)	365,186.44	1,637,791.98	6.24
	*				7.05			70,937	0.1654	0.742			0.182
	1-63 incl.				(2.32)	52,041.81	120,623.68	313,621.59	(10.02)	(99.78)	3,142,473.63	31,293,128.49	12.237
	*				7.61			345,705	0.2920	2.910			0.356

Possible Additional Reserves on Structures Near No. 1 Vein

ITEM	SECTION NO.	HOLE NO.	X-SECTION DIP	X-SECTION WIDTH MIN.	X-SECTION WIDTH HORIZ.	LONG SECT. VERT. AREA	VOLUME	@S.G.=2.6 TONNES	Au g/ton	Ag g/ton	GRAMS Au	GRAMS Ag	**Au EQUIV.
64	1184	86FW	52°	1.84	2.33	1187.09	2771.85	7206.80	21.85	10.50	157,468.65	75,671.44	22.1
65	1247	43FW	74°	1.62	1.69	491.93	829.04	2155.51	3.33	52.00	7177.85	112,086.52	4.5
66	1260	78HW	70°	1.50	1.60	975.80	1557.64	4049.86	3.79	108.48	15,348.96	439,328.40	6.2
	(Flat Vein)				(Vert)								
67	1265	84HW	15°	3.81	3.81	182.26	694.41	1805.47	58.68	1193.48	105,944.84	2,154,789.42	85.2
	64-67 incl.				(2.06)	2837.08	5852.94	15,217.63	(18.79)	(182.81)	285,940.29	2,781,875.78	22.85
	*				6.76			16,774.40	0.5480	5.332			0.66
	1-67 incl.				(2.30)	54,878.89	126,476.62	328,839.22	(10.43)	(103.62)	3,428,413.92	34,075,004.27	12.72
	*				7.55			362,479	0.3041	3.022	110,227.75	1,095,553	0.37
Northern Miner 02/04/81								313,000	0.35	3.21	109,550	1,004,730	0.42

*Width in Feet, Weight in short dry tons, Grades in Troy Ounces/S.D.T.

**Gold Equivalent = g/ton Au + g/ton Ag ÷ 45