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REPORT ON THE PRELIMINARY COMPUTER EVALUATION OF THE IN-SITU MINERAL RESERVES AT THE AR/HN PROPERTY OF THE REA GOLD CORPORATION

Prepared for:

SENTINEL MANAGEMENT CORPORATION
#304 - 1155 West Pender Street
Vancouver, B.C.
V6E 2P4

Prepared by:

STEFFEN ROBERTSON AND KIRSTEN (B.C.) INC.
#801 - 1030 West Georgia Street
Vancouver, B.C.
V6E 2Y3

April, 1986

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DECLARATION

1) I, Peter I. Clarke, am a professional engineer registered with the Province of British Columbia and an associate consultant with Steffen, Robertson and Kirsten (B.C.) Inc.

2) I am a graduate of the University of Leeds, 1975 with a B.Sc. in Mining. I have practised my profession for 10 years.

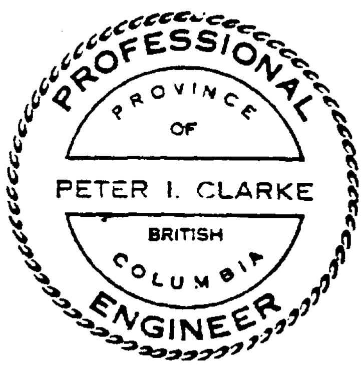
3) I have worked with Mr. Peter Franklin on the preparation of this report.

4) The report is based on exploration data provided by Minorex Consultants Ltd. of Kamloops, B.C.; Assay certificates provided by Kamloops Research and Assay Laboratories; and additional data provided by Rea Gold Corporation of Vancouver, B.C.

I declare that I have no direct or indirect interest in the Rea Gold property and will not receive any interest, direct or indirect in the said property. I do not beneficially own directly or indirectly securities in Rea Gold Corporation nor any affilitates of the company.

Signed Peter I. Clarke.

Peter I. Clarke, P.Eng.



DECLARATION

1) This report was written by myself, Peter J. Franklin, in conjunction with Peter I. Clarke. I am an employee of Steffen Robertson and Kirsten (BC) Inc., Consulting Engineers, resident at 3673 Sunset Blvd., North Vancouver, B.C.

2) I graduated as a Mining Engineer with an Associateship of the Camborne School of Mines, England, in 1976 and have been practising as a Mining Engineer continuously for the last ten years.

3) I am registered as an Engineer in Training in the Association of Professional Engineers in the Proivnce of British Columbia, and also as a Chartered Engineer in the United Kingdom.

4) This report is based on exploration data provided by Minorex Consultants Ltd. of Kamloops, B.C.; Assay certificates provided by Kamloops research and Assay Laboratories of Kamloops, B.C.; and additional data provided by Rea Gold Corporation of Vancouver, B.C. No site inspection was made.

5) The date of this report is April 25, 1986.

6) I declare that I, either directly or indirectly, have not received or do not expect to receive any interest, direct or indirect, in the property of the Rea Gold Corporation or any affiliate of that company, and that I do not beneficially own, directly or indirectly, any securities of the Rea Gold Corporation or any affiliate of that organization.

Signed, this 29th day of April, 1986

Peter J. Franklin

8.0 RECOMMENDATIONS FOR FURTHER WORK

The work carried out here under the project terms of reference was sufficient to define probable ore reserves for the three lenses 97+00, 98+00 and 100+00.

A report detailing the geology of the deposit and the exploration program was prepared in parallel to this by Minorex Consultants Ltd., and only was made available at the conclusion of the study. This report contained detailed geological sections and structural plans of the three lens that differed slightly to those used to construct the three dimensional block models. In addition, all drillholes have now been accurately surveyed and located on plan.

It is recommended that, in order to upgrade the reserve category from 'probable' to 'proven' the following steps be taken:

- o update the drillhole database to reflect the accurately surveyed hole locations
- o re-construct the geological rock-type block model using the definitive cross-sections prepared by Minorex Consultants to establish consistency between reports
- o re-model the lenses using separate three dimensional block models for each lens, with smaller block sizes to reduce possible dilution and ore loss
- o re-calculate ore-reserves for the new models

This work should be undertaken if any decisions are made to proceed with further evaluation work on this property.

CUT-OFF GRADE	WASTE SHORT TONS	ORE SHORT TONS	STRIP RATIO TONS:TONS	GRADES				
				Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
0.100 oz/t	410930	8260	49.7	0.307	1.200	1.245	1.266	0.181
0.075 oz/t	409090	10100	40.5	0.267	1.069	1.095	1.098	0.162
0.050 oz/t	404340	14850	27.2	0.200	0.861	0.868	0.848	0.127

TABLE 26 - 100+00 PRELIMINARY PIT RESULTS

CUT-OFF GRADE	WASTE SHORT TONS	ORE SHORT TONS	STRIP RATIO TONS:TONS	GRADES				
				Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
0.100 oz/t	1055880	61860	17.1	0.317	4.015	3.131	3.431	0.560
0.075 oz/t	1049860	67880	15.5	0.297	4.026	2.923	3.178	0.521
0.050 oz/t	1045590	72150	14.5	0.283	3.903	2.803	3.052	0.498

TABLE 25 - 97+00 PRELIMINARY PIT RESULTS

7.0 PIT GENERATION

7.1 Methodology

Open pits were computer generated for both the 97+00 lens and the 100+00 lens. The pits were generated using the PC-MINE pit generation facilities, where pit slopes are projected upwards to topographic surfaces from polygons defining pit bases on specific elevations. All pits were generated with an assumed slope angle of 45° in all rock-types. Mining reserves were then calculated from these pits and are presented below.

7.2 97+00 Pit

A pit was designed for this lens that extracted all material above a gold cut-off grade of 0.05 oz per ton.

To generate this pit, a series of six polygons were drawn up around this lens on the 1547 m, 1541 m, 1535 m, 1529 m, 1523 m and 1517 m elevations respectively. These polygons were designed as progressively deeper pit bases around the lens and included a minimum operating width for vehicles and mining equipment of 25 metres. The polygons are illustrated in Figure 3.

Total tonnage, grades and stripping ratios for this pit are presented in Table 25 and a mid bench contour map of the pit in Figure 4.

7.3 100+00 Pit

A second pit was designed for the 100+00 lens. This pit was designed in the same fashion as the 97+00 pit, but utilized a cut-off grade of 0.075 oz/gold. A lower cut-off grade of 0.05 oz/ton gold was considered but, due to the very low grade and disseminated nature of the ore bearing ground each side of this lens, lead to a unacceptable pit design in terms of pit shape and waste stripping.

Again, six polygons were used to define successively deeper bases, on the 1505m, 1499m, 1493m, 1487m, 1481m and 1475m elevations. These polygons were drawn around all mineralized areas and again constrained to a minimum operating width of 20 to 25 metres. The polygons are illustrated in Figure 3.

Total tonnages, grades and stripping ratios for this pit are presented in Table 26 and a mid bench contour map of the pit in Figure 6.

CUT OFF GRADE oz/ton Au	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
0.050	18580	0.163	0.469	0.567	0.477	0.085
0.070	16380	0.172	0.507	0.559	0.464	0.079
0.100	15140	0.179	0.513	0.581	0.454	0.069

TABLE 24 - PROBABLE RESERVES BELOW 100+00 LENS

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	580	2910	0.398	1.366	0.148	0.086	0.032
Semi-Massive Sulphide	6662	-	-	-	-	-	-	-
Massive Barytes	7770	-	-	-	-	-	-	-
Disseminated Mineralization	9999	3200	10350	0.182	0.626	0.509	0.345	0.043
TOTAL PROBABLE RESERVES		3780*	13270*	0.230	0.788	0.430	0.288	0.041

TABLE 23 - PROBABLE RESERVES FOR 100+00 LENS; CUT-OFF GRADE = 0.100 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	720	3580	0.340	1.182	0.194	0.133	0.034
Semi-Massive Sulphide	6662	-	-	-	-	-	-	-
Massive Barytes	7770	-	-	-	-	-	-	-
Disseminated Mineralization	9999	4060	13120	0.163	0.594	0.495	0.330	0.049
TOTAL PROBABLE RESERVES		4770*	16710*	0.201	0.721	0.430	0.288	0.046

TABLE 22 - PROBABLE RESERVES FOR 100+00 LENS; CUT-OFF GRADE = 0.075 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	1530	7610	0.192	0.986	0.202	0.141	0.043
Semi-Massive Sulphide	6662	-	-	-	-	-	-	-
Massive Barytes	7770	-	-	-	-	-	-	-
Disseminated Mineralization	9999	6850	22160	0.120	0.611	0.422	0.288	0.054
TOTAL PROBABLE RESERVES		8370*	29780*	0.138	0.707	0.366	0.250	0.051

TABLE 21 - PROBABLE RESERVES FOR 98+00 100 LENS; CUT-OFF GRADE = 0.05 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	8140	40560	0.191	2.153	2.787	3.125	0.681
Semi-Massive Sulphide	6662	2790	11530	0.190	2.121	2.696	3.138	0.611
Massive Barytes	7770	1980	9640	0.218	2.384	1.952	2.470	0.436
Disseminated Mineralization	9999	14170	45930	0.180	1.954	2.601	2.319	0.992
TOTAL PROBABLE RESERVES		27090*	107650*	0.189	2.085	2.622	2.724	0.785

TABLE 20 - PROBABLE RESERVES FOR 98+00 LENS; CUT-OFF GRADE = 0.100 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	9400	46830	0.177	2.014	2.727	3.074	0.667
Semi-Massive Sulphide	6662	3650	15060	0.166	1.852	2.382	2.706	0.572
Massive Barytes	7770	1980	9640	0.218	2.384	1.952	2.470	0.436
Disseminated Mineralization	9999	17500	56720	0.163	1.822	2.504	2.238	0.932
TOTAL PROBABLE RESERVES		32530*	128250*	0.172	1.938	2.529	2.603	0.756

TABLE 19 - PROBABLE RESERVES FOR 98+00 LENS; CUT-OFF GRADE = 0.075 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	10430	51980	0.165	1.881	2.656	2.997	0.642
Semi-Massive Sulphide	6662	4410	18220	0.148	1.662	2.172	2.488	0.509
Massive Barytes	7770	1980	9640	0.218	2.384	1.952	2.470	0.436
Disseminated Mineralization	9999	20790	67360	0.147	1.659	2.306	2.107	0.819
TOTAL PROBABLE RESERVES		37620*	147210*	0.158	1.785	2.389	2.480	0.693

TABLE 18 - PROBABLE RESERVES FOR 98+00 LENS; CUT-OFF GRADE = 0.05 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	6580	32760	0.371	3.969	3.948	4.263	0.691
Semi-Massive Sulphide	6662	460	1860	0.195	2.430	1.727	0.849	0.315
Massive Barytes	7770	3440	16750	0.267	4.390	2.192	2.615	0.413
Disseminated Mineralization	9999	3240	10490	0.255	3.839	2.330	2.594	0.427
TOTAL PROBABLE RESERVES		13710*	61860*	0.317	4.015	3.131	3.431	0.560

TABLE 17 - PROBABLE RESERVES FOR 97+00 LENS; CUT-OFF GRADE = 0.100 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	6630	32980	0.369	3.958	3.952	4.250	0.698
Semi-Massive Sulphide	6662	550	2230	0.178	2.208	1.745	0.767	0.322
Massive Barytes	7770	3940	19160	0.244	4.559	1.998	2.341	0.371
Disseminated Mineralization	9999	4170	13500	0.218	3.739	1.918	2.147	0.355
TOTAL PROBABLE RESERVES		15270*	67880*	0.297	4.026	2.923	3.178	0.521

TABLE 16 - PROBABLE RESERVES FOR 97+00 LENS; CUT-OFF GRADE = 0.075 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	6680	32980	0.369	3.958	3.952	4.250	0.698
Semi-Massive Sulphide	6662	600	2420	0.169	2.092	1.751	0.721	0.326
Massive Barytes	7770	4090	19890	0.237	4.446	1.955	2.285	0.361
Disseminated Mineralization	9999	5200	16850	0.187	3.415	1.708	1.947	0.311
TOTAL PROBABLE RESERVES		16500*	72150*	0.283	3.903	2.803	3.052	0.498

TABLE 15 - PROBABLE RESERVES FOR 97+00 LENS; CUT-OFF GRADE = 0.05 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	16290	81160	0.269	2.754	3.017	3.303	0.624
Semi-Massive Sulphide	6662	3250	13380	0.191	2.165	2.562	2.820	0.570
Massive Barytes	7770	5420	26400	0.249	3.657	2.100	2.562	0.423
Disseminated Mineralization	9999	23760	76990	0.192	1.846	2.024	1.854	0.666
TOTAL PROBABLE RESERVES		48720*	197920*	0.231	2.481	2.478	2.608	0.610

TABLE 14 - PROBABLE RESERVES FOR WHOLE MODEL; CUT-OFF GRADE = 0.100 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	17780	88560	0.254	2.612	2.946	3.211	0.614
Semi-Massive Sulphide	6662	4190	17280	0.168	1.899	2.300	2.457	0.540
Massive Barytes	7770	5920	28810	0.235	3.830	1.979	2.385	0.394
Disseminated Mineralization	9999	29190	94580	0.172	1.773	1.919	1.757	0.626
TOTAL PROBABLE RESERVES		57070*	229220*	0.211	2.365	2.352	2.450	0.586

TABLE 13 - PROBABLE RESERVES FOR WHOLE MODEL; CUT-OFF GRADE = 0.075 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

	ROCK TYPE	VOLUME m3	SHORT TONS	Au oz/t	Ag oz/t	Pb %	Zn %	Cu %
Massive Sulphide	6660	19710	98200	0.235	2.426	2.773	3.024	0.578
Semi-Massive Sulphide	6662	5000	20630	0.150	1.713	2.123	2.282	0.487
Massive Barytes	7770	6070	29540	0.231	3.772	1.950	2.346	0.387
Disseminated Mineralization	9999	36840	119370	0.149	1.587	1.690	1.578	1.526
TOTAL PROBABLE RESERVES		67620*	267720*	0.190	2.145	2.149	2.247	0.527

TABLE 12 - PROBABLE RESERVES FOR WHOLE MODEL; CUT-OFF GRADE = 0.05 OZ/TON GOLD

*NOTE: Volume and tonnage totals may not exactly summate due to rounding.

- average spacing of assayed intersections within the defined lenses is 25 metres both along strike and down dip.

- all drillholes with intersections through lenses have been logged and assayed (with the exception of drillhole RG-10 where 100% core loss was experienced in the massive sulphide horizon).

- sufficient reserve calculations have been carried out to define preliminary quantities and qualities of reserve.

6.3 Probable in-situ reserves

Probable in-situ reserves are presented individually for the total area modelled, the 97+00 lens, 98+00 lens and the 100+00 lens. The reserves were determined by rock-type and cut-off grade for three separate cut-off grades of 0.05 oz/ton gold, 0.075 oz/ton gold and 0.1 oz/ton gold and are presented in tables 12 through 24 in the following pages.

Areas of the model defining the lenses are as follows:

- 97+00 Lens

Model	Columns	61-90
Model	Rows	58-90
Model	Levels	1-20

- 98+00 Lens

Model	Columns	36-60
Model	Rows	30-80
Model	Levels	1-20

- 100+00 Lens

Model	Columns	1-30
Model	Rows	3-70
Model	Levels	1-40

A further area of mineralization directly below the 100+00 Lens has been identified and has been included into the total probable reserve statements. The tonnages and grades of this increment are not included in the separate tabulations and are summarized in Table 24.

Detailed printouts of the reserve statements produced by the computer system are shown in the attached Annexure to this report.

6.0 IN-SITU RESERVES

6.1 Methodology

In-site reserves were calculated from the three dimensional model by categorizing each block volume and tonnage according to grade and rock type. Individual reserves were determined for the whole property and individually for each lens.

Prior to calculating a reserve, an estimated surface topography was modeled by constructing a gridded model. No detailed topographic information (ie. contour maps) was available, therefore drillhole collar co-ordinates and elevations were used to construct this model.

6.2 Categorization of Reserves

In-situ reserves can be broadly grouped into four categories as follows:

- drill indicated - zones of mineralization of ore giving potential, indicated in location and approximate extent by exploration drilling.

- possible - zones of mineralization of ore giving potential, defined in location and extent on one or more sides by exploration drilling and assay data, with sufficient correlation possible between data to indicate continuity and extent.

- probable - zones of mineralization of ore giving potential, defined thoroughly in location and extent on two or more sides by exploration drilling and assay data, with sufficient correlation between data and analysis data to define preliminary quantities and qualities of the reserve.

- proven - zones of mineralization of ore giving potential, defined thoroughly in location and extent on all sides by exploration drilling and assay data, with complete correlation between data and sufficient analysis data to accurately define quantities and qualities of the reserve.

The reserve estimates described in the following sections could be defined as probable for the following reasons:

- sufficient exploration data was available to define all three lenses on three sides to a depth of 100 metres below ground surface.

- correlation of geology is possible between intersections of mineralized zones although complete geologists reports were unavailable at the time of the reserve study.

direction, and then dips the major axis into the plane of the lenses. The shape and orientation of the search volume then becomes a flattened sphere in the plane of the lenses, with the degree of flattening controlled by the vertical anisotropy factor.

Tonnage-grade curves where constructed after each run and also sketches made of the block grade distributions along section 97+00. The results of these tests indicated that Run #3 provided the most optimal results in terms of grade distributions and also contained the most ounces of gold.

Five grade models, for gold, silver, lead, zinc and copper respectively, were calculated using the inverse distance squared technique and the parameters defined above. These models then formed the basis for all in-situ ore reserve statements.

- Horizontal and vertical anisotropy angles and factors - rotation and dip angles and factors designed to alter the shape of the spherical search volume to take into account orebody shape and structure, and the presence (if any) of grade trends.

- Mineralized rock-types - a list of rock-type codes designating model blocks to be assigned grade values. Blocks with any other rock-type codes have zero grade values assigned to them.

To obtain a suitable values for these parameters a series of test models were constructed after a set of base parameters were defined. The various parameters used are summarized in Table 11 and explained more fully below.

	BASE RUN	RUN 2	RUN 3*	RUN 4
Range	30 m	30	30	30
Minimum Samples	2	2	2	2
Maximum Samples	20	20	20	20
Horizontal Rot.	0°	90°	90°	90°
Vertical Rot.	0°	-45'	-45°	-45°
Horizontal Factor	1	1	1	1
Vertical Factor	1	3	10	20

*Final modelling parameters selected.

TABLE 11 - TEST PARAMETERS FOR GRADE MODELLING

The range of 30 metres was chosen because it is slightly greater than the spacing between drillhole section lines. This will ensure that all blocks will fall within range of at least two section lines of drillhole data.

The minimum number of samples was set at 2 to restrict the overestimation of grade values in areas with little data (i.e., at the edges of the deposits).

The maximum number of samples was set at 20 to ensure sufficient samples were used when blocks were surrounded by a large number of samples. However, in most cases this maximum number was never reached.

The base run was set with isotropic conditions.

This is not considered to be the best condition due to the narrow, almost tabular nature of the lenses. Isotropic modelling in this case incurs dilution of grade values with lower values.

All other runs set the horizontal rotation angle and vertical dip angle to 90 degrees and -45 degrees respectively, with the horizontal factor set at 1 and the vertical factor set at a number greater than 1. This effectively compresses the spherical search volume, rotates it so that its major axis is in the approximate dip

Detailed statistical reports for gold, silver, lead, zinc and copper are presented in the attached printouts. A summary of the pertinent statistics for each mineral are presented in Table 10 below.

MINERAL	NO. OF SAMPLES	MEAN	MEDIAN	MINIMUM	MAXIMUM	STD. DEVIATION	VARIANCE
Gold	175	0.122	0.665	0.011	1.320	0.198	0.031
Silver	199	1.296	14.800	0.100	29.500	2.884	8.320
Lead	185	1.349	5.950	0.100	11.800	2.100	4.410
Zinc	209	1.547	6.600	0.100	13.100	2.481	6.153
Copper	90	0.573	4.095	0.100	8.100	0.933	0.871

TABLE 10 - SUMMARY STATISTICS OF ASSAY DATA

Initial attempts at geostatistical analysis were unsuccessful. A downhole experimental semi-variogram was calculated for the gold assays and this showed slight indications of a spherical variogram model. However, there were insufficient samples to investigate this further and also insufficient assay samples to calculate three dimensional experimental semi-variograms.

5.3 Three Dimensional Grade Modelling Techniques

The three dimensional grade models were all constructed using a distance weighting technique whereby weighted average grades were calculated, for each block designated as mineralized, by weighting samples by the inverse of the distance squared of samples from each block centre.

Certain other parameters were defined during the modelling procedure as follows:

- Search range - the radius of a sphere that defines a search volume for inclusion or exclusion of samples/assays into the weighted average calculation. The shape of this spherical volume may be changed by the application of anisotropy parameters (e.g.: a sphere is isotropic, an ellipsoid or oblate spheroid is anisotropic).

- Minimum number of samples - the minimum number of samples that must be present within the search volume before a block grade value is calculated.

- Maximum number of samples - the maximum number of samples to be used for the calculation of a block grade. If there are more than this maximum number present within the search volume, the samples are sorted into order of increasing distance and the closest samples used.

- Horizontal and vertical anisotropy angles and factors - rotation and dip angles and factors designed to alter the

5. GRADE MODELLING

5.1 Methodology

To calculate an ore reserve from a three dimensional block model, each block of a mineralized rock-type must be allocated grade values for each mineral being modelled. Grade values are allocated to each block by using all the sample assay values surrounding the block, weighted in some manner, and subject to a set of constraints. These constraints are:

- spatial distribution of samples and assays
- statistical distribution of samples and assays
- shape and dimensions of the deposit
- separation or distance apart of the samples and assays
- mineralization and geology of the deposit

The first two constraints can be investigated by statistical and geostatistical analyses of drillhole, sample and assay data. The results of these investigations determine the weighting technique to be used to calculate average grade values in each block. The latter three constraints are dictated by physical characteristics of the deposit and are used to determine certain modelling parameters that affect the weighting or interpolation procedures.

5.2 Assay Statistics and Geostatistics

Assay values were extracted from the drillhole database and from the trench data for each mineral in turn and subjected to a statistical analysis.

At this stage, cut-off values were applied to the assays, as one set from Corporation Falconbridge, contained background assays for most core, and the second set, from Minorex/KRAL did not. The cut-off values applied were:

MINERAL	CUT-OFF	UNITS
Gold	0.01	oz/ton
Silver	0.1	oz/ton
Lead	0.1	percent
Zinc	0.1	percent
Copper	0.1	percent

TABLE 9 - CUT-OFF VALUES APPLIED TO RAW ASSAY DATA

The individual data sets for each mineral were then input into a statistical analysis program, and detailed classical statistics (means, standard deviations, variances, etc.) calculated, together with frequency distribution tables and histogram plots.

The following values were used:

MATERIAL	ROCK-TYPE	KRAL NUMBER	S.G. (TONNE/BCM)	DENSITY (S.TONS/BCM)
Massive sulphide	6660	27615	4.52	4.98
Semi-massive sulphide	6662	*1	3.78	4.14
Massive barytes	7770	27627	4.42	4.87
Disseminated mineralization	9999	27530	2.94	3.24
Chert Breccia	-	27521	3.00	3.31

*1 Average between 27615 and 27521.

TABLE 7 - DENSITY VALUES USED FOR MODELLING

4.8 COMPARISON OF DIGITIZED ROCK-TYPES AND MODEL ROCK-TYPES

To confirm that the block size selected was suitable and to check if any dilution or loss of mineralized material was taking place in the creation of the block model, a comparison was made between the digitized rock-types and the modelled rock-types.

The results of this tabulation, shown in Table 8 indicate a difference between the two techniques of 18226 tons, or 3 percent, for mineralized rock-types.

The modelled tonnages are consistently slightly lower than the digitized tonnages, with the more significant differences occurring in the semi-massive and massive barytes rock-types. These rock types occur in predominantly narrow zones (less than 3 metres) and are likely to be less effectively modelled by the three metre wide by five metre long blocks.

This would indicate that the selected block size is marginally too big, and modelling resolution would improve slightly with a smaller block (say 2 m by 3 m). However, the overall error is estimated to be a 3 percent loss, which is considered satisfactory for this preliminary ore reserve evaluation study.

ROCK-TYPE	6660	6662	7770	9999	TOTAL
Polygon area (m2)	10174	2951	2312	35976	51413
Volume	32155	8855	7710	110434	159154
Density	4.98	4.14	4.87	3.24	-
Tonnage	160132	36660	37548	357806	592146
Model Tonnage	157140	33640	32970	350170	573920
Difference	-2992	-3020	-4578	-7636	-18226
Percent	-1.87	-8.23	-12.19	-2.13	-3.08

TABLE 8 - COMPARISON OF DIGITIZED AND MODEL TONNAGES

In summary, the block sizes were:

LEVELS	COLUMNS		ROWS		HEIGHT
	LENGTH	NUMBER	WIDTH	NUMBER	
1-5	5	90	3	90	5
5-50	5	90	3	90	3

TABLE 6 - INDIVIDUAL BLOCK DIMENSIONS

A plan view of the block model location and size is presented in Figure 1.

4.5 Transposition of Sectional Data to Level Plans

In order to build the geological block model the interpreted geology on sections was transposed onto level plans. 50 level plans were constructed (1 for each level) showing surface topography, overburden, mineralized rock-types and un-mineralized rock-types.

These level plans were then digitized (a technique to transfer map data in the form of polygons into a computer database) and the polygons representing each rock-type boundary were allocated a corresponding rock-type code and then loaded into the PC-MINE database and checked.

4.6 Construction of a Three Dimensional Rock-Type Model

The three dimensional rock-type model consists of the block model, as defined in section 4.4, with a single rock-type code allocated to each block. This is done by overlaying each level in turn by the rock-type polygons digitized from the respective level plan. The centre point of each block is then tested to determine the polygon in which it lies, and the block is then allocated the rock-type code belonging to the appropriate polygon.

A detailed printout for each level in the block model, itemizing the polygons, rock-type codes and number of blocks filled is presented as an annexure to this report.

4.7 Construction of a Three Dimensional Density Model

To calculate block tonnages, each rock-type was assigned a density value, and a second block model of equal size and dimensions was generated with a density value in each block.

The density values for each rock-type used were determined from specific gravity testing performed on core samples by Kamloops Assay and Research Laboratory. The specific gravity values were then modified to density values in short tons per bank cubic metre by multiplying by 1.1025.

Each section plot showed the following information:

- drillhole location and trace projected horizontally onto the plane of the section
- drillhole lithology by geological code
- assays for gold and silver in oz/ton
- computer generated surface topography
- plan view of the section line with drillhole locations

A preliminary geological interpretation was then performed using the cross-sections. Major lithological units and geological structures were plotted onto the sections by hand, with reference to the rock-type codes defined above. Particular care was given to the definition of mineralized zones.

Three primary areas of mineralization were defined on these sections and are referred to as:

- 97+00 lens - a massive sulphide and massive barytes lens striking parallel to the baseline, with a surface expression projecting down dip at 45° to the north-east to a depth of approximately 40 metres. The lens ranges from 2 to 8 metres in thickness and is present in sections 96+75, 97+00 and 97+25.
- 98+00 lens - a massive and semi-massive sulphide lens striking parallel to the baseline, with no surface expression. The lens dips at approximately 55° to the northeast from a depth of 20 metres to an interpreted depth of 100 metres. The lens ranges in thickness from 2 to 8 metres of sulphide with approximately 10 metres of disseminated mineralization. It is present in sections 97+75, 98+00, and 98+25. It is believed that the lens is cut-off from the 97+00 lens by a major fault and cut-off on the northern end by another fault.
- 100+00 lens - a massive and semi-massive sulphide lens with a surface expression projecting down-dip at approximately 45° to the north-east to a depth of 30 metres. The lens ranges from 2 to 4 metres in thickness and has considerably disseminated mineralization with low assay values above and below the sulphide zones. The lens is present in sections 99+25, 99+50, 99+75, 100+00 and 100+25.

4.4 Block Model Orientation and Size

A three dimensional block model was defined to cover all three lenses from surface to a maximum depth of 100 metres. The model was oriented with the rows parallel to the baseline and the top datum elevation at the highest point of topography is the area covered by the model.

The dimensions of the model were as follows:

Length:	450 metres
Width:	270 metres
Height:	165 metres
Datum:	1580 metres

and the corners located at the following local co-ordinate points:

CORNER	NORTHING	EASTING
Bottom left	9900.00	-10050.00
Bottom right	9900.00	- 9600.00
Top right	10170.00	- 9600.00
Top left	10170.00	-10050.00

TABLE 5 - BLOCK MODEL CORNER CO-ORDINATES

The size, thickness and orientation of the sulphide lenses necessitated the selection of a relatively small block size in order to model the geology in a representative way.

The final block size was selected with the following two points in mind:

- o the total number of columns and rows in the model must be kept at a manageable level

- o the shape and size of the blocks should effectively model the geology both in plan and in section

The property sloped downhill from the left of the model at an elevation of 1580 metres to the right of the model at an elevation of 1470 metres. Only a very small proportion of the blocks in the model were above the 1555 metre elevation therefore these blocks were allocated a height of 5 metres per level, for the first 5 levels. All other levels (6 through 50) were allocated a height of 3 metres.

In plan, the length of the block (parallel to baseline and along strike) was defined as five metres, and the width of the block as three metres.

REC	STAT	ROCK CODE	DESCRIPTION	RELATIVE DENSITY [tn/bcm]	PEN	SLOPE ANGLES [DEGREES] NW	N	NE	W	E	SW	S	SE
1	1	0	AIR	.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
2	1	1	MAFIC TUFFS - WEAKLY SERICITIC - HANGING WALL	3.240	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
3	1	2	MAFIC TUFFS - HIGHLY SERICITIC _ HANGING WALL	3.240	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
4	1	5	MAFIC TUFF AND ARGILLITE - FOOTWALL	3.220	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
5	1	6	ARGILLITES AND SILTITES - FOOTWALL	3.230	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
6	1	10	OVERBURDEN AND WEATHERED ROCK	2.210	2	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0
7	1	6660	MASSIVE SULPHIDES - ORE	4.980	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
8	1	6662	SEMI MASSIVE SULPHIDES - ORE	4.130	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
9	1	7770	MASSIVE BARITES - ORE	4.870	5	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
11	1	9999	DISSEMINATED MINERALIZATION - ORE	3.240	6	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0

TABLE 4 - MODEL ROCK-TYPE CODES

4. GEOLOGICAL INTERPRETATION AND MODELLING

4.1 Methodology

A computer modelling technique known as three dimensional block modelling was used to model the deposit. A three dimensional block model is a regular matrix of discrete blocks where each block represents a homogeneous volume of material with certain definable characteristics such as rock-type, density and grade. Part of the modelling process is the assignation of rock-types to each block in the model, and this is carried out in the following manner:

- o define block model rock-types and codes
- o perform a geological interpretation on drillhole cross-sections
- o determine a suitable block model location and size
- o transpose geology from cross-sections to level plans
- o construct a three dimensional rock-type model
- o construct a three dimensional density model

The following sections describe each of the five steps in more detail.

4.2 Definition of Block Model Rock-Types and Codes

A set of numeric codes representing unique rock types had previously been defined for the drillhole data. These were defined in some detail, with major rock types and alterations and were considered to be too detailed for the purposes of geological modelling. Consequently, a second set of rock-type codes was defined, solely for the construction of the geological model. These are defined in Table 4. Four mineralized rock types were defined, being massive sulphide (6660), semi-massive sulphide (6662), massive barytes (7770) and disseminated mineralization (9999). This latter category was defined as all material other than massive sulphides, semi-massive sulphides or massive barytes having gold assays greater than 0.01 oz/ton.

4.3 Interpretation of Drillhole Cross-Sections

A set of vertical cross-sections perpendicular to the base line was plotted. The sections were spaced 25 metres apart to coincide with the approximate spacing of the drilling pattern. A horizontal search distance of 12.5 metres was defined each side of the section line to ensure all drillholes were projected onto the section lines.

and lead, zinc and copper are presented in percent. Missing assay values are indicated by the numerals -99.

3.5 Conversion of Assay Value Units

As the Corporation Falconbridge assay units for gold and silver were in grams per tonne and the balance of the gold and silver assays were in troy ounces per short ton, it was necessary to apply a conversion factor to the Corporation Falconbridge assays to derive consistent assay units of Troy ounces per short ton. The conversion factor was calculated as follows:

Tonnes per short ton	=	0.90703
Troy ounces per gram	=	0.03215 multiply
Grams per tonne	=	0.02916 oz/short ton

3.6 Drillhole Survey Data

The holes drilled in all programs were a combination of vertical and angled holes. All non-vertical holes had down-hole surveys carried out using a combination of acid etch and/or tropari tests to determine hole dip angles and azimuths. These were all entered into the drillhole database after conversion from Magnetic North to local grid North.

3.7 Trenching Data

A number of trenches had been cut through overburden to expose the surface expression of the sulphide zones during early exploration. These trenches had been located roughly on plan and no detailed data was available as to the exact location of each sample in each trench.

Therefore only a limited amount of trench data could be used (one set of average values per trench) with the locations being scaled off the appropriate map. Average trench assay values were obtained from a Corporation Falconbridge sketch plan locating trenches and summarizing assays.

TRENCH	AU oz/ton	Ag oz/ton	Pb %	Zn %	Cu %
9675	0.20	2.18	2.65	2.11	0.39
9700	0.34	4.10	1.04	0.93	0.47
9725	0.39	29.50	1.41	2.57	1.16
9950	0.52	1.87	2.78	5.36	0.37
9975	0.66	1.85	3.35	0.46	0.09
10000	1.32	7.30	7.80	3.20	2.60

TABLE 3 - TRENCH ASSAY DATA

of each drillhole log, the development of a set of appropriate rock-type codes for major lithological units and altered rock-units and the subsequent assignation of these rock codes to each geological intersection in each hole.

As three separate drilling programs had been undertaken, with core logging carried out by separate individuals as well, rock-type descriptions for similar rock units in different holes showed significant variation. In this respect, ambiguity as to the exact rock type code to be used often arose, which was subsequently overcome by reference to the general geological descriptions and sequences outlined in the various reports provided. The majority of the ambiguities arose as a result of the variety of description used for the alterations that have taken place in both the hanging wall and the footwall host rocks.

As the major mineralization occurred in either massive sulphides, semi-massive sulphides, massive barytes or sulphide stockworks, all of which were well defined in the logs, any resultant ambiguity in the drillhole database was not considered to be of serious consequence to the reserves calculated.

A summary listing of the numeric rock-type codes defined is presented in Table 2. Detailed printouts of individual drillholes and drillhole lithology are presented in the attached Annexures to this report.

3.4 Drillhole Assay Data

Assay data was provided in two main formats. The first, for all drilling carried out by Corporation Falconbridge in 1983, consisted of assays for gold and silver in grams per tonne, Lead, Zinc and Copper in percent, some Arsenic and some Barium in percent. Assays were carried out on the majority of the core, and were presented on the drillhole log sheet with the top and bottom of each sample defined. The second format, for the more recent drilling program consisted of geochemical samples from core, used to indicate zones of mineralization, and detailed assays for mineralized zones, indicated by higher values in the geochemical samples. Assay values were present for Gold and Silver, both in Troy ounces per short ton, Lead, Zinc, Copper, Arsenic and Barium all in percent. The assays were presented on Assay certificates from Kamloops Research and Assay Laboratories (KRAL) which detailed both sample numbers and assay values. Sample numbers had been cross referenced to drillhole name and sample location by Minorex Consultants. Project terms of reference dictated the use of gold, silver, lead, zinc and copper assays. All available assays for these minerals were obtained from both the Corporation Falconbridge drillhole logs and the KRAL assay certificates and entered into the database.

Detailed printouts of the drillhole assay values for all drillholes are presented in the attached Annexures to this report. Note that all gold and silver assays are presented in ounces per short ton,

REC	STAT	ROCK CODE	DESCRIPTION	RELATIVE DENSITY (tn/bcm)	PEN	NW	N	NE	W	E	SW	S	SE
						SLOPE ANGLES (DEGREES)							
1	1	10	Casing	.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
2	1	11	Erratics and lost core	.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
3	1	22	Faults and fault zones	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
4	1	111	Schists	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
5	1	100	Quartz veins	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
6	1	200	Footwall basalts	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
7	1	210	Chloritic mud	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
9	1	220	Dykes	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
10	1	330	Generic cherts	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
11	1	331	Pyritic cherts	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
12	1	332	Sericitic cherts	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
13	1	333	Pyritic/sericitic cherts	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
14	1	334	Cherts and interbedded siltites	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
15	1	335	Cherty debris flow	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
16	1	4440	Generic tuffs	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
17	1	4441	Felsic tuffs	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
18	1	4442	Felsic Lapilli tuff	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
19	1	4443	Pyritized mafic lapilli tuffs	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
20	1	4444	Tuffaceous cherts	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
21	1	4445	Interbedded volcanics and argillites	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
22	1	4446	Siliceous tuff and quartz veins	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
23	1	4447	Sulphide rich tuffs	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
24	1	550	Generic breccias	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
25	1	551	Selicious breccia	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
27	1	552	Sulphide rich breccia	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
28	1	553	Mafic to intermediate volcanic breccias	2.000	3	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
29	1	6660	Generic sulphides	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
30	1	6661	Massive sulphides	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
31	1	6662	Semi-massive sulphides	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
32	1	6663	Stringer zones;stockworks and sulphide rich debris	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
33	1	6664	Sulphide rich fault zones and gouge	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
34	1	7770	Massive barytes	2.000	4	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
35	1	880	Generic siltites and sandstones	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
36	1	881	Finely bedded siltites and argillites	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
37	1	882	Epiclastic sandstones	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
38	1	883	Grits and lithic wackes	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
39	1	884	Sulphide rich siltites	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
41	1	885	Pyritic mudstones'	2.000	2	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
42	1	90	Generic argillites	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
43	1	91	Black argillites	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
44	1	92	Intermediate and interbedded argillites	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0
45	1	1	Country rock	2.000	1	45.0	45.0	45.0	45.0	45.0	45.0	45.0	45.0

TABLE 2 - DRILLHOLE ROCK-TYPE CODES

PRINTOUT OF BOREHOLE INFORMATION

SUMMARY PRINTOUT

RECORD	BOREHOLE	TYPE	NORTHING CO-ORD [m]	EASTING CO-ORD [m]	COLLAR ELEVATION [m]	LENGTH [m]	GEOLOGICAL INTERSECTS	ASSAY INTERVALS	SURVEY INTERVALS	DATE	STATUS
66	86-136	1986	10102.00	-9874.00	1523.40	94.50	7	0	3	21/ 2/1986	1
67	86-137	1986	10096.00	-9925.00	1512.30	68.10	9	2	2	21/ 2/1986	1
68	86-138	1986	10009.00	-10001.00	1497.20	22.80	3	0	1	21/ 2/1986	1
69	86-139	1986	10085.00	-10029.00	1478.20	76.20	4	2	1	21/ 2/1986	1

TABLE 1 - DIRLLHOLE SUMMARY PRINTOUT (continued)

PRINTOUT OF BOREHOLE INFORMATION

SUMMARY PRINTOUT

RECORD	BOREHOLE	TYPE	NORTHING CO-ORD [m]	EASTING CO-ORD [m]	COLLAR ELEVATION [m]	LENGTH [m]	GEOLOGICAL INTERSECTS	ASSAY INTERVALS	SURVEY INTERVALS	DATE	STATUS
33	85-102	1985	10021.00	-10001.00	1495.57	35.70	7	6	1	20/ 2/1986	1
34	85-103	1985	10024.00	-9975.00	1502.93	28.60	8	10	1	20/ 2/1986	1
35	85-104	1985	10024.00	-9975.00	1502.93	38.70	7	7	1	20/ 2/1986	1
36	85-105	1985	10024.00	-9950.00	1510.79	25.90	9	11	1	20/ 2/1986	1
37	85-106	1985	10023.00	-9924.50	1521.25	34.70	6	9	1	20/ 2/1986	1
38	85-107	1985	10034.00	-9900.00	1529.80	50.90	6	2	1	20/ 2/1986	1
39	85-108	1985	10032.00	-9900.00	1529.57	83.40	10	1	1	20/ 2/1986	1
40	85-109	1985	10027.00	-9799.00	1548.17	106.10	12	17	1	20/ 2/1986	1
41	85-110	1985	10027.00	-9799.00	1548.17	88.10	10	8	1	20/ 2/1986	1
42	85-111	1985	10032.00	-9773.00	1551.85	112.50	11	7	1	20/ 2/1986	1
43	85-112	1985	10032.00	-9773.00	1551.85	82.30	12	6	1	20/ 2/1986	1
44	85-113	1985	10033.00	-9737.00	1556.38	97.50	11	1	1	20/ 2/1986	1
45	85-114	1985	10038.00	-9823.00	1545.38	115.80	10	8	1	20/ 2/1986	1
46	85-115	1985	10038.00	-9823.00	1545.38	94.79	10	14	1	20/ 2/1986	1
47	85-116	1985	10048.50	-9800.00	1547.10	89.60	8	0	1	20/ 2/1986	1
48	86-117	1986	10048.00	-9800.00	1547.10	121.00	12	6	1	21/ 2/1986	1
49	86-118	1986	10049.00	-9773.00	1547.60	123.70	10	8	4	21/ 2/1986	1
50	86-120	1986	10056.00	-9745.00	1548.40	114.30	9	8	4	21/ 2/1986	1
51	86-121	1986	10026.00	-9749.00	1555.40	93.30	8	0	4	21/ 2/1986	1
52	86-122	1986	10050.00	-9850.00	1530.00	75.60	9	0	4	21/ 2/1986	1
53	86-123	1986	10001.00	-9826.00	1541.20	57.00	10	10	2	21/ 2/1986	1
54	86-124	1986	10000.00	-9773.00	1541.20	42.40	5	0	3	21/ 2/1986	1
55	86-125	1986	10000.00	-9773.00	1550.50	71.60	11	7	3	21/ 2/1986	1
56	86-126	1986	9998.00	-9851.00	1533.20	43.90	6	0	4	21/ 2/1986	1
57	86-127	1986	9987.00	-9720.00	1561.60	61.00	7	0	2	21/ 2/1986	1
58	86-128	1986	9991.00	-9697.00	1567.80	76.10	8	1	3	21/ 2/1986	1
59	86-129	1986	9977.00	-9671.00	1573.10	51.50	8	10	3	21/ 2/1986	1
60	86-130	1986	9966.00	-9650.00	1577.60	57.60	8	1	2	21/ 2/1986	1
61	86-131	1986	9999.00	-9595.00	1588.50	94.20	12	1	3	21/ 2/1986	1
62	86-132	1986	10000.00	-9594.00	1588.30	129.20	8	0	3	21/ 2/1986	1
63	86-133	1986	10051.00	-9673.00	1571.20	118.90	8	0	2	21/ 2/1986	1
64	86-134	1986	10050.00	-9673.00	1571.20	131.10	10	0	4	21/ 2/1986	1
65	86-135	1986	10058.00	-9640.00	1576.00	130.90	6	0	4	21/ 2/1986	1

TABLE 1 - DRILLHOLE SUMMARY PRINTOUT (continued)

PRINTOUT OF BOREHOLE INFORMATION

SUMMARY PRINTOUT

RECORD	BOREHOLE	TYPE	NORTHING CO-ORD [m]	EASTING CO-ORD [m]	COLLAR ELEVATION [m]	LENGTH [m]	GEOLOGICAL INTERSECTS	ASSAY INTERVALS	SURVEY INTERVALS	DATE	STATUS
1	RG-1	1983	10095.00	-9975.00	1492.61	157.30	28	60	5	20/ 2/1986	1
2	RG-2	1983	10050.00	-10000.00	1494.33	92.00	14	24	1	20/ 2/1986	1
3	RG-3	1983	10085.00	-10025.00	1478.14	110.60	19	36	3	20/ 2/1986	1
4	RG-4	1983	10063.50	-9950.00	1511.51	110.00	20	43	1	20/ 2/1986	1
5	RG-5	1983	10136.40	-10000.00	1470.36	139.00	24	80	5	20/ 2/1986	1
6	RG-6	1983	10133.60	-9950.00	1489.94	129.50	19	72	4	20/ 2/1986	1
7	RG-7	1983	10050.00	-9925.00	1524.94	81.70	15	44	1	20/ 2/1986	1
8	RG-8	1983	10000.00	-9700.00	1567.64	98.50	20	54	3	20/ 2/1986	1
9	RG-9	1983	10050.00	-9900.00	1530.45	82.90	15	38	1	20/ 2/1986	1
10	RG-10	1983	10020.00	-9800.00	1548.71	93.00	12	52	3	20/ 2/1986	1
11	RG-11	1983	10165.00	-9900.00	1501.53	164.40	25	108	5	20/ 2/1986	1
12	RG-12	1983	10110.00	-9850.00	1526.96	146.90	13	67	4	20/ 2/1986	1
13	RG-13	1983	10100.00	-10100.00	1447.44	93.30	9	24	3	20/ 2/1986	1
14	RG-14	1983	10141.00	-10200.00	1404.84	128.30	16	32	1	20/ 2/1986	1
15	RG-15	1983	10142.00	-10300.00	1386.28	98.50	11	39	3	20/ 2/1986	1
16	RG-16	1983	9957.00	-9725.00	1561.31	46.60	8	14	1	20/ 2/1986	1
17	RG-17	1983	9976.50	-9750.00	1555.65	60.10	11	21	1	20/ 2/1986	1
18	RG-18	1983	9990.00	-9725.00	1561.23	90.85	20	24	1	20/ 2/1986	1
19	RG-19	1983	10002.00	-9700.00	1567.51	125.90	16	19	3	20/ 2/1986	1
20	RG-20	1983	9981.00	-9675.00	1572.95	102.70	15	27	3	20/ 2/1986	1
21	RG-21	1983	9953.00	-9675.00	1572.21	79.90	13	30	3	20/ 2/1986	1
22	RG-22	1983	9974.50	-9650.00	1576.70	93.60	13	29	3	20/ 2/1986	1
23	RG-23	1983	10001.00	-9650.00	1578.89	121.30	13	24	3	20/ 2/1986	1
24	RG-24	1983	9967.00	-9625.00	1584.54	62.80	10	28	2	20/ 2/1986	1
25	RG-25	1983	9954.00	-9700.00	1567.53	45.10	8	24	1	20/ 2/1986	1
26	RG-26	1983	10170.00	-10000.00	1464.63	198.10	18	24	5	20/ 2/1986	1
27	RG-27	1983	10167.00	-9900.00	1501.52	225.00	19	38	7	20/ 2/1986	1
28	RG-28	1983	9957.00	-9760.00	1557.04	30.50	3	0	1	20/ 2/1986	1
29	RG-29	1983	10181.00	-9700.00	1547.00	60.00	5	0	1	20/ 2/1986	1
30	RG-56	1983	10070.00	-9775.00	1550.00	131.10	4	3	1	20/ 2/1986	1
31	RG-57	1983	10075.00	-9725.00	1555.00	152.40	4	0	1	20/ 2/1986	1
32	85-101	1985	10048.00	-10023.00	1485.64	68.30	9	5	1	20/ 2/1986	1

TABLE 1 - DRILLHOLE SUMMARY PRINTOUT

3.0 DRILLHOLE DATABASE

3.1 Database Creation

The available drillhole geological logs supplied and the assay data supplied were sorted by year of drilling. A database to hold the drillhole and assay data was created using the PC-MINE system in the following manner:

- o entry of drillhole header data
- o assignation of geological rock-type codes to lithology and entry of drillhole lithology data
- o entry of drillhole assay data
- o conversion of assay value units
- o entry of drillhole survey data
- o entry of trench data

These activities are explained in more detail in the sections following.

3.2 Drillhole Header Data

The drillhole header data consisted of the following information per drillhole:

- drillhole name
- year of drilling
- collar co-ordinates
- length of hole
- no. of geological intersections
- no. of assay intervals
- no. of survey intervals

The information for this header data was obtained from the drillhole log sheets. It should be noted that the drillhole co-ordinates used were obtained from these logs and are therefore field recorded. A complete survey plan showing surveyed positions of all drillholes was unavailable at the time of data entry and therefore could not be used.

A summary listing of header data for all 70 drillholes entered into the database is presented in Table 1. A drillhole location map is presented as Figure 2.

3.3 Assignation of Geological Rock-Type Codes and Entry of Drillhole Lithology

The PC-MINE system requires a numeric rock type code to be assigned to each unique lithological unit. This necessitated detailed study

- probable reserves at 0.100 oz/ton gold cut-off

197 920 short tons at	0.231 oz/ton Au
	2.481 oz/ton Ag
	2.478 % Pb
	2.608 % Zn
	0.610 % Cu

Series of pits were generated for both the 97+00 lens and the 100+00 lens. The 98+00 lens was not considered due to its tonnage and depth. Preliminary pit tonnages indicate stripping ratios between 14 tons waste per ton of ore and 17 tons waste per ton of ore for the 97+00 lens depending on cut-off grade; and 27 tons waste per ton of ore and 50 tons waste per ton of ore for the 100+00 pit. Both pits were designed to extract the maximum possible tons of ore at each cut-off grade by mining the total lens tonnages in each pit.

2.0 SUMMARY OF INFORMATION USED

2.1 Computer Database and Orebody Modelling

The PC-MINE computer software system, designed for orebody modelling and mine planning, was used for this ore reserve evaluation.

A total of 70 drillholes were loaded into the database, each hole having lithological data and assays for gold, silver, lead, zinc and copper. Assays were all converted to ounces per ton gold, ounces per ton silver, percent lead, percent zinc and percent copper.

A three dimensional block model was used to represent the deposit, with separate models constructed for geology, density and each of the five minerals. Model dimensions were 90 columns each 5 metres wide, 90 rows each 3 metres wide, and 50 levels of 3 to 5 metres in height. The model was located and oriented in such a position to cover the deposit including the 97+00 lens, 98+00 lens and 100+00 lens. The co-ordinate system used co-incided with the field co-ordinate system set up during exploration.

An initial statistical and geostatistical analysis indicated that geostatistical orebody modelling would not be appropriate, therefore the grade models were interpolated using the inverse distance squared method.

Probable reserves were calculated for three separate gold cut-off grades of 0.05, 0.075 and 0.100 ounces per ton respectively. They can be summarized as:

o probable reserves at 0.05 oz/ton gold cut-off

267 720 short tons at	0.190 oz/ton Au
	2.145 oz/ton Ag
	2.149 % Pb
	2.247 % Zn
	0.527 % Cu

o probable reserves at 0.075 oz/ton gold cut-off

229 220 short tons at	0.211 oz/ton Au
	2.365 oz/ton Ag
	2.352 % Pb
	2.450 % Zn
	0.586 % Cu

1.0 INTRODUCTION AND TERMS OF REFERENCE

1.1 Introduction

This report was prepared by Steffen Robertson and Kirsten (B.C.) Inc. at the request of Sentinel Management Corporation of Vancouver. The report details the computerized evaluation of a massive sulphide deposit with gold, silver, lead, zinc and copper mineralization under option to the Rea Gold Corporation of Vancouver.

The property is situated in the Adams Plateau region of the Kamloops Mining District in the Province of British Columbia.

The REA (AR/HN) property consists of seven located mineral claims located immediately southwest of Johnson Lake, approximately 60 kilometres north-northeast of Kamloops, B.C. The geographic coordinates of the property are Latitude 51°09' North, Longitude 119°49' West.

The registered owner of the property is Corporation Falconbridge Copper, subject to the terms of an option agreement signed by Rea Gold Corporation in November 1983. Within the claim group there is a nine hectare area called the Rea Gold concession, which covers the L97+00, L98+00 and L100+00 lenses of gold bearing, massive sulphide mineralization. This concession area is wholly owned by the Rea Gold Corporation; the amendment is dated November 1, 1985.

Exploration and local geological investigations were carried out by both Corporation Falconbridge Copper and Minorex Consultants Ltd. of Kamloops, B.C. All drilling results, geological descriptions, assays and basic data were provided to Steffen Robertson and Kirsten directly by the Rea Gold Corporation and Minorex Consultants.

1.2 Terms of Reference

Project terms of reference were defined in a proposal from Steffen Robertson and Kirsten (B.C.) Inc. to Sentinel Management Corporation dated January 6th, 1986. In summary, the proposal outlined the following scope of work:

- use the PC-MINE system to perform the computer evaluation by three dimensional block modelling

- enter 45 drillholes from the 1983 and 1985 drilling programs into the database

- enter an additional 25 drillholes for the 1986 drilling programs into the database when available

- plot drillhole cross-sections to interpret orebody structure and size

LIST OF FIGURES

22. PROBABLE RESERVES FOR 98+00 LENS; CUT-OFF GRADE = 0.075 OZ/TON GOLD

23. PROBABLE RESERVES FOR 100+00 LENS; CUT-OFF GRADE = 0.100 OZ/TON GOLD

24. PROBABLE RESERVES BELOW 100+00 LENS

25. 97+00 PRELIMINARY PIT RESULTS

26. 100+00 PRELIMINARY PIT RESULTS

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