Reserves and Mine Evaluation Study

ADANAC DEPOSIT

Prepared by:
AMAX Operations Research Dept.

November, 1975

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INTRODUCTION

The following study presents the results of two independent evaluations of the Adanac deposit, with respect to:

A). Conversion of M. David's Kriged Ore Reserve Model into MEDS 3-Dimensional format.

B). Development of the Adanac deposit into 3-dimensional format directly from the DDH, RDH and bulk sample information supplied by Climax Molybdenum Corporation. This involved the generation of assay interval averages for each sample interval with respect to DDH and RDH information (a similar averaging technique as done by M. David in his data preparation for the development of his interpretation of the Adanac deposit), bench composite computations using DDH, RDH and bulk sample data, and the grade interpolation of the deposit.

C). Geologic Reserves at specified grade cutoff are given for each mineral model.

D). Pit design with specified mine/mill grade cutoff and economic parameters using a whole block approach (discussion in summary) for M. David's Mineral Model and for the MEDS Mineral Model.

E). Mine Reserves of each incremental pit for both mineral models at specified cutoff grades, and a special summary for total mine reserves at specified cutoff grades for the ultimate pit of each mine model.

Statistical distributions are presented for the MEDS assay data by laboratory with average grade computed by MEDS as the comparison base. The composite data is presented statistically with the MEDS prepared data as the comparison base against similar composites developed from data unique to each laboratory. Appropriate comparisons and statistical presentation of the two sampling methods (DDH and RD versus bulk sampling) are made by laboratory for the Assay Data and Composite Data.

A special comparison (Appendix 14 - computer output) has been made of the assay intervals for DDH, RDH and bulk samples comparing the average grade developed by MEDS (involving the simple arithmetic average of assay values in the same interval when more than one assay value appears for one or more laboratories) and the SELECT grade.

A statistical summary is presented for the MEDS 3-dimensional Mineral Model and M. David's 3-dimensional Mineral Model.

A separate mineral model was developed from the same assay data as given M. David in order to reasonably answer why the 3-dimensional block model prepared by the Kriging technique created such a rectangular model, why mine grade data appears at depth and pinches out with such abruptness, and to essentially validate the mineral model prepared by the Kriging technique.

The MEDS System referred to in the introduction and throughout the report is the Mine Evaluation and Design System used by AMAX to evaluate surface ore bodies as to their mineral content and economic value through the sequential design of incremental open pits to the ultimate pit. The MEDS System is a computerized systematic procedure, and reports that appear in the various appendices are essentially the computer output generated in the various stages and steps of the evaluation study.

CONCLUSIONS

The results of the Adanac deposit grade interpolation based on the Kriging method, and that of the MEDS procedure used are quite similar from a grade and tonnage point of view. Based on Geologic Reserves, M. David's Mineral Model yields, at a .08% MoS_2 cutoff, 148,480,000 tons with an average grade of .112% MoS_2, while the MEDS Mineral Model yields, at .08% MoS_2 cutoff, 139,200,000 tons with an average grade of .118% MoS_2.

At a .12% cutoff the MEDS Mineral Model contains 47,232,000 tons at an average grade of .159% MoS_2. M. David's Mineral Model contains 46,016,000 tons at an average grade of .147% MoS_2 at the .12% MoS_2 cutoff. The two different techniques show a few percentage points variation in their interpretation of the ore body. It is noteworthy that 90-100 million tons (dependent on the interpretation) lay in the range of .08 to .12% MoS_2. There is a shift in the overall grade distribution comparing the MEDS Geological Reserves with the Kriged Model Geological Reserves above a .10% MoS_2 cutoff, however the shift is less than 7% and due to the apparent smoothing of grade variations attendant in the Kriging Technique.

Comparisons of the Ultimate Pit designed within each Mineral Model yields the same overall ore tonnage. M. David's Mineral Model and the MEDS Pit Design algorithm give

38,633,000 tons with an average grade of .140% MoS_2 and a stripping ratio of .899 at a .10 cutoff. The MEDS developed Mineral Model yields an ultimate pit with 38,533,000 tons with a .150% MoS_2 average grade and a 1.034 stripping ratio at a .10 cutoff. The slight grade differential is coincident to that observed in Geologic Reserves Comparison. The higher stripping ratio is the cause of a slight shift in the ultimate pit comparing both models, and because of the preponderance of material that is clustered about the cutoff .10 in the ultimate pit.

Because of the sensitivity of the overall minable tonnage that is demonstrated by an upward shift in grade interpretation (see Table 3 Pit ADB04), a look at the assay samples (DDH and Bulk samples) should determine whether some adjustment of the DDH grade assays are warranted or unwarranted.

Since the only relevance in grade comparison is to that of a proposed mining plan, the logical comparison interval should be based on the proposed bench height or 40 feet. As the mining plan also considers material ore or waste, only composites (40 feet in verticle dimension) above the economic grade cutoff are considered. Table 6 shows the comparison of 6 raises with coincident drill hole sampling. The average grade of raise composites is .202% MoS_2, while the average grade of coincident drill hole composites is .212% MoS_2. The composite variation of grade is so small as to indicate no reason to consider any type of adjustment.

Considering the quantitative comparison made for coincident DDH and Bulk Sampling using standard statistical comparisons to determine sample population limits, there appears to be no significant difference between Diamond Drilling and Bulk Sampling. From an examination of the various methods of sampling, there is insufficient information to use the bulk sampling results to adjust the drill hole assay intervals. Certainly from the limited areas and locations analyzed it is not proper to apply any adjustment of grade intervals on a deposit wide basis.

Since a very high percentage of blocks reside on economic grade cutoff, it is apparent that a slight shift in grade interpretation for the overall block model results in a wide shift in tonnage. This in itself would present a significant problem in the physical mining and grade control problems. No financial evaluations were attempted as the studies made on both models indicate a sub-marginal deposit.

COMMENTS

The basis of comparison of M. David's Mineral Reserve Model with a Mineral Model developed within the MEDS System guidelines and technique lay in the initial obscure reasons for the Kriged models' rectangular pattern and sharp grade cutoff at depth. It was difficult if not impossible to determine precisely the reasons without evaluating all the Sample Assays (DDH and RDH and Bulk samples) statistically and through composite evaluation with surrounding blocks on a bench by bench basis.

Primarily the reasons for the rectangular pattern lay in:

A). The Adera fault is basically vertical in nature and cuts the ore zone at 10N (950N in the MEDS Model). As composites to the north contain only trace grade of MoS_2, we cut the interpolation distance to the north by limiting the interpolation to 950N (M. David did the same). Although it was possible to have limited interpolation by the insertion of a fault plane, the absence of grade to the north eliminated the need for an additional complex procedure.

B). The drilling pattern and resultant composites, limit grade interpolation in both the Kriged ore model and in the MEDS Model.

C). With the acquisition of sketch geological interpretation section displays, it is apparent why the Kriged ore model contains sharp grade pinches at depth in the south-western portion of the deposit. The drilling stops on occasion in the ore zones and yields essentially ore blocks on the edge of the model. Since there is a practical extrapolation limit within both the MEDS procedure and M. David's Kriging method, the answer of insufficient data for further interpolation is reasonable.

The deposit appears as essentially a banded one, with occasional sharp plunges of ore zones intermingled with gently sloping lenses. Our decision was to eliminate a 3-dimensional search approach since there appears no justification for searching the vertical dimension for composites in the interpolation procedure. A 2-dimensional search technique was agreed upon and a series of search distance options, extrapolation distance and power weighting were experimented with. These test results can be found in Appendix 6 (computer output).

Essentially these tests comprise a specified interpolation area on specific benches (39, 40 and 41) using M620V1, bench displays of those areas on benches 39, 40 and 41 using M606V2, and statistics of the sample areas using M608V1. The displays include the interpolated grade from MEDS as well as the interpretation by M. David of the same area.

The results of the test applications were the selection of a search distance in the X and Y direction of 1000 feet, utilizing a Z dimension of 10 feet or same bench limitation for composites influence. Third order inverse weighting was selected rather than second order, as to minimize smoothing effects that result from inverse square of the distance weighting. An eliptical pattern of search was utilized with maximum extrapolation distance of 200' considered reasonable. An additional technique of utilizing an expanding area of search based on 100 foot increments to a maximum of 1000 feet (or the edges of the deposit) until composites were located was tried for block grade assignments. This technique proved inadequate.

Appendix 7 (computer output) contains a bench display of composites for use in overlaying interpolated bench plans.

Appendices 8 and 9 contain M. David's Ore Model in Plan from levels 24 to 58 in the former while the latter contains the MEDS interpolated Mine Model in Plan for the same benches. It is noteworthy to mention that in comparison with the two techniques, Kriging tends to smooth out grade variations more than the MEDS technique used in the interpolation.

Care was taken to average data for assay intervals when more than one assay evaluation appeared for any interval to eliminate any select bias. It is apparent that there is a great deal of low grade material in the west portion of the deposit, but based on current costs and economics, the ore material cannot support the extensive stripping required for systematic mining in the increasing elevations in the west edge of the mineral model.

Discussions on the drill holes and bulk samples appear later in this study. All output data relating to the MEDS Mineral Model development including the Project Control File, Mine Model Parameters, Assay and Survey listings, Composite Calculations and 3-D Mine Model and Topographic Matrix preparation appear in Appendix 10 (computer output). A thorough discussion of the development of the computerized MEDS Mine Model appears in a later section of this report.

It should also be noted that we considered it more appropriate to utilize a rock density of 12.5 cubic ft. per ton or a total 100 x 100 x 40 foot block weight of 32,000 tons for each mineral model. All tonnage calculations for both mineral models utilize this tonnage parameter.

SUMMARY OF RESULTS

I. Pit Design and Mine Reserves with M. David's Mineral Reserve Model

Total geologic reserves of M. David's Kriged Mineral Model are given in Table 1. Each incremental pit was prepared by initially setting a minimum cutoff grade for mill feed and a minimum grade for the base block in the center of the cone used to expand the pit to the surface. In order to determine the ultimate pit, the minimum grade was successively lowered to the lowest economic cutoff grade that would produce a profit when run through the mill, while the higher grade in the sequence would support waste stripping.

The base information used to design each incremental pit was:

A).	Average mining cost per ton		$ .50
B).	Processing cost per ton of ore comprising:		
	(1). Depreciation	$ .50	
	(2). Administrative and General	.75	
	(3). Milling and Concentrating	1.25	
C).	Value per pound of molybdenum		$ 2.62
D).	Expected milling recovery		90%
E).	Weight percentage of Mo per lb of MoS_2		60%
F).	Effective value per lb of MoS_2 contained per ton		$ 1.42
G).	Pit slope angle		45°

In Table 2, six incremental pits are given, each with the incremental ore tonnage at the specified cutoff grades, the respective average grade and stripping ratio. The cutoff grade of .10 is assumed to be the lowest possible economic grade that can be put through the mill (actual figure is .106 MoS_2). Cumulative reserves of 38,633,000 tons of ore with a .140 average MoS_2 grade and a .899 stripping ratio are contained in the ultimate .10 economic pit.

The last case in Table 2 is an ultimate pit with $3.00 processing cost per ton and $.50 per ton mining cost. The effect is to raise the economic cutoff to .13 and lower the mine reserves by 50% in comparison to the same cutoff with $2.50 per ton processing cost.

Table 3 contains three sensitivity studies using a .10 cutoff and a $2.50 processing cost. The first case is an ultimate pit using a 15% fixed adjustment factor on the block model grade values. The effect indicates the sensitivity of the ore body in terms of tonnage to interpretation or grade assignment differences.

The second case in Table 3 uses M. David's mineral model to determine the mine reserves at .10 MoS_2, $2.50 processing cost per ton of ore, and a $.50 per ton mining cost, a 25% price increase. The third case uses M. David's mineral model to determine mine reserves at .10 MoS_2, \$2.50 processing cost per ton of ore, a \$.50 per ton mining cost, and a 50% price increase.

TABLE 1 - GEOLOGIC RESERVES

M. David's Adanac Deposit Mineral Model - Block = 32,000 tons*

CUTOFF % MoS_2	BLOCKS	TONS (000's)	AVERAGE GRADE % MoS_2
.02	13,657	437,024	.070
.04	10,685	341,920	.082
.06	7,279	232,928	.097
.08	4,640	148,480	.112
.10	2,587	85,984	.129
.14	696	22,272	.167
.16	318	10,176	.189
.18	162	5,184	.211
.20	76	2,432	.236

* Tonnage factor 12.5 cubic feet/ton

TABLE 2 - MINE RESERVES (in Thousands)

INCREMENTAL PIT	CUTOFF GRADE	CUTOFF MoS_2 .10	.11	.12	.13	.14
ADA01	.15	7,167	6,333	5,933	5,467	4,733
Grade		.169	.178	.182	.187	.195
S.R.		1.074	1.347	1.506	1.720	2.141
ADA02	.15	4,800	4,533	4,300	3,833	3,300
Grade		.155	.158	.161	.165	.170
S.R.		.368	.449	.527	.713	.990
ADA03	.14	2,700	2,533	2,300	2,067	1,433
Grade		.142	.144	.147	.150	.156
S.R.		.051	.632	.797	1.000	1.884
ADA04	.13	3,300	3,000	2,700	2,300	1,167
Grade		.135	.138	.141	.144	.154
S.R.		.505	.656	.840	1.159	3.257
ADA05	.12	5,900	5,033	4,267	2,567	1,533
Grade		.131	.135	.139	.149	.159
S.R.		.819	1.132	1.516	3.182	6.000
ADA06	.11	2,233	2,033	1,133	633	433
Grade		.128	.130	.141	.154	.164
S.R.		.851	1.033	2.647	5.526	8.538
ADA07	.10	12,533	9,100	5,633	3,900	2,467
Grade		.126	.134	.146	.155	.168
S.R.		1.231	2.073	3.964	6.171	10.338
Cumulative Reserves						
Ultimate		38,633	32,567	26,267	20,767	15,067
Pit	Grade	.140	.147	.155	.163	.173
ADA07	S.R.	.899	1.253	1.793	2.533	3.869
Ultimate	.13	14,833	13,733	12,867	11,600	9,500
Pit		.159	.163	.167	.171	.179
ADX01		.652	.784	.904	1.112	1.579

TABLE 3 - MINE RESERVES (in Thousands)

ULTIMATE PIT	CUTOFF GRADE	CUTOFF MoS_2 .10	.11	.12	.13	.14
ADB04	.10	52,933	47,133	40,900	33,633	26,767
Grade		.150	.156	.162	.170	.179
S.R.		.773	.992	1.295	1.791	2.507
ADB01	.10	41,333	34,600	27,600	21,600	15,600
Grade		.139	.146	.154	.162	.173
S.R.		.995	1.383	1.988	2.818	4.286
ADB02	.10	47,467	38,133	29,500	22,633	16,200
Grade		.136	.144	.153	.162	.173
S.R.		1.254	1.805	2.626	3.726	5.603

II. Pit Design and Mine Reserves with MEDS Ore Reserve Model

Total Geologic Reserves of the MEDS derived Ore Reserve Model is given in Table 4. The preparation of the incremental pits to the computed ultimate pit proceeded exactly as those for M. David's model. The interpolated ore body was evaluated using the same economic constraints of \$.50 mining cost/ton, processing cost of \$2.50 ton of ore, \$1.42 value per lb of MoS_2 contained per ton, and a pit slope of 45^{o} for the pit design.

The results are tabulated in Table 5, and compare on an incremental pit basis with those given in Table 3.

The output for each of the pit designs, incremental pit mine reserves at the specified cutoff grades, and the cumulative reserves appear in Appendix 5 (computer output) as output from M720V1 (Economic Pit Limits Calculations) and M723V1 (Mine Reserves by bench and Cumulative).

TABLE 4 - GEOLOGIC RESERVES

MEDS Adanac Deposit Mineral Model - Block = 32,000 tons*

CUTOFF % MoS_2	BLOCKS	TONS (000's)	AVERAGE GRADE % MoS_2
.02	12,998	415,936	.072
.04	10,377	332,064	.083
.06	6,956	222,592	.099
.08	4,350	139,200	.118
.10	2,549	81,568	.138
.12	1,476	47,232	.159
.14	898	28,736	.178
.16	509	16,288	.201
.18	315	10,080	.220
.20	180	5,760	.244

* Tonnage factor 12.5 cubic feet/ton

TABLE 5 - MINE RESERVES (in Thousands)

INCREMENTAL PIT	CUTOFF GRADE	CUTOFF MoS_2 .10	.11	.12	.13	.14
MED01	.15	11,967	11,067	10,167	9,100	8,367
Grade		.173	.179	.185	.192	.197
S.R.		1.401	1.596	1.826	2.158	2.434
MED02	.15	5,333	4,933	4,800	4,200	3,500
Grade		.159	.164	.165	.171	.178
S.R.		.425	.541	.583	.810	1.171
MED03	.14	3,967	3,633	3,067	2,567	2,100
Grade		.144	.145	.151	.156	.161
S.R.		.546	.688	1.000	1.390	1.921
MED04	.13	3,533	3,133	2,467	1,867	1,000
Grade		.141	.146	.154	.164	.189
S.R.		.774	1.000	1.541	2.357	5.267
MED05	.12	3,000	2,533	2,133	1,333	967
Grade		.140	.147	.153	.170	.184
S.R.		1.022	1.395	1.844	3.550	5.276
MED06	.11	2,933	2,600	1,533	1,067	700
Grade		.133	.136	.151	.163	.178
S.R.		.784	1.013	2.413	3.906	6.476
MED07	.10	7,800	5,367	3,800	2,300	1,767
Grade		.125	.135	.143	.156	.162
S.R.		1.350	2.416	3.825	6.971	9.377
Ultimate		38,533	33,267	27,967	22,433	18,400
Pit	Grade	.150	.157	.165	.175	.184
Cumulative	S.R.	1.034	1.356	1.802	2.493	3.259
Reserves						

STATISTICAL ANALYSIS OF ASSAY VALUES

Due to the methods of sampling and the use of different laboratories for analyzing the samples, two questions of a statistical nature are presented.

A). How should the assay results from different laboratories for the same interval be treated?

B). Are the bulk sample and drill hole assays in agreement, and if not, should some adjustment be made of the drill hole assays based upon the bulk sampling.

The data available to answer these questions is the MEDS Assay File and the programs used for M401 - Statistical Analysis of Assay Data and M403 - Regional Analysis of Assay Data. The results are summarized and discussed in the following sections and the computer output is in Appendix 11 (computer output).

COMPARISON OF DIAMOND DRILLING AND BULK SAMPLING

There are at least two major problems in the comparison of diamond drilling and bulk sampling in addition to the most apparent problem of trying to compare two dissimilar quantities:

(1) the distribution and continuity of the metal content of the samples; and

(2) finding an appropriate comparison which has relevance to a physical mining plan.

Distribution and Continuity of Samples

The following table summarizes drilling and bulk sampling data at the mine grid location (OE, ON).

DDH093	(MEDS #1067)	Raise 0-0	(MEDS #3119)
Interval	% MoS_2	Footage	% MoS_2
40 to 50	.019	10	.123
50 to 60	.010	·8	.164
60 to 100	.009	8	.066
70 to 80	.012	8	.211
80 to 90	.007	10	.115
90 to 100	.033	7	.071
100 to 110	.118	7	.050
110 to 120	.084	7	.040
120 to 130	.086	7	.113
130 to 140	.076	6	.027
140 to 150	.110	11	.068
150 to 160	.118	8	.071
		10	.015
Average =	.057	Average =	.088

Why such a low grade raise? What is % diff all raises.

Considering only the average values, it would seem that there is a significant difference between them. But what is significant? Determining the spread or dispersion of data about the average value (mean) is the recognized method which provides a measure of significance.

The spread of data about the mean is measured by the variance of the samples which may be regarded as the averaged squared deviation of the samples from the mean. Squared deviations are used instead of the simple average of deviations because the simple average will always be zero.

Discounting the unequal sample lengths in the Raise 0-0 as being of negligible difference (the unweighted average of values ($\bar{X}$) is 0.087), the variance sx^2 is calculated as:

$$sx^2 = \frac{n\,\Sigma x_i^2 - (\Sigma x_i)^2}{n\,(n-1)} = \frac{(13)(.136656) - (1.134)^2}{13\,(13-1)}$$

$$sx^2 = 0.003145$$

The standard deviation is the square root of the variance and this statistic describes the dispersion of samples around the mean in the units of measurement of the samples.

$$sx = \sqrt{0.003145} = .0561$$

At the 95% confidence level, the interval around the mean is expected to be

$$CI = \pm \left(\frac{sx}{\sqrt{n}} \right) \quad ({}^{t}0.05)$$

$$CI = \pm \left(\frac{.0561}{\sqrt{13}} \right) \quad (2.179) = \pm \ .034$$

Note: The value 2.179 used above is the "Student t" statistic for 12 degrees of freedom with 2.5% in each extreme (critical region) of the + - distribution, (i.e., this is a two-tailed test).

Therefore in 19 times out of 20 we would be correct in saying that two average grades <u>are</u> <u>not</u> significantly different if they lie within the range (X-CI) to (X+CI), 0.087 $\pm$ .034 or from 0.053 to 0.121 % MoS_2.

Since the value of the diamond drilling samples is 0.057 % MoS_2, we <u>cannot</u> say that it is significantly different than the bulk sample.

To carry this logic to its conclusion, there is another diamond drillhole within 15 feet of the coincident drilling and bulk sampling which should be examined. This is DDH001 (MEDS #1068), and it shows an average value of 0.102% MoS_2. This average value also lies within the 95% confidence limits calculated above. It is interesting to note that the average of the two diamond drillholes DDH093 and DDH001 over like intervals is 0.080% MoS_2 which agrees quite closely with the unweighted average of 0.087% MoS_2 from the Raise. I see

The continuity of the metal content in the relatively small drillholes does appear to be poor which makes comparison between drilling and bulk sampling difficult. However, using the preceding quantitative basis to determine significance, THERE DOES NOT SEEM TO BE A SIGNIFICANT DIFFERENCE BETWEEN DIAMOND DRILLING AND BULK SAMPLING. Indeed using the preceding logic there is only 1 chance in 20 that there is a significant difference considering only the single drillhole coincident with the raise and ignoring the drillhole only 15 feet away.

COMPARISON OF DATA RELEVANT TO MINING

agreed

Comparing assay intervals is not relevant to the proposed mining scheme, since vertical intervals of 7 to 10 feet would not be mineable on a 40 foot bench. Therefore, it would seem that intervals of 40 feet should be compared to be relevant to mining. Also, since there is a grade cutoff below which material is uneconomic, the comparison should be made only on values above the cutoff.

In the raise at (OE,ON) there is a single 40 foot composite above the cutoff with an average of 0.140% MoS_2. The two drillholes 1067 and 1068 have single 40 foot composites of 0.175 and 0.174% MoS_2, respectively. Table 6 summarizes coincident drilling and bulk sampling for 40 foot composites above 0.10% MoS_2.

It is our conclusion that there is no significant difference between the diamond drilling and bulk sampling. Since it is apparent that the data resides in the same population and coincident with the previous analysis, there is no justification for any statistical upgrading of the sample assays.

SEL?

TABLE 6 -

RAISE	COMPOSITES	AVERAGE	DRILLHOLE	COMPOSITES	AVERAGE
OE,ON	1	0.140	1067 1068	1 1	0.175 0.174
Actual DDH composites		.094 .116			
OE,1N	2	0.150	1065 1066	2 2	0.150 0.151
OE,2N	3	0.169	1064	3	0.172
OE,4N	4	0.268	1063	4	0.293
2W,2N	2	0.175	1055	2	0.223
2E,2N	2	0.230	1081	2	0.188
Average	14	~~0.202~~ 0.209	Average	14	~~0.213~~ 0.155

JACK.

Is this the only DDH v Raise relation where DDH is higher?

JACKS FIGURES.

Raise factor 15% high?

STATISTICAL ANALYSIS OF ASSAYS FROM DIFFERENT LABORATORIES

Program M401V2 was used to compute the frequency distribution of a selected assay and compute the average grade of the selected assay for a range of cutoff grades. The program also computes the average of the assays available from different laboratories associated with the selected assays above each cutoff grade. The option in M401V2 to weight the assays by interval length was not used.

Table 7 summarizes the results for all assays from all laboratories. Since each sample was not analyzed by each laboratory, the mean MoS_2 % and standard deviation of assays from each laboratory would be directly comparable only if the samples of each laboratory were representative of the whole deposit.

Table 8 contains the rotary and diamond drillhole assay data summarized by laboratory and Table 9 contains the bulk sample assay data. The rotary drillholes are included with the diamond drillholes even though the sample size is larger, since there are only three rotary drillholes with 104 assays.

The average assay value (AVRG) was computed by averaging the values for the laboratories together.

Jack I were samples only send to one Assay office how many x checks on assays if any

No significant differences between the assays from the various laboratories is indicated by these tables. A further comparison was made between the three major laboratories using various cutoff grades and these results in Table 10 also indicate no significant differences between laboratories.

JACK –

Are there any direct comparisons of assays

ie same sample different labs.

LORING

TABLE 7 - ALL ASSAY DATA SUMMARIZED BY LABORATORY*

LABORATORY	NUMBER OF ASSAYS	MEAN MoS_2 %	STANDARD DEVIATION
AVRG	6008	.085	.103
CE	3337	.076	.098
LO70	2817	.070	.083
ADAC	1691	.094	.106
LOR	475	.091	.097
ML	361	.051	.076
WH	322	.123	.153
OTHR	369	.070	.079
SELC	6007	.082	.104

is this on hole

* See Page 39 for an explanation of Laboratory Codes

Why Adams consistently high? except to WH who can't assay MoS_2 anyway

TABLE 8 - ROTARY AND DIAMOND DRILLHOLE

ASSAY DATA SUMMARIZED BY LABORATORY *

LABORATORY	NUMBER OF ASSAYS	MEAN MoS_2 %	STANDARD DEVIATION
AVRG	5501	.078	.094
CE	3337	.076	.098
LOR70	2817	.070	.083
ADAC	1585	.089	.104
LOR	369	.070	.085
ML	361	.051	.076
WH	322	.123	.153
OTHR	369	.070	.079
SELC	5500	.075	.096

is this a lab?

* The rotary assay data has been combined with the diamond drillhole data even though the samples are of different sizes, since there are only 104 rotary assay intervals.

TABLE 9 - BULK SAMPLE ASSAY DATA SUMMARIZED BY LABORATORY

LABORATORY	NUMBER OF ASSAYS	MEAN MoS_2 %	STANDARD DEVIATION
AVRG	507	.160	.149
ADAC	106	.173	.105
LOR	106	.165	.101
SELC	507	.160	.149

TABLE 10 - COAST ELDRIDGE ASSAYS VS. LORING 1970 ASSAYS VS. ADANAC ASSAYS

CUTOFF GRADE % MoS_2	CE		LO70		ADAC	
	No.	% MoS_2	No.	% MoS_2	No.	% MoS_2
0.00	3337	.076	2817	.070	1691	.094
.08	1073	.161	819	.159	646	.186
.10	793	.187	604	.184	535	.206
.12	578	.217	433	.214	431	.229
.14	419	.251	311	.248	347	.254
.16	326	.280	256	.269	288	.275
.18	248	.316	197	.299	237	.298
.20	197	.349	152	.332	197	.320
.30	76	.521	66	.449	83	.432

PREPARATION OF M. DAVID'S MINERAL RESERVE MODEL FOR PIT DESIGN

The mineral model of the Adanac deposit was received by Climax Molybdenum in card format in the form:

(I,J,K,T,IV,IR,IP)

where

I is the index of the section along the X axis where I=1 is section 27W and I=41 is section 13E

J is the index of the section along the Y axis where J=1 is section 13S and J=23 is section 9N

K is the index of the level along the Z axis where K=15 is elevation 3860 and K=50 is elevation 5260

T is the Kriged estimate of the average molybdenite grade in a 100 x 100 x 40 foot block in % MoS_2

IV is the relative standard error on T in %

IR is the estimated height of the block in feet and defines the bottom of the overburden for the upper blocks

IP is the absolute standard error on IR in feet

It was necessary to reorganize the storage arrangement of the mineral model for subsequent pit design efforts and the following chronicled steps taken.

(1) To prove the completeness of the data and insure there was no data lost, the program which listed the Kriged model in section was converted to the AMAX Engineering System and

a complete listing and tape dump were made. These were heavily spot checked to proof the model.

(2) The 3-dimensional area of the model was expanded to accommodate sensitivity pit design evaluations given by testing highest MoS_2 price considerations and lower cost mining.

The 3-dimensional block model was initialized to the same block dimensions (100 x 100 x 40) as M. David's block size. The model is expanded to 50 blocks along the X-axis (column), 40 blocks along the Y-axis (row) and 58 levels on the Z-axis. It was necessary to convert the system to cartesian coordinates and to allow for a 50 foot shift of the original block model to store the west and north edges of the mineral model on the coincident coordinates. This was to reflect M. David's centering the blocks on section. The coordinates of the new system are:

- 3270 at the west edge of the model
  1750 at the east edge of the model
- 2050 at the south edge of the model
  1950 at the north edge of the model
  The maximum elevation is 6180
  The minimum elevation is 3860

It is quite simple to compare location of blocks since 27W corresponds to -2700, 13E corresponds to 1300, 13S corresponds to -1300 and 9N corresponds to 900.

The only data utilized by the MEDS System were the 3-dimensional block location parameters, the Kriged estimate of the block and the estimated height of the block in feet defining the bottom of the overburden for the upper blocks.

NOTE: If more than 50% of the block was in the overburden area or the block grade was less than .08. The block was set to zero (0.0) grade for pit design purposes.

(3) In order to allow for the usage of the MEDS procedure, with minimal or redundant input, a PROJECT CONTROL FILE (PCF), is required to be initialized. The PCF essentially contains all dimensions of the system, a log of all files used, and their size, mine model descriptors, assay and survey descriptors and locations.

(4) Concurrently, a 3-dimensional block model was created that would allow for the definition of:

MoS_2 Average block grade,
TOPO Topography matrix
OVRB Overburden

This was accomplished by usage of M601V1 (Generation of 3-D Block Model).

A routine M610V1 which allows for manually coded data to be entered into the grade matrix was prepared in such a fashion that a prepared revised data base of M. David's original ore model could be loaded to the grade matrix.

(5) In order to proof the resulting grade matrix, M606V1 was utilized to display the ore body in plan level by level. The bench plans were checked against the original section displays to insure the model was loaded correctly.

(6) A topographic matrix, the dimensions of the X-Y axis of the 3-D model, was coded manually from the Topographic Map supplied by Climax Molybdenum Co. The two dimensional matrix was then loaded to a file by means of M630V1.

At this point a 3-D block model existed along with a 2-D topo matrix. Since the objective was to determine total matrix. Since the objective was to determine total geologic reserves (as a proof check with M. David's findings) and total mine reserves in an ultimate pit design (discussed in the summary), the files were converted to the DIPPER System format.

(7) The block values were then compressed into a condensed mine model so that pit design could be done rapidly.

A DIPPER PCF (Project Control File) was concurrently created and performs the same function as the Project PCF. Essentially the condensed form of the grade model contains only one grade per block and only whole blocks are allowed. The 2-Dimensional topographic surface was condensed in the same fashion.

For a specified set of cutoff grades for mine and mill, as well as economic constraints, a pit limit is determined and stored for future use. The rationale in storing each pit surface is to constrain the sequential pit evaluation to the new pit bottom and the layer between the two surface skins. The DIPPER Mine Reserves (M723V1) provides for mine reserves between the pit limits and the succeeding pit design or for a total reserve tonnage between the ultimate pit and the surface topography. The condensed mineral model of M. David's was evaluated by a series of incremental pits (M720V1), Reserves Tabulated for each incremental pit (M723AD) and Symbol and Scale maps in Section and Plan (M722V0) were run, as well as a Surface Bench Display for each incremental pit (M721AD).

SUMMARY

The original ore model was initially entered into a separate 3-D block matrix. With the development of a 3-D block matrix prepared by the MEDS System, the two models were stored in the same 3-D block matrix for easy comparison. All run decks for the M. David model evaluation exist in Appendix 1 (run decks). Results of the Pit Designs appear in the Summary (Part I).

The output for each of the pit designs and the incremental pit mine reserves as well as cumulative reserves appear in Appendix 2 (computer output) as output from M720V1 (Economic Pit Limit Calculations), and M723V1 (Mine Reserves - by bench and cumulative). A special series of output runs appear in Appendix 3 (computer output) and represent W-E Section, N-S Sections and level displays from the mineral model on a block/bench basis. There are outputs from M722V0 (symbol plots). These displays indicate block grade value by multiplying the listed number by two (2) to yield the block grade in hundredths. Sight checking the sections and plans indicate the rationale behind the pit design to the skillful user.

Also included in Appendix 3 (computer output) are the bench outlines (in sketch format) of each incremental pit included in Appendix 2. These are the output of M721V1 (Mine Design Surface Display).

In Appendix 4 (computer output) are the original sections prepared on the AMAX Engineering Computer System. These are prepared to insure data integrity and exact similarity with M. David's original model. Also included in Appendix 4 is a complete sequential listing of M. David's block model and the data conversion prepared to put the model into the MEDS 3-Dimensional block model (M610AD).

DEVELOPMENT OF THE MEDS MODEL

The development of a mineral model by the MEDS procedure involved a great deal more effort and preparation than for a normal deposit. Because of the number of laboratories involved in the processing of assay samples involving DDH, RDH, Raises, Drifts and Cross-Cuts it was desirable to carry all assays done by each lab in the assay file for statistical comparisons on assay and composite data.

The Adanac Assay Data used in this study is based upon card input data supplied to us, and is the same data as that given Michel David for his study. This data was converted to MEDS standard format and loaded in the MEDS files.

The samples were processed by several different laboratories and are divided into the groups below on each interval.

	Code
Whitehorse Laboratory	WH
Coast Eldridge	CE
Metallurgical Laboratory	ML
Loring Laboratories 1969	LOR
Loring Laboratories 1970	LO70
Mine Laboratory	ADAC

Other Laobratories	OTHR
Selected Assay	SELC
Average Assay (MEDS)	AVRG

The average assay (AVRG) is the arithmetic average of the data from all laboratories for each assay interval.

Many of the samples have been assayed several times. In cases where the same interval was assayed more than once by the same lab the assays were averaged. For each interval there is a selected assay which is deemed to be the most reliable by Chapman, Wood and Griswold Ltd. (SELC).

The assay data is divided into five classes of data using the codes:

Diamond Drillhole	1
Rotary Drillhole	2
Bulk samples - Raises	3
Bulk samples - Drifts	4
Bulk samples - Cross-cuts	5

The coordinates for each set of assay data is based upon a north-south grid system which is not coincident with the mine grid used for the mine model. The rotation of the N-S coordinates to the mine grid was accomplished using the equations:

$$X_{mine} = -13671 + X_{ns}\ .8949 + Y_{ns}\ .4462$$

$$Y_{mine} = -4932 + Y_{ns}\ .8949 - X_{ns}\ .4462$$

These equations were based upon a rotation angle of 26.5° and a coincident point measured from the topography maps. The coincident point is:

North-South		Mine Grid
X	10000	-260. Easting
Y	10000	-445. Northing

In order to contain M. David's model and the MEDS prepared mineral model the PCF mine model descriptors were altered. The PCF listing (M101V1 and M102V2) are contained in Appendix 2 (computer output).

The Adanac assay data was loaded into the system using M201V1. The output is contained in Appendix 10 (computer output). A special version was prepared for listing the loaded assay data which compared MEDS average computed assay intervals with those under the select values. Where the select value was higher than AVRG a "##" appears next to that value. The special run is in Appendix 14 (computer output).

Table 11 contains a listing of the DDH, RDH and Bulk Samples as they are identified in the MEDS System and under ADANAC Identification.

A special version of USR208 was prepared to average all the data for each sample interval where more than one assay existed and to insert into the model as a type code the type of sample (DDH, RDH, Bulk) each assay came from.

Groups of statistics on the assay data were prepared using M401AD and are discussed separately. (See Statistical Analysis). The results of the 401 statistical comparisons are to be found in Appendix 11 (computer output).

In preparation for interpolation, the assay data for the vertical and horizontal samples were composited using M501V1. The vertical assay data was composited into 40 foot vertical composites to correspond to bench height of 40 feet. The horizontal assays for drifts, and cross-cuts were composited into 100 foot composites corresponding to the horizontal block size of 100 feet by 100 feet. The results of compositing are in Appendix 12 (computer output). A comparison of composites and assay interval data is given in Table 12. It is interesting to note that the average assay grade MoS_2 for all data above .10 cutoff is .207, and the composites above .10 cutoff average .169 MoS_2 for all data. The results of compositing show the expected dilution of material on each mining bench.

Table 13 is a comparison of composite versus assay interval data for all drillholes. At the .10 cutoff the average grades are .198 MoS_2 for the drillholes and .164 for the composites.

The composites were then sorted for interpolation with M506V1. Bench display maps of the intersecting composites were developed using M504V1 and are in Appendix 7 (computer output).

The 3-dimensional block file had already been loaded with M. David's Mineral Model, eliminating the necessity of regenerating the 3-dimensional block model. M620V1 was run to interpolate the MEDS interpretation of the Adanac deposit using 1000 foot X Y search distance and limiting the vertical search to the same bench. The maximum extrapolation distance was set at 200 feet with an inverse weighting of the third power based on distance.

Statistics were run on the block model with M608V1 and are included in Appendix 11 (computer output). Bench maps were produced with M606V2 and exist in Appendix 9 (computer output).

The same topographic surface file used for pit design with M. David's model was used for the MEDS pit design topographic surface. The results are discussed in the Summary, Part II.

It was decided to prepare an Ultimate Pit design with the Partial Block Subsystem and plot the results. However prior to the development of the 3-D model for this design, it was necessary to enter an overburden matrix into the 3-dimensional block file.

Interpolation of grade into the blocks of the MEDS Mine Model is done independently of the topography and overburden surfaces so it is necessary to add a code to each block indicating whether the block is 50% or more above the rock surface (code=1) or 50% or less below the overburden surface (code=2). The steps required to add this code to each block were:

1. The rock/overburden surface is defined by the first interval in each drillhole. Program M209V1 was modified to compute the coordinates and overburden thickness at each intersection of a drillhole and the rock/overburden surface.
2. The coordinates and overburden thicknesses were used as input to Program M631V1 to interpolate the overburden thickness for column of blocks in the mine model and stored in the topography file.
3. The overburden code was inserted into each block in the mine model using Program M633V1.

4. A user subroutine Program M612V1 is used to set the grade values in the overburden blocks (block with a code of one is set to 0.0).

In order to use the MEDS partial block system of pit design, Program M721V1 was used to plot a printer map of the ultimate pit MED07. Using this plot, an ultimate pit was drawn by smoothing the pit walls and coded for input to Program M701V1 which compute the pit limits by bench and stores the limits in the MEDS Pit Outlines File. After setting up the ore reserve descriptor with Program M710V1, the reserves were computed using Program M711V1 and summarized by Program M712V1. Appendix 13 (computer output) contains these results.

The purpose of doing the partial block pit design was twofold - first to illustrate its use and secondly compute the overburden tonnage in the ultimate pit MED07. The overburden tonnage in the partial block version of Pit MED07 is 6.55 million tons.

TABLE 11

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION	
1	2005	3	RDH
2	2006	6	RDH
3	2007	11	RDH
4	2008	13	RDH
5	1010	71	DDH
6	1011	72	DDH
7	1012	73	DDH
8	1013	74	DDH
9	1014	57	DDH
10	1015	56	DDH
11	1016	55	DDH
12	1017	49	DDH
13	1018	58	DDH
14	1019	54	DDH
15	1020	53	DDH
16	1021	48	DDH
17	1022	60	DDH
18	1023	59	DDH
19	1024	42	DDH
20	1025	35	DDH

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION	
21	1026	22	DDH
22	1027	52	DDH
23	1028	26	DDH
24	1029	70	DDH
25	1030	20	DDH
26	1031	75	DDH
27	1032	76	DDH
28	1033	62	DDH
29	1034	41	DDH
30	1035	34	DDH
31	1036	21	DDH
32	1037	77	DDH
33	1038	69	DDH
34	1039	78	DDH
35	1040	79	DDH
36	1041	68	DDH
37	1042	80	DDH
38	1043	81	DDH
39	1044	82	DDH

SURVEY RECORD	MEDS IDENTIFICATION	ADÁNAC IDENTIFICATION	
40	1045	51	DDH
41	1046	40	DDH
42	1047	33	DDH
43	1048	83	DDH
44	1049	67	DDH
45	1050	84	DDH
46	1051	66	DDH
47	1052	85	DDH
48	1053	86	DDH
49	1054	87	DDH
50	1055	61	DDH
51	1056	88	DDH
52	1057	19	DDH
53	1058	17	DDH
54	1060	89	DDH
55	1061	50	DDH
56	1062	90	DDH
57	1063	65	DDH
58	1064	91	DDH

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION
59	1065	92 DDH
60	1066	09 DDH
61	1067	93 DDH
62	1068	1 DDH
63	1069	2 DDH
64	1070	13 DDH
65	1071	16 DDH
66	1072	94 DDH
67	1073	95 DDH
68	1074	96 DDH
69	1075	8 DDH
70	1076	3 DDH
71	1077	4 DDH
72	1078	18 DDH
73	1079	97 DDH
74	1080	32 DDH
75	1081	24 DDH
76	1082	6 DDH
77	1083	5 DDH

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION	
78	1084	14	DDH
79	1085	7	DDH
80	1086	47	DDH
81	1087	46	DDH
82	1088	31	DDH
83	1089	64	DDH
84	1090	98	DDH
85	1091	99	DDH
86	1092	30	DDH
87	1093	63	DDH
88	1094	45	DDH
89	1095	38	DDH
90	1096	29	DDH
91	1097	23	DDH
92	1098	44	DDH
93	1099	43	DDH
94	1100	100	DDH
95	1101	37	DDH
96	1102	28	DDH
97	1103	12	DDH

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION
98	1104	27 DDH
99	1105	39 DDH
100	1106	36 DDH
101	1107	11 DDH
102	1108	15 DDH
103	1109	25 DDH
104	1110	10 DDH
105	1111	JM-2 DDH
106	1112	JM-1 DDH
107	3114	1 RAISE
108	3115	2 RAISE
109	3116	3 RAISE
110	3117	4 RAISE
111	3118	5 RAISE
112	3119	6 RAISE
113	3120	7 RAISE
114	3121	8 RAISE
115	4122	W 1/1 DRIFT

SURVEY RECORD	MEDS IDENTIFICATION	ADANAC IDENTIFICATION	
116	4123	W 1/2	DRIFT
117	4124	W 1/3	DRIFT
118	5125	XS1/1	CROSS-CUT
119	5126	XS1/2	CROSS-CUT
120	5127	XS1/3	CROSS-CUT
121	5128	XS1/4	CROSS-CUT
122	5129	XN1/1	CROSS-CUT
123	5130	XN1/2	CROSS-CUT

TABLE 12 - COMPARISON OF COMPOSITE AND ASSAY INTERVAL DATA FOR ALL DATA

CUTOFF GRADE MoS_2	ASSAY INTERVALS No.	ASSAY INTERVALS % ABOVE CUTOFF	ASSAY INTERVALS % MoS_2	COMPOSITES No.	COMPOSITES % ABOVE CUTOFF	COMPOSITES % MoS_2
.00	6008	100.0	.085	1476	100	.078
.08	2058	34.3	.179	532	36.0	.149
.10	1573	26.2	.207	398	27.0	.169
.12	1255	20.9	.231	297	20.1	.190
.14	988	16.4	.259	222	15.0	.210
.16	812	13.5	.283	160	10.8	.234
.18	656	10.9	.310	124	8.4	.252
.20	548	9.1	.334	83	5.6	.283
.30	246	4.1	.453	21	1.4	.411