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GRUM DEPOSIT

(ROUGH DRAFT)

e2ic

GRUM DEPOSIT INDEX

e2ig

GRUM DEPOSIT

LIST OF ILLUSTRATIONS

e2ik

GRUM DEPOSIT

I REVIEW OF EXPLORATION AND DEVELOPMENT 1973-1976.

The Grum deposit was discovered in late 1973 by Aaro Aho of the AEX Syndicate while drilling a broad area of high Turam response and a coincident gravity anomaly in the general area of the already known Champ Deposit.

e2in

AEX Syndicate drilled 16 holes totalling 4000 meters to July 1974 when Kerr Addison assumed control of the project and drilled a further 13,000 meters of surface diamond drill holes spaced 60 meters apart on section lines 120 meters apart. This work indicated a deposit 1,680 meters long and 366 meters wide between sections 62W and 88W with a preliminary estimate of 25,400,000 short ton~~nes~~s averaging 4.01% lead, 6.67% zinc and 2.5 oz/SDT silver.

e2j3

Upon completion of this initial drilling phase, an extensive underground development program was initiated. The deposit was explored from underground by means of an 800 meter long decline on a 16% grade. Two ramps, 120 meters apart, were driven along the plunge of the mineralization with connecting cross cuts at 120 meter intervals. Ring drilling in the plane of the geologic sections was conducted from the ramps at 60 meter intervals (see Plan of Underground Workings). The program included 2,900 meters of openings and 15,000 meters of diamond drilling.

e2jb

At the same time as the underground development, a further 24,000 meters of surface diamond drilling was carried out to complete the grid at 60 meter spacing on lines 60 meters apart. from section 60W to 88W. Fill-in holes at 30 meter spacing were also drilled on sections 64W and 68W (see DDH Plan Map - Figure I).

e2jf

II GEOLOGY

a) Regional

The Grum deposit occupies a similar position within the regional stratigraphy of the Anvil Range as the Vangorda Deposit. The section is generally dominated by calcareous chlorite and chlorite-sericite phyllites in the hanging wall and non-calcareous biotite-sericite phyllites and biotite--garnet-staurolite schists in the footwall. It appears that the amphibolite facies footwall sequence is separated from the hanging wall green schist facies by a low angle thrust
* fault ~~(at the base of the sulfide horizon)~~. The sulfide horizon is closely associated with graphitic phyllite.

e2jl

b) Structure

The sulfide horizon(s) and enclosing rock units have been subjected to at least four periods of plastic deformation. The main structure controlling the distribution of rock type is a north-westerly plunging series of more or less
* concentric "S" shaped recumbent folds (D2). This folding was accompanied by a pervasive axial plane foliation (S2) which in turn has been kink folded and also gently warped into broad synforms and antiforms.

The deposit has been further complicated by northeast. ~~N.E.~~ striking high angle faults which cut all structures except the footwall thrust fault. (Figure II).

e2k2

c) Sulfide Horizon Stratigraphy

The normal top to bottom sulfide sequence is as follows:

i) Massive pyritic sulfides, variably base metal bearing variably baritic.

ii) Pyritic quartzites, banded and/or foliated, generally base metal deficient.

There are, however, variations on this sequence caused by overturned folds and lack of massive sulfides (Figures III and IV).

e2k6

III CONSTRUCTION OF GEOLOGIC SECTIONS AND PLANS.

The procedure used by Kerr Addison in the preparation of geological sections and plans was as follows:

i) The surface drill holes were plotted in their deviated positions in plan view. These traces were then projected orthographically to cross and long sections at 1:500 scale.

ii) Underground holes were plotted on the appropriate cross sections.

iii) Primary and seconday foliation angles were plotted on drill holes in cross and long section.

iv) Geology of all underground workings were plotted on appropriate sections.

v) The cross sections 62W to 84W were interpreted for graphitic phyllite, chloritic phyllite, all other phyllites, massive sulfide and quartz sulfide rock units. Since the deposit is both stratiform and stratabound, it was possible to treat mineralized zones as rock types in constructing sections.

vi) By numbering axial planes, major fold hinges and faults, it was possible to correlate structures from cross section to cross section along longitudinal sections.

vii) Geologic plans were constructed from the sections at 9 meter intervals corresponding to the mine levels.

e2kj

IV DEVELOPMENT OF THE KERR ADDISON COMPUTER BASED TONNAGE AND GRADE MODEL.

The following procedure was used by Kerr Addison in the construction of a computer based tonnage and grade model (due to time constraints, Cyprus Anvil did not create an independent model for this deposit):

i) Development of a Drill Hole Data Base:

The Drill Log File (DLF) contains drill hole co-ordinates, down hole survey data, and lead, zinc and silver assays. The assaying was restricted to the sulfide horizon lithologies with all other intervals indicated as N/A. "Not Assayed" intersections include many thin (less than 3 meters) bands of internal phyllite waste (see discussion on Grade Adjustment Factors).

e211

ii) Development of the Actual Value File (AVF)

- A three dimensional block grid system was developed for the deposit. The unit block size is:

  DZ = bench height = 9.0 meters
  DX = width on easting = 15.0 meters
  DY = width on northing = 15.0 meters

- Actual value blocks are those which are intersected by drill hole(s). The drill hole samples within each block are weighted by sample length within the block.

- "Not Assayed" intersections were originally included in the actual value calculations with zero grade assumed. This was later changed to ignore these intersections assuming that the geology could be used as an ore control guide during mining.

- A minimum sample length of 0.4 meters was used. Blocks containing less than the minimum sample length of sulfide mineralization were ignore in the AVF calculation.

iii) The geology of the Grum Deposit was represented by three "rock" types in the computer model: massive sulfides (including baritic), quartz sulfides, and overburden. The complex geometry of the deposit required a 5m x 5m x 9m block size to define the geology. Geologic bench plans were used to enter the geology codes.

e2lg

iv) Development of the Mineral Inventory File (MIF) for 5x5x9 Meter Blocks.

* - The 15 meter AVF was used to interpolate grades into a 15 meter MIF. (Blocks in the 15 meter MIF contain from one to nine 5 meter sulfide blocks)

- Volumes of influence about each block were described using parallelepipeds.

- The AFV data within the volume of influence was weighted inversely by the distance cubed to interpolate the grade for the block.

- All 5 meter sulfide blocks with the 15 meter block were assigned the same grade.

e2ll

V MODIFICATIONS TO THE KERR ADDISON COMPUTER BASED MODEL.

The 5 x 5 x 9 meter Kerr Addison MIF was converted into a 15 x 15 x 9 meter MINTEC model for the purposes of efficient mine planning. All blocks in the model have the following information stored:

1. % Lead
2. % Zinc
3. Decagrams/metric tonne silver.
4. Number of 5 meter massive sulfide blocks.
5. Number of 5 meter quartz sulfide blocks.
6. % Lead + % Zinc.
7. % of block below topography.
8. Metric tonnes.

e2ml

a) ORE GRADES

Lead, zinc and silver values come directly from the Kerr Addison 5 meter MIF. The average value of all 5 meter sulfide blocks within the 15 meter block was used.

b) ROCK CODES

The number of 5 meter quartz sulfide (#QS) and massive sulfide (#MS) blocks within each 15 meter block was tabulated. These values were used in the tonnage calculation.

c) TOPOGRAPHY

The topographic surface for the Grum was digitized from the 1:5000 scale Kerr Addison base map.

d) TONNAGE CALCULATIONS

From selected drill core and hand specimens it was determined that massive sulfides had an average specific gravity of 4.10 and quartz sulfides had an average of 3.18. To provide a small factor of safety, the tonnage for each 15 meter sulfide block was calculated as:

(#MS x 4.0 + #QS x 3.0) x 225

e2mc

VI GEOLOGIC RESERVES

The following geologic reserves have been calculated for the Grum Deposit between sections 62W and 86W:

e2me

CUT-OFF GRADE (Pb+Zn)	4.0	4.5	5.0
Metric Tonnes (000's)	27650	25233	22945
% Pb	3.12	3.25	3.38
% Zn	4.87	5.11	5.34
gms/MT Ag	47.9	50.0	52.0

e2mi

VII MINE PLANNING

a) ECONOMIC "ULTIMATE" PIT

/ The following prices, costs, metallurgical data, and geotechnical criteria were assumed in the Grum economic geotechnical criteria were assumed in the Grum economic pit design:

i) Concentrate values (F.O.B. loadout)

Lead concentrate = $452/tonne
Zinc concentrate = $225/tonne
Silver in lead conc. = $152/tonne Pb conc.

ii) Metal recoveries

Lead = 80%
Zinc = 83%
Silver = 70%

iii) Concentrate metal grades

Lead Conc. = 60% lead with 685 grams/tonne silver.
Zinc Conc. = 55% zinc.

* iv) Lead equivalent grade formula

.043

Equivalent Pb% = %Pb + .85 x %Zn + 0.43 x gms/MT Ag.

v) Processing Cost

Milling Cost = $5.71/tonne
Power Cost = $.75/tonne
Mining Cost = $2.37/cubic meter
Transportation = $1.00/tonne ore.

vi) Cut-off grade = 4.5% Pb + Zn

(Note: not an "economic" cut-off grade)

vii) Pit Slopes

DIRECTION	SLOPE ANGLE
N.E.	42°
S.W.	45°
N.W.	43.5°
S.E.	43.5°

e2n2

Using these criteria, a nine phase ultimate pit (Figure V) was designed using the moving cone technique (see explanation in Vangorda Section).

Undiluted mining reserves by phase are as follows:

i) cut-off grade = 4% Pb + Zn
(note: volumes are measured within the same pit outline as for a 4.5% cut-off)

e2n6

PHASE	ORE TONNES (000's)	WASTE (000's M^3)	S.R.*	% Pb	% Zn	g/MT Ag
1	1711	6812	13.6	4.16	6.79	62.2
2	1016	2676	8.6	3.12	5.29	48.5
3	2088	5230	8.0	3.11	5.11	47.5
4	1662	5623	10.9	3.21	5.22	48.4
5	1855	4556	8.1	3.49	5.79	53.8
* 6	~~2136~~ 1784	5488	10.5	3.42	5.31	52.0
7	2136	5414	8.0	2.94	4.76	45.3
8 (a)	1112	3866	11.2	2.81	4.59	44.7
8 (b)	1112	3866	11.2	2.81	4.59	44.7
9	1107	2550	7.8	3.06	4.97	49.3
TOTAL	15583	46081	9.7	3.25	5.29	50.0

* note: S.R. = stripping ratio (M^3 waste ÷ M^3 ore)

e2ni

ii) cut-off grade = 4.5% Pb + Zn

PHASE	ORE TONNES ~~(000's)~~ ~~(000's)~~ (000's)	WASTE ~~(000's M^3)~~ ~~(000's M^3)~~ (000's M^3)	S.R.*	% Pb	% Zn	g/MT Ag
1	1660	6827	14.0	4.24	6.91	63.4
2	953	2695	9.2	3.23	5.46	50.2
3	1955	5272	8.6	3.22	5.28	49.2
4	1559	5655	11.7	3.31	5.39	50.0
5	1793	4575	8.4	3.55	5.90	54.8
6	1694	5514	11.1	3.50	5.47	53.2
* 7	1991	5460	~~12.4~~ 8.7	3.02	4.93	46.8
8 (a)	1009	3898	12.4	2.93	4.80	46.6
8 (b)	1009	3898	12.4	2.93	4.80	46.6
9	1056	2565	8.2	3.13	5.09	50.5
TOTAL	14679	46359	10.3	3.34	5.46	51.5

* note: S.R. = stripping ratio (M^3 waste ÷ M^3 ore)

e2o2

iii) cut-off grade = 5.0% Pb + Zn

PHASE	ORE TONNES (000's)	WASTE (000's M^3)	S.R.*	% Pb	% Zn	g/MT Ag
1	1581	6850	14.8	4.36	7.11	65.2
2	841	2729	10.6	3.43	5.78	53.1
3	1820	5314	9.3	3.34	5.44	50.7
4	1430	5695	12.8	3.46	5.61	52.0
5	1719	4597	8.9	3.62	6.03	55.9
6	1590	5545	11.9	3.59	5.66	54.9
7	1790	5523	9.8	3.15	5.17	48.8
8 (a)	911	3929	13.8	3.05	5.01	48.6
8 (b)	911	3928	13.8	3.05	5.01	48.6
* 9	1039	~~1570~~ 2570	8.3	3.15	5.13	50.8
TOTAL	13632	46680	11.2	3.46	5.66	53.3

* note: S.R. = stripping ratio (M^3 waste ÷ M^3 ore)

e2oe

b) ANNUAL PRODUCTION SCHEDULES

Ore delivery schedules and annual feed grades were calculated from the phase inventories (at a 4.5% cut-off) using the following parameters and guidelines:

- Ore production rate of 4660 tonnes/day
- Minimum of three months developed ore exposed at all times during schedule (see large graph - Figure VI).
- 12,000,000 cubic meters of pre-production stripping distributed as follows:

e2oi

PHASE	PRE-PROD. M^3 (000's)
1	5,000
2	1,000
3	1,000
4	1,000
5	1,000
6	1,000
7	1,000
8	1,000
9	0
Total	12,000

- Ore grades were adjusted downward by 6% to allow for mining dilution at geological ore/waste contacts (see discussion of Grade Adjustment Factors).

e2pi

cut-off grade = 4.0% Pb + Zn
dilution adjusted grades

YEAR	1	2	3	4	5	6	7	8	9	10
Waste (000's M^3)	4010	4010	4010	4010	4010	4010	4010	4010	2001	0
Ore (000's MT)	1701	1701	1701	1701	1701	1701	1701	1701	1701	274
% Pb	3.91	2.93	2.94	3.06	3.27	3.19	2.76	2.67	2.75	2.88
% Zn	6.38	4.92	4.82	5.01	5.40	4.96	4.47	4.34	4.49	4.67
Ag (g/MT)	58.5	45.3	44.8	46.5	50.4	48.5	42.6	42.1	44.1	~~4.63~~ 46.3 *

Total Mining Reserve: 15,583,000 metric tonnes
3.05% Pb
4.97% Zn
47.0 g/MT Ag

e2om

~~cut-off grade = 4.0% Pb + Zn~~

ii) cut-off grade = 4.5% Pb + Zn
dilution adjusted grades

YEAR	1	2	3	4	5	6	7	8	9
Waste (000's M^3)	4300	4300	4300	4300	4300	4300	4300	4300	0
Ore (000's MT)	1701	1701	1701	1701	1701	1701	1701	1701	1071
% Pb	3.97	3.04	3.05	3.21	3.32	3.13	2.82	2.75	2.94
% Zn	6.46	5.06	4.99	5.25	5.40	4.96	4.62	4.51	4.78
Ag (g/MT) grams	59.3	46.7	46.4	48.8	50.9	47.9	43.9	43.8	47.4

Total: 14,679,000 metric tonnes
Mining Reserve: 3.14% Pb
5.13% Zn
48.4 g/MT Ag

e2p6

iii) cut-off grade = 5.0%
dilution adjusted grades

YEAR	1	2	3	4	5	6	7	8	9
Waste (000's M^3)	4730	4730	4730	4730	4730	4730	4730	1570	0
Ore (000's MT)	1701	1701	1701	1701	1701	1701	1701	1701	24
% Pb	4.03	3.18	3.20	3.36	3.38	3.07	2.90	2.92	2.96
% Zn	6.60	5.25	5.20	5.54	5.44	4.99	4.76	4.78	4.82
* Ag (g/MT) grams.	60.4	48.6	48.3	51.3	51.9	47.5	45.8	46.9	47.8

Total: 13,632,000 metric tonnes
Mining Reserve: 3.25% Pb
5.32% Zn
50.1 g/MT Ag

e2q4

c) FEED GRADE ADJUSTMENT FACTORS

As in the case of the Vangorda deposit, negative adjustment factors have been applied to the Grum annual feed grade estimates. The Grum situation, however, is more complex with the geometry of the ore horizon(s) resulting in multiple geologic ore/waste contacts per bench. Dilution will also occur from the thin (less than 3 meters) bands of "not assayed" internal geological waste that was ignored in the AVF calculations.

The magnitude of the dilution adjustment was determined from the following analysis:

i) A MIF "geologic" inventory, assuming "not assayed" intersections at zero grade, had a combined lead/zinc average of 7.0%.

ii) The MIF "geologic" inventory, ignoring "not assayed" intersections had a combined average of 8.0% (note: this is the true geologic inventory).

iii) The "mineable" inventory should lie between the above two extremes.

iv) A grade adjustment of -6% applied to the true "geologic" ~~inventory results in a "mineable" combined average of~~
inventory results in a "mineable" combined average of 7.5% (note: this "mineable" figure refers to the total geologic reserves and not to a particular pit design).

This dilution could be reduced somewhat with 6 meter bench heights and with improved methods of ore control and sorting.

e2qi

VIII PLANT FOR FURTHER WORK

VIII PLANS FOR FURTHER WORK

a) Tonnage and Grade Model

Before production can commence on the Grum Deposit, much work remains to be done to verify and expand the current geological model.

- A re-logging program will be required to expand the section interpretation to include baritic, pyrrhotitic, and graphitic ore types.

- A re-sampling program will be required to provide additional assays such as copper, gold, and mercury.

- Specific gravity determination on core samples will be required to calculate weighted assay composites and accurate tonnage factors for each rock type.

- Holes should be re-plotted using the C.A.M.C. computer system.

e2qo

b) Geotechnical Model

A geotechnical model of the Grum should be constructed concurrently with the revised tonnage and grade model. Data required should be gathered during the re-logging program. Additional drilling may be required to extend this model to the ultimate pit limits.

Results of this program will be rock and overburden slope angles for pit walls to be used in mine design.

e2r3

c) Mine Planning

When the geotechnical and tonnage and grade models have been completed, production plans and schedules will be constructed. An important part of this planning will be the calculation of the break-even stripping ratio for open pit vs. underground ~~mining~~. ore .

e2r6

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4. Grum Computer Model and Pit Design - NORCOMP MODEL - December 1979.

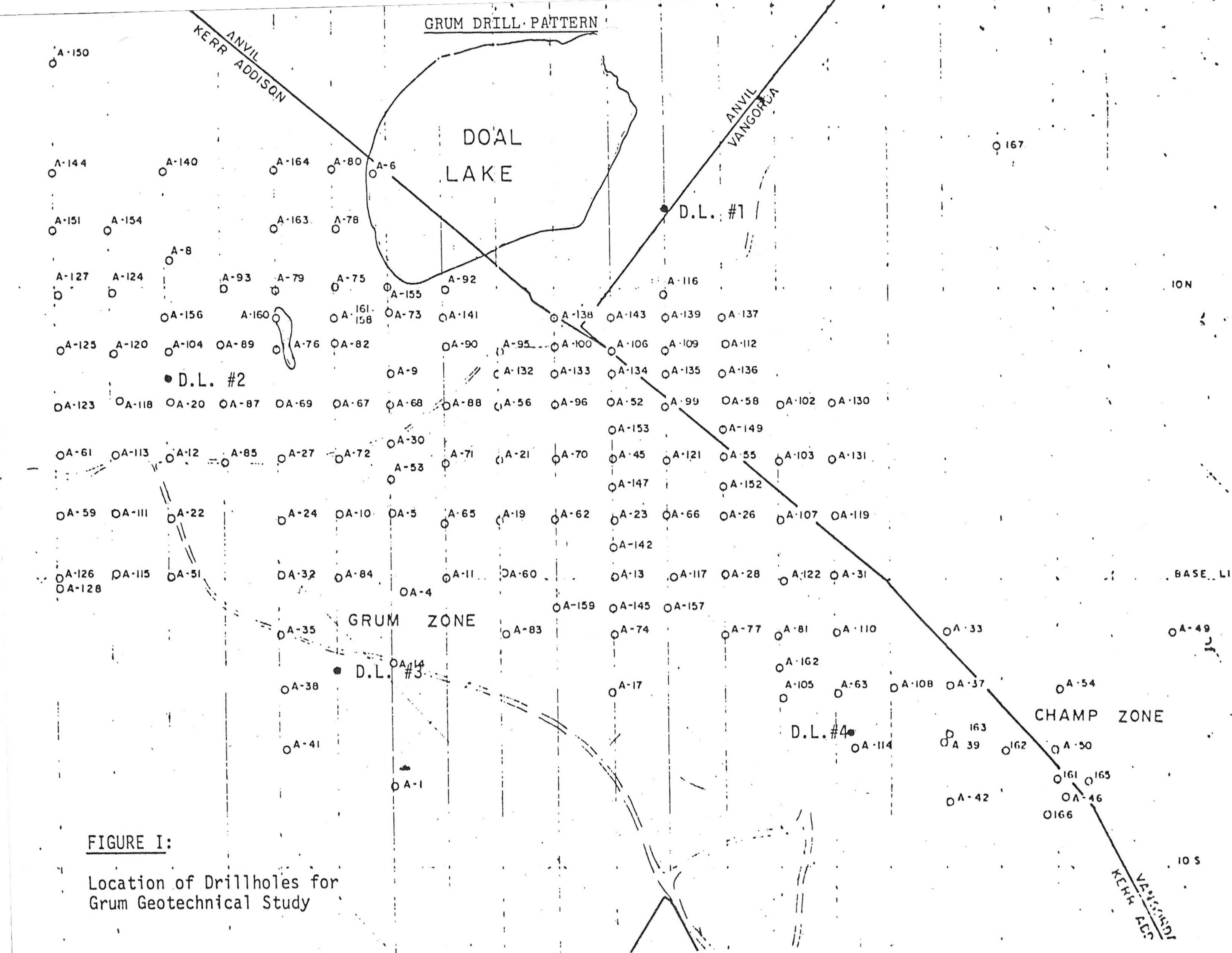
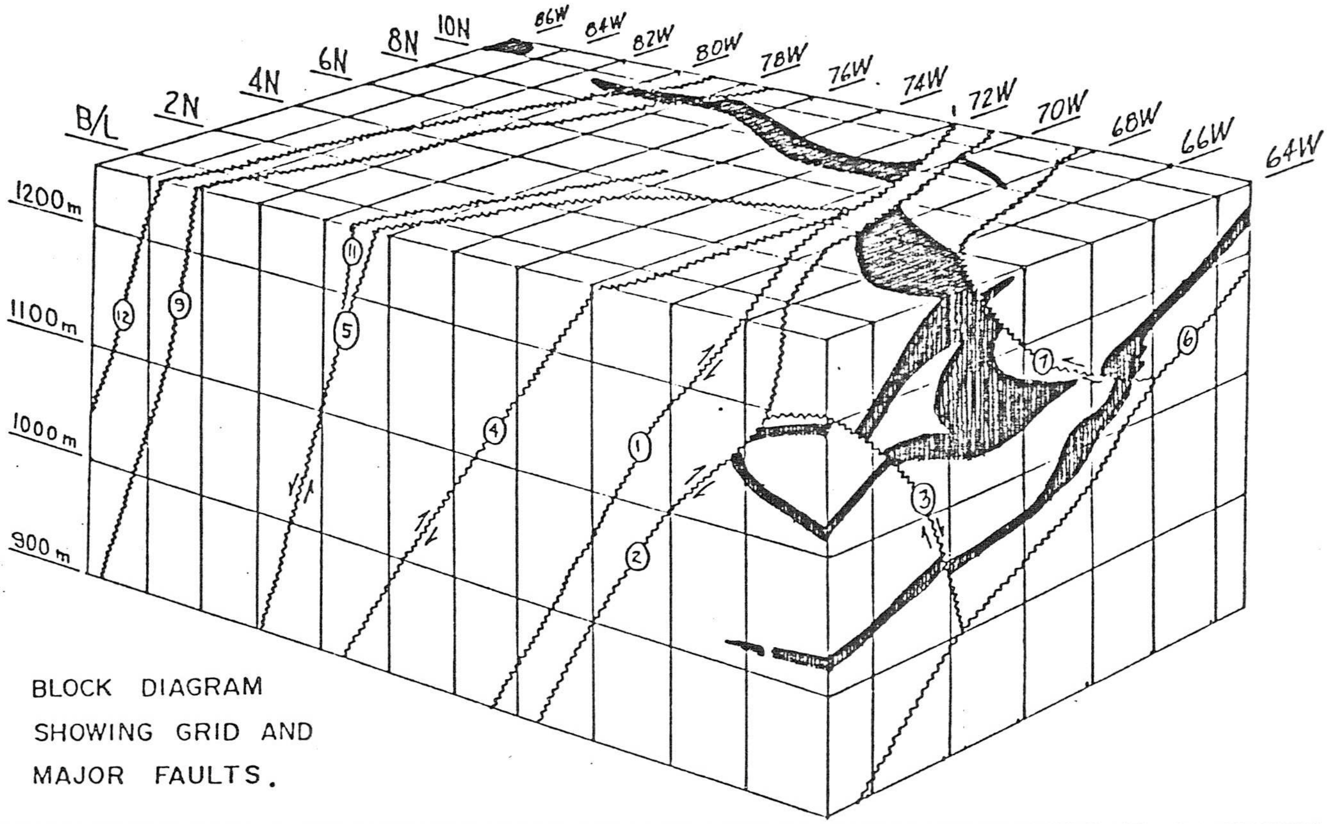


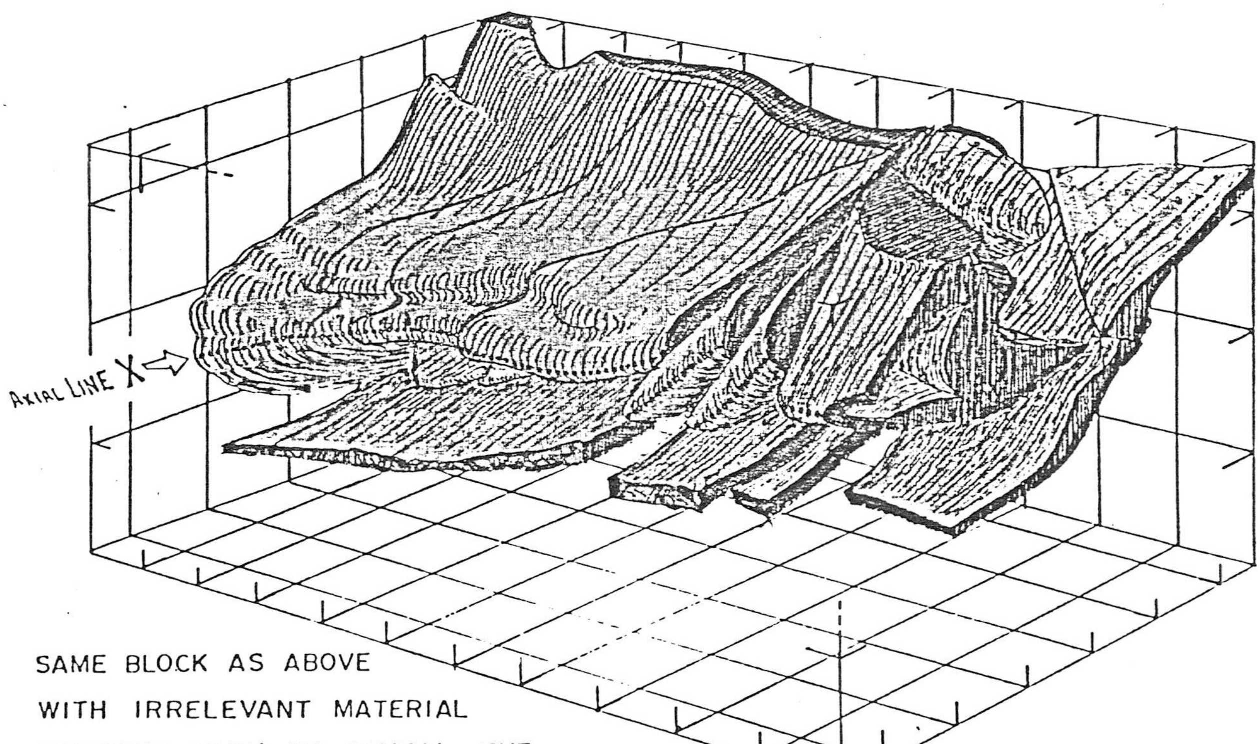
FIGURE I:

Location of Drillholes for Grum Geotechnical Study

STRUCTURAL DIAGRAMS - GRUM ORE ZONE



BLOCK DIAGRAM SHOWING GRID AND MAJOR FAULTS.

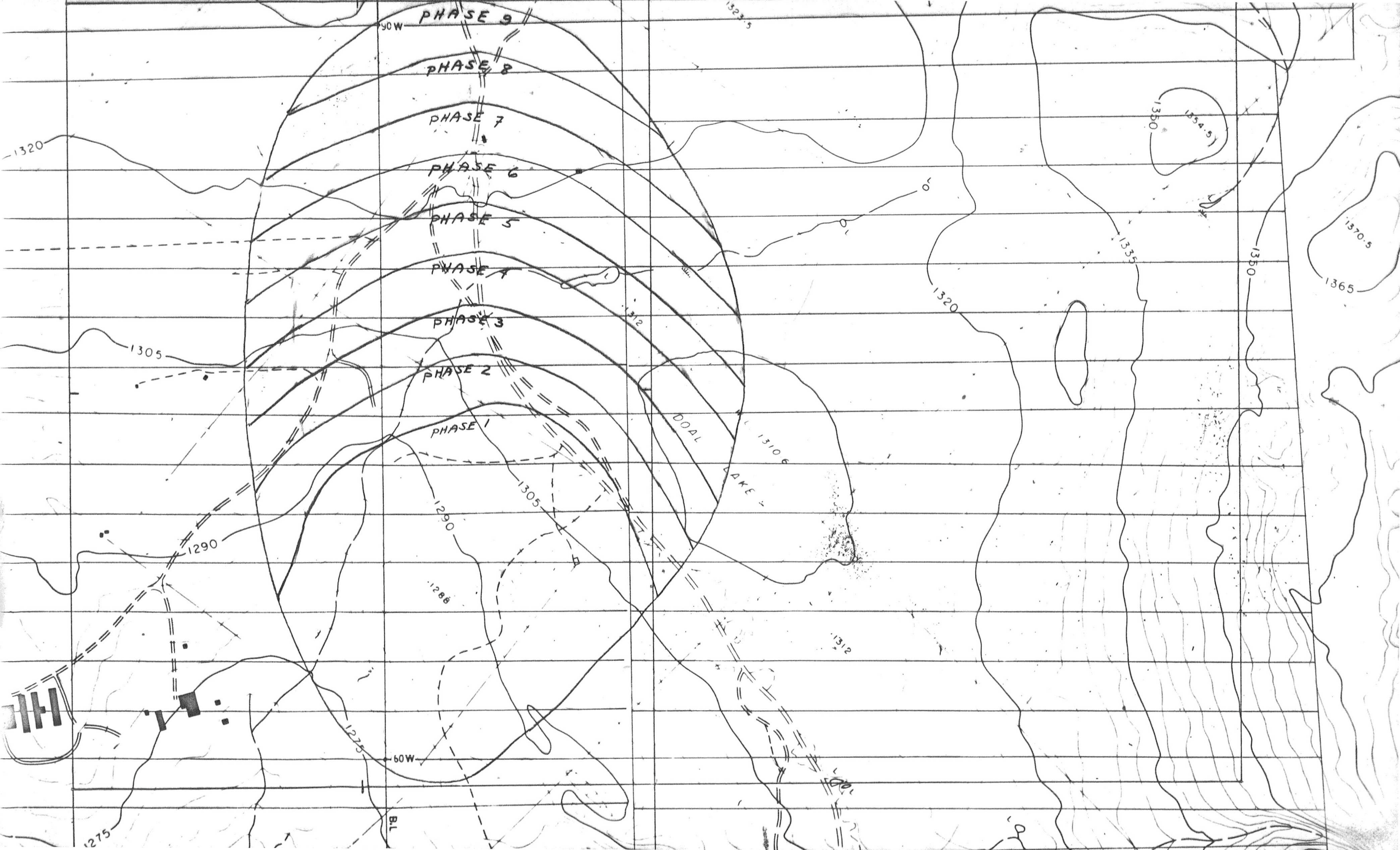


SAME BLOCK AS ABOVE WITH IRRELEVANT MATERIAL STRIPPED AWAY TO REVEAL THE X FOLD AXIS.

Jim Paxton
March 1977

FIGURE II

FIGURE 5



PHASE 9
PHASE 8
PHASE 7
PHASE 6
PHASE 5
PHASE 4
PHASE 3
PHASE 2
PHASE 1
90 W
60 W
B.L
DOAL
LAKE
1320
1305
1290
1275
1288
1312
1310.6
1335
1350
1354.5
1370.5
1365