RB 40

REPORT

on

INVESTIGATION OF URANIUM DEPOSITS AT MOUNT PAINTER, SOUTH AUSTRALIA.

June 1944 to September 1945.

Undertaken at the request of The British Government

by

The Government of the Commonwealth of Australia

in conjunction with

South Australian Government.

8TH NOVEMBER, 1945.

TABLE OF CONTENTS.

AUTHORISATION.

PART I. GENERAL STATEMENT.

PART II. CONSTRUCTION AND MINING OPERATIONS.

PART III. GEOLOGICAL SURVEY REPORTS.

PART IV. GEOPHYSICAL SURVEY REPORTS.

PART V. ORE TREATMENT REPORTS.

PART VI. MISCELLANEOUS REPORTS.

AUTHORISATION.

The investigation into the possibilities of the uranium deposits at Mount Painter, South Australia, were undertaken at the instance of the British Government under the auspices of the Commonwealth Department of Supply and Shipping.

The work was carried through under the direction of Mr. J. Malcolm Newman, B.E., Commonwealth Controller of Minerals Production; Sir David Rivett, K.C.M.G., M.A., D.Sc., F.R.S., Chief Executive Officer of the Council for Scientific and Industrial Research and Mr. S. B. Dickinson, M.Sc., Director of Department of Mines and Government Geologist of the State of South Australia.

PARTICIPATING SCIENTIFIC RESEARCH AND INVESTIGATIONAL ORGANIZATIONS.

A. COUNCIL FOR SCIENTIFIC AND INDUSTRIAL RESEARCH.

1. Division of Industrial Chemistry:

Dr. I.W. Wark, Ph.D., D.Sc.
(Chief of Division).

R. Grenfell Thomas, B.Sc.
P. Dixon, M.Sc.
T.R. Scott, M.Sc. B.Ed.

2. Mineragraphic Investigations:

Dr. F.L. Stillwell, D.Sc.
Dr. A.B. Edwards, Ph.D., D.Sc.

3. Ore-dressing Investigations (Adelaide):

Professor H.W. Gartrell, M.A., B.Sc. (deceased).
(Officer-in-Charge).

D.R. Blaskett, B.E.

4. Ore-Dressing Investigations (Melbourne):

H.H. Dunkin, B.Met.E.
(Officer-in-Charge).

J.G. Hart

5. Information Section:

J.S. Hosking, M.Sc., Ph.D.
(Officer-in-Charge).

B. GEOLOGICAL SURVEY OF SOUTH AUSTRALIA.

S.B. Dickinson, M.Sc.
(Government Geologist)

E. Broadhurst, B.M.E., M.Sc.
R.C. Sprigg, M.Sc.
H.J. Ward, B.Sc.
K.M. Llewellyn.
T.W. Dalwood.

C. MINERAL RESOURCES SURVEY, DEPARTMENT OF SUPPLY & SHIPPING:

Dr. H.G. Raggatt, D.Sc.
(Director)
P.B. Nye, B.M.E., M.Sc.
(Assistant Director)

J.M. Rayner, B.Sc.
R.F. Thyer, B.Sc.
J.C. Dooley, M.Sc.
C.J. Sullivan, M.Sc.
L.C. Noakes, B.A.
D.E. Gardner, B.Sc.

D. DEPARTMENT OF PHYSICS, UNIVERSITY OF ADELAIDE:

Professor Kerr Grant, M.Sc.
K.B. Mather, M.Sc.

E. DEPARTMENT OF GEOLOGY, UNIVERSITY OF ADELAIDE:

Professor Sir Douglas Mawson, O.B.E., D.Sc., B.E., F.R.S.
A.W. Kleeman, M.Sc.

F. PUBLIC LIBRARY OF SOUTH AUSTRALIA:

H.C. Bridson, B.A.
(Officer-in-Charge of Research Service).

PARTICIPATING ORGANIZATIONS IN OPERATIONAL ACTIVITIES.

1. SOUTH AUSTRALIAN DEPARTMENT OF MINES -

S.B. Dickinson, M.Sc.
(Director of Mines)
E.B. Jensen.
C.F. Duffield.

2. SOUTH AUSTRALIAN RAILWAYS -

C.B. Anderson, C.M.G.
(Railways Commissioner)
R.H. Chapman, B.E.
(Chief Engineer)
C.H. Williams.
F.J. Shaw.
T.R.L. Alderman.

3. SOUTH AUSTRALIAN DEPARTMENT OF ENGINEERING & WATER SUPPLY -

H.T.M. Angwin, C.M.G., B.E.
(Engineer-in-Chief)
R.G. Wilton
(Deputy Engineer-in-Chief)
E.R. Laurie.
J.R. Dridan, B.E.
G.G. Poole, B.E.
J.C. Gunner.
C.E.F. Jacob, B.E.

4. ROYAL AUSTRALIAN AIR FORCE -

Air-Commodore F.N. Wright, O.B.E., M.V.O.
Group-Captain L.C. Holswich
Squadron Leader G.H.N. Shiells, D.F.C. (Parafield)
Wing Commander E.A. Beaumont, A.F.C. (Mallala)
Squadron Leader L.R. Marsh (Port Pirie).

5. AUSTRALIAN ARMY -

Major General J.H. Cannan, C.B., C.M.G., D.S.O., V.D.
Major General C.E.M. Lloyd, C.B.E.
Lt. Col. J.G. Gillespie.
Major C.J.H. Grenfell.
Major P.C.A. Head.
Major R. Mainwaring.

6. MINISTRY OF MUNITIONS -

A.A. Topp
(Director Explosives Supply).
J.R. Cochrane, B.Sc.
W. Fowler, B.E.

7. DEPARTMENT OF SUPPLY AND SHIPPING -

J. Malcolm Newman, B.E.
(Controller of Minerals Production)
A.C. Smith
(Executive Officer, Minerals).
W.A. Hughes.
B.S. McCarthy.
P.J. Scanlan.

8. ZINC CORPORATION LTD., BROKEN HILL -

A.J. Keast
(General Manager)
J.P. Morgan, B.E.
R. Nankivell.

9. DEPARTMENT OF INTERIOR, CANBERRA -

F.M. Johnstone
(Surveyor-General)
A.C. Booth.

I N D E X.

AUTHORIZATION.

PART I. - GENERAL STATEMENT - URANIUM INVESTIGATION, MOUNT PAINTER, SOUTH AUSTRALIA.

by

S.B. DICKINSON

I. INTRODUCTION.

II. ORGANIZATION.

III. HISTORICAL REVIEW.

IV. PAST PRODUCTION.

V. REVIEW OF EXPLORATORY OPERATIONS.

1. Introduction.
2. Mount Painter Road.
3. First Geological Camp.
4. Balcanoona Aerodrome.
5. Mining operations at No.6 Workings.
6. Second Geological Camp.
7. East Painter Road.
8. Mining Operations at East Painter.

VI. EXPENDITURE.

VII. SUMMARY OF RESULTS.

VIII. CONCLUSION.

PART II. - CONSTRUCTION AND MINING OPERATIONS.

by

E.B. JENSEN

1. PLANS.

2. ACCESS ROADS TO DEPOSITS.

3. AERODROME AT BALCANOONA.

4. TELEPHONE COMMUNICATION.

5. WATER SUPPLIES.

A. Water Supply at Mount Painter Camp.
B. Water Supply at East Painter Camp.
C. Cost of Water Supplies.
   (i) Cost of boreholes.
   (ii) Cost of carting water.
   (iii) Cost of pumping water at Mount Painter Camp.
D. Additional Sources of Water.

6. ELECTRIC LIGHTING.

7. SANITATION AND HYGIENE.

8. CONSTRUCTION, MOUNT PAINTER CAMP.

    A. Introduction.
    B. Carting.
    C. Construction Schedule.
        (i) Engine house and machinery.
        (ii) Haulage way.
        (iii) Air and Water lines to mine faces.
        (iv) Buildings.
        (v) Sleeping quarters and other buildings.

9. CONSTRUCTION, EAST PAINTER CAMP.

10. PREPARATION OF MINE FACES AT MOUNT PAINTER.

11. DEVELOPMENT AND EXPLORATION FOOTAGES.

12. DESCRIPTION OF MINING OPERATIONS.

    A. Introduction.
    B. Mining Methods.
    C. Plate laying.
    D. Underground trucking.
    E. Surface prospecting shaft at Mount Painter.
    F. Diamond Drilling

13. GENERAL CAMP COSTS.

    A. Mess organization.
    B. Supervision at Mines.
    C. Surveying.
    D. Supervision at Copley.
    E. Holidays and travelling time.

14. MISCELLANEOUS COSTS.

15. DISPOSAL OF PLANT AND EQUIPMENT.

16. ACKNOWLEDGEMENTS.

PART III. - GEOLOGICAL SURVEY REPORTS.

I. URANIUM MINERALS FROM MOUNT PAINTER.

by

F.L. STILLWELL and A.B. EDWARDS.

1. OCCURRENCE.

2. URANIUM MINERALS.

    A. Autunite.
    B. Torbernite and Meta-Torbernite.
    C. Uranophane.
    D. Hydrous Uranium Silicate.
    E. Gummite.
    F. Fergusonite.

3. ASSOCIATED MINERALS.

4. WEATHERING OF TORBERNITE.

5. BOXWORK STRUCTURES.

6. ORIGIN OF URANIUM MINERALS.
7. PRIMARY MINERALS.
8. CONCLUSION.
9. LIST OF REFERENCES.
10. LIST OF ILLUSTRATIONS.

II. RECONNAISSANCE GEOLOGICAL SURVEY OF THE MOUNT PAINTER AREA

by

R.C. SPRIGG.

1. PLANS.
2. INTRODUCTION.
3. ORGANIZATION AND SCOPE OF WORK.
4. CLIMATE, RAINFALL, VEGETATION AND FAUNA.
5. PHYSIOGRAPHY.
    - A. Landform analysis by topographic maps.
6. GEOLOGY.
    - A. Previous investigations.
    - B. General.
    - C. The Sedimentary Sequence.
    - D. Stratigraphical Sequence.
    - E. The Igneous Rocks.
        - (i) The Red Granite.
        - (ii) Leucogranites.
        - (iii) The Basic Igneous Rocks.
    - F. Metamorphic Rocks.
    - G. Geological Structures.
        - (i) Folding.
        - (ii) Faulting.
            - (a) The Paralana Group.
            - (b) The Mount Saturday Radial Faults.
    - H. Epithermal Activity.
    - I. The Age of the Erosion Surfaces.
    - J. Geological History of the Area.
    - K. References.

III. REPORTS ON INDIVIDUAL URANIUM OCCURRENCES.

1. MOUNT PAINTER GROUP
    - A. Nos. 1, 2 and 3 Workings, Radium Ridge by C. J. Sullivan.
        - (i) No. 1 Working.
        - (ii) No. 2 Working.
        - (iii) No. 3 Working.

B. No.4 Workings by C.J. Sullivan.
    (i) Situation.
C. No.5 Workings by E. Broadhurst.
D. No.6 Workings by E. Broadhurst.
    (i) Location.
    (ii) Geology.
    (iii) Ore Deposit.
    (iv) Development.
    (v) Future Development.
    (vi) Results of Sampling.
    (vii) Diamond Drill Logs.
E. No.7 Workings by E. Broadhurst.
    (i) Location.
    (ii) Geology.
    (iii) Workings.
F. No.8 Workings by C.J. Sullivan
    (i) Situation.
    (ii) Geology and Workings.
G. Locality "G" by R.C. Sprigg.
H. Locality "J" by R.C. Sprigg.

3. EAST PAINTER GROUP.

A. General Statement by E. Broadhurst
    (i) Introduction.
    (ii) Geology (East Painter or Greenwood's Camp Area).
    (iii) Geiger Muller Survey.
B. Nos.1 and 2 Workings, East Painter (Greenwood) Camp by E. Broadhurst.
    (i) Description.
    (ii) Sampling results.
    (iii) Conclusion.
C. No.3 Workings, East Painter (Greenwood) Camp by E. Broadhurst.
D. No.5 Workings, East Painter (Greenwood) Camp by E. Broadhurst.
    (i) Development.
    (ii) Geology.
    (iii) Ore occurrences.
    (iv) Conclusions.
E. No.1E Workings, East Painter (Greenwood) Camp by C.J. Sullivan.
F. No.2E Workings by C.J. Sullivan.
G. No.4E Workings by C.J. Sullivan.
H. Bentley Greenwood Deposit by C.J. Sullivan.
    (i) Situation.
    (ii) Geology.
    (iii) Uranium Minerals.
    (iv) Testing.
I. Smiler Greenwood Deposit by C.J. Sullivan.
    (i) Situation.
    (ii) History.
    (iii) Geology.
    (iv) Testing.
    (v) Conclusion.
J. Locality "B" by C.J. Sullivan.

3. PARALANA HOT SPRINGS GROUP.
    A. Localities D., E. and F. by E. Broadhurst.
4. REFERENCES.

IV. A POSSIBLE EXPLANATION OF THE GENESIS OF THE MOUNT PAINTER URANIUM DEPOSITS, LARGELY BASED ON FIELD OBSERVATIONS

by

C.J. SULLIVAN.

1. INTRODUCTION.
2. BRIEF DESCRIPTION OF THE DEPOSITS.
3. IRONSTONE.
4. ASSOCIATION OF THE SECONDARY URANIUM MINERALS WITH IRONSTONE.
5. LEACHING OF URANIUM.
6. AGE OF THE TORBERNITE.
7. TEMPERATURE OF FORMATION OF TORBERNITE.
8. ASSOCIATION OF URANIUM DEPOSITS WITH GRANITE.
9. TORBERNITE DEPOSITS IN OTHER PARTS OF THE WORLD.
10. SUGGESTED HISTORY OF THE FORMATION OF THE DEPOSITS.
11. CONCLUSIONS.

V. ORIGIN OF THE MOUNT PAINTER URANIUM DEPOSITS

by

E. BROADHURST.

1. INTRODUCTION.
2. CASE 1.
3. CASE 2.
4. CASE 3.
5. CASE 4.
6. CASE 5.
7. CONCLUSIONS.

PART IV. - GEOPHYSICAL SURVEY REPORT.

GEOPHYSICAL SURVEYS OF URANIUM DEPOSITS, MOUNT PAINTER

by

R.F. THYER and J.C. DOOLEY.

I. INTRODUCTION.

II. FUNDAMENTALS OF RADIOACTIVITY.

III. RADIOACTIVITY IN THE MOUNT PAINTER DISTRICT.

IV. THE GEOPHYSICAL SURVEYS.
    A. Surveys Undertaken.
        (i) Field work.
        (ii) Estimation of Uranium in Samples.
    B. The Instruments used.

V. FIELD SURVEYS.
    A. Procedure.
    B. Results of Field Surveys.
        (i) The Smiler Greenwood Area, East Painter.
        (ii) The Bentley Greenwood Area, East Painter.
        (iii) No.6 Workings, Mount Painter.
        (iv) East Painter (Greenwood) Camp Area.
        (v) Radium Ridge Area, Mount Painter.

VI. DETERMINATION OF URANIUM CONTENT BY RADIATION MEASUREMENTS.
    A. Introduction.
    B. Theoretical Considerations.
    C. Equipment and Procedure.
    D. Calibration.
    E. Tests on Unassayed Mine Samples.

VII. GENERAL CONCLUSIONS.

VIII. RESULTS OF ACTIVITY TESTS ON MINE SAMPLES.

IX. REFERENCES.

PART V. - ORE TREATMENT REPORTS.

I. URANIUM ORES FROM MOUNT PAINTER (first report)

by

PROFESSOR H.W. GARTRELL and D.R. BLASKETT.

1. SAMPLES.

2. INFORMATION REQUIRED.

3. DESCRIPTION OF SAMPLES.
    A. Autunite.
    B. Torbernite and Meta-torbernite.

4. GENERAL DISCUSSION ON METHODS OF CONCENTRATION.
    A. Wet processes.
    B. Dry processes.

5. DETAILS OF TESTS.
    A. Gravity concentration tests.
    B. Flotation.
    C. Differential Grinding.
    D. Magnetic concentration.
    E. Sticky-Surface concentration.
        (i) Sizing of material for testing.
        (ii) Testing procedure and results.

6. SUMMARY.

II. <u>URANIUM ORES FROM MOUNT PAINTER</u> (second report)

by

<u>PROFESSOR H.W. GARTRELL and D.R. BLASKETT</u>.

1. GENERAL.

2. DIFFERENTIAL GRINDING.
    A. Details of Tests.

3. STICKY-SURFACE CONCENTRATION TEST ON ORE FROM
   50 FT. LEVEL WINZE NO.6 WORKINGS, MOUNT PAINTER.

4. SUMMARY.

III. <u>DIFFERENTIAL GRINDING OF URANIUM ORES FROM MOUNT PAINTER</u> (third report).

by

<u>PROFESSOR H.W. GARTRELL and D.R. BLASKETT</u>.

1. GENERAL.

2. DIFFERENTIAL GRINDING.
    A. Details of Tests.
        (i) Series A.
        (ii) Series B.

3. DISCUSSION OF RESULTS.

IV. CONCENTRATION TESTS ON URANIUM ORES FROM MOUNT PAINTER (fourth report)

by

PROFESSOR H.W. GARTRELL and D.R. BLASKETT.

1. GENERAL.
2. DETAILS OF TESTS.
   - A. Test A.
   - B. Test B.
3. SUMMARY.

V. CONCENTRATION TESTS ON URANIUM ORES FROM MOUNT PAINTER (fifth report)

by

D.R. BLASKETT.

1. GENERAL.
2. DESCRIPTION OF THE STICKY-SURFACE CONCENTRATING MACHINE.
3. DETAILS OF TESTS.
   - A. Test A.
   - B. Test B.
4. GENERAL SUMMARY OF CONCENTRATION PROCESSES TESTED ON MOUNT PAINTER ORES.
   - A. Introduction.
   - B. Concentration Processes.
     - (i) Wet Processes of Concentration.
     - (ii) Dry Processes of Concentration.
   - C. Conclusion.

VI. ELECTROSTATIC CONCENTRATION OF URANIUM ORES FROM MOUNT PAINTER AND EAST MOUNT PAINTER, STH. AUSTRALIA.

by

H.H. DUNKIN and J.G. HART.

1. SYNOPSIS.
2. MINERAGRAPHIC WORK.
3. SIEVES.
4. NOTES ON THE ELECTROSTATIC METHOD USED.
5. MOUNT PAINTER SAMPLE.
   - A. Preparation and Sizing.
   - B. Magnetic Treatment.

    C. Electrostatic Treatment of Non-magnetics.
    D. Overall Magnetic-electrostatic Recoveries.
    E. Electrostatic Treatment of Magnetic Fractions.

6. EAST PAINTER SAMPLE.

    A. Preparation of Samples.
    B. Magnetic Treatment.
    C. Electrostatic Treatment of Non-magnetics.
    D. Overall Magnetic-electrostatic Treatment.

7. SUMMARY.

8. CONCLUSIONS.

VII. THE RECOVERY OF URANIUM FROM TORBERNITE-AUTUNITE ORES OF MOUNT PAINTER, SOUTH AUSTRALIA

by

P. DIXON and T.R. SCOTT.

1. SUMMARY.

2. INTRODUCTION.

3. ANALYTICAL METHODS.

    A. Uranium.
    B. Iron.

4. SOLUBILITY OF TORBERNITE AND AUTUNITE.

    A. Solubility in Acids.
    B. Mass Action Effects.
    C. Rate and Selectivity of Leaching.

5. LEACHING OF MOUNT PAINTER ORE.

    A. Single Extraction Leaching.
    B. Counter-current Leaching.
    C. Cyclic Process with Counter-current Leaching.

6. RECOVERY OF URANIUM FROM LEACH LIQUORS.

    A. Reduction.
    B. Precipitation.

7. THE RECOVERY OF RADIUM.

8. DISCUSSION OF METHODS.

9. CONCLUSIONS.

10. ACKNOWLEDGEMENTS.

11. REFERENCES.

VIII. METHOD OF ASSAY ADOPTED FOR THE ESTIMATION OF URANIUM IN MOUNT PAINTER ORE

by

T.W. DALWOOD.

1. INTRODUCTION.

2. PROCEDURE.

PART VI. - MISCELLANEOUS REPORTS.

I. PRELIMINARY REPORT ON SPECTROGRAPHIC ANALYSES OF URANIFEROUS ORES BY PROFESSOR KERR GRANT AND K.B. MATHER.

II. NOTES ON THE PHOTOGRAPHIC ACTION OF RADIOACTIVE MATERIALS BY PROFESSOR KERR GRANT AND K.B. MATHER.

III. TOPOGRAPHIC MAP OF MT. PAINTER AREA, SOUTH AUSTRALIA.

REPORT ON COMPILATION METHODS USED BY A.C. BOOTH.

1. Introduction.
2. Procedure.
3. Conclusion.

LIST OF PLANS

Sheet No.	Plan No.	Location	Title
1	3184	No. 6 Workings Mount Painter	Surface Geological Plan
2	2951	" "	Surface Assay Plan
3	3176	" "	25' Level Geological Plan
4	3179	" "	25' Level Assay Plan
5	3175	" "	50' Level Geological Plan
6	3183	" "	50' Level Assay Plan
7	3177	" "	80' Sub-level Geological Plan
8	3180	" "	80' Sub-level Assay Plan
9	3178	" "	100' level Geological Plan
10	3182	" "	100' level Assay Plan
11	3185	" "	Composite Mine Plan
11A	3193	" "	Composite before October '44
12	3187	" "	S.N. Section at Main Shaft
13	3196	" "	S.N. Section through No. 1, 2 D.D. Hole
14	3190	" "	S.N. Section through No. 3, 4 D.D. Hole
15	3181	" "	W.E. Section at 1200S
16	3188	" "	W.E. Projection on main lode showing geology on south side
17	3189	" "	W.E. Projection on main lode showing geology on north wide
18	3186	" "	W.E. Projection on main lode showing position of samples
19	2997	No. 7 Workings Mount Painter	Surface Plan
20	2949	East Painter area	Greenwood's camp
21	3191	No. 5 Workings East Painter	Surface plan and inset of 40' level
22	3195	" "	Assay plan and projection
23	3170	Copley to Mt. Painter	Road Plan
4	3174	Mt. Painter area	Surface layout
25	3194	East Painter area	Surface layout
26	3206	Mount Painter area	Cubicles camel transport
27	3207	" " "	Men's sleeping quarters
28	3200	" " "	Plan of sledge hauling
29	3198	" " "	Diagram showing lay-out of holes for drag cut in driving
30	3199	" " "	Diagram showing lay-out of holes for post hole cut in driving
31	3285	South Australia	Radio-active occurrences
32	2979	Radium Hill	Locality plan
33	3286	Australia	Showing Mount Painter and Radium Hill
34	3201	Mount Painter No. 6 Workings	No. 6 D.D. Hole Section through
35	3205	East Painter	Smiler Geological plan and section
36	3204	" "	Bentley Geological plan

0

PART I. GENERAL STATEMENT - INVESTIGATION OF URANIUM DEPOSITS.

Mount Painter, South Australia.

by

S. B. DICKINSON

Director of Mines, Government Geologist
South Australia.

(Plans 3286, 3285, 2979, Sheets 31, 32, 33 illustrate this report).

I. INTRODUCTION.

On 17th May, 1944, the Hon. F.M. Forde, M.H.R. Acting Prime Minister of Australia was urgently requested by the British Government to have immediate steps taken to investigate the possibilities of the production of uranium from Australian ore deposits. In response to the request attention was at once given to the old Mount Painter and Radium Hill fields of South Australia where during the past 40 years a number of attempts had been made by small mining interests to exploit their deposits for the recovery of radium. Although these ventures were financially unsuccessful, a small quantity of radium was produced, but the recovery of uranium was not considered seriously both on account of unfavourable market conditions and the difficulties of the treatment of the ore. Apart from these attempts no other occurrences of radio-active minerals have been of anything beyond scientific interest.

Mount Painter is situated 94 miles by road north-east of Copley, which is 373 miles by rail from Adelaide on the Quorn-Alice Springs line. Radium Hill lies 12 miles south south-east of Cutana Siding, 275 miles from Adelaide on the Broken Hill railway line (see locality map). Both these fields occur in Crown land reservations in which mining operations now are completely under the control of the South Australian Government. In the event of new deposits being discovered outside the reservations, a war-time emergency regulation limiting the sale of uranium and uranium ores to the South Australian Government, or through its agency, has been enacted. Steps are also being taken to ensure a continuance of this control after the removal of National Security Regulations by an amendment to the South Australian Mining Act.

The investigations at the Mount Painter and Radium Hill fields were planned and executed solely under conditions imposed by the exigencies of war without regard for strict economic principles, definitely only with the object of assisting the purposes of the Government of the United Kingdom.

II. ORGANISATION.

The primary objective of the investigation was to determine the capacity of the Mount Painter and Radium Hill fields for producing uranium and for acquiring this knowledge in the minimum of time.

Sheet No.	Plan No.	Location	Title
37	3197	Mount Painter area	Radium Ridge preliminary Geological Plan
38	3203	" " "	No. 4 prospect geological plan and Section
[illegible]	3215	" " "	No. 8 Prospect
[illegible]	3202	" " "	Mt. Painter Trig Survey
[illegible]	3209	" " "	Mt. Painter and East Painter locality plan of bores
[illegible]	3208	" " "	Balcanoona Aerodrome
[illegible]	3210	" " "	Mt. Painter area Land form analysis
[illegible]	3287	" " "	Yudnamutana Gorge shifting divide
45A	3288	" " "	Mt. Painter Area Geological Reconnaissance maps
45B	3289	" " "	" " " " " "
45C	3290	" " "	" " " " " "
45D	3291	" " "	" " " " " "
46A	3292	" " "	N.S. Section across 45A,B,C,D
47	3294	" " "	N.E. Flinders Ranges; sketch Geological reconnaissance map and N.S. Section
48	3299	" " "	Geiger Muller Detector, Assay Model 3
49	3300	" " "	Portable Geiger Muller Headphones detector
50	3301	" " "	Locality plan
51	3302	" " "	Graphs Radioactivity and Uranium Content
52	3303	" " "	" " " " "
53	3304	" " "	Gamma Ray Absorption Curves
54	3305	East Painter Area	Smiler Greenwood Radiation Contours
55	3306	" " "	Bentley Greenwood Radiation "
[illegible]	3307	Mount Painter Area	No. 6 Deposit Mount Painter
[illegible]	3308	East Painter	Mount Painter East Locality Plan
[illegible]	3309	" "	Radiation contours
[illegible]	3310	" "	Radiation contours

The work was carried out under the instructions of Mr. J. Malcolm Newman, Commonwealth Controller of Minerals Production and in conjunction with the Department of Supply and Shipping.

The organisation of the work in the field and that of the engineers and workmen, and also of the scientists whose services were required in the investigation, was directed by Mr. S. B. Dickinson, Director of Mines and Government Geologist of South Australia.

FIG. 1. Mount Painter and No.6 Workings. re photo

FIG. 1. Mount Painter and No.6 Workings.

The area surrounding Mount Painter is in the trackless inhospitable mountain country of the Flinders Ranges. Consequently for the transportation of stores and materials access roads had to be constructed and heavy mechanical plant, besides pack camels and horses was needed. Also an aerodrome was made at Balcanoona, distant about 25 miles from Mount Painter.

Owing to the time consumed in these preparatory operations the actual mining operations had to be compressed into a period of less than three months. These operations were carried out without a hitch and no praise is too high for the working miners who spared no effort to get results and to contribute to a successful issue.

For the scientific work besides the strong and well-equipped surveying, geological and geophysical field parties, several research authorities willingly lent their aid for the study of mineral associations and the recovery of the uranium from the different ores.

The execution of the investigation according to plan was largely due to the solid co-operation of all bodies participating in the work, and with the exception of a small sum expended in fees and allowances to special technical officers, all the scientific services referred to below were rendered without charge.

The Council for Scientific and Industrial Research arranged for various lines of research work to be undertaken expeditiously. These included the mineragraphic studies by Dr. F.L. Stillwell and Dr. A.B.Edwards, the leaching of uranium ores under the direction of Mr. Grenfell Thomas, and the ore-dressing investigations in the Bonython Laboratory of the South Australian School of Mines and in the Metallurgy School of the University of Melbourne.

The Geological Survey of South Australia, with assistance of the Mineral Resources Survey, carried out the geological mapping and sampling, and the Assay Department of the South Australian School of Mines assayed the samples and undertook a number of precise determinations for special purposes.

The Mineral Resources Survey conducted the geophysical surveys and detailed laboratory studies on the assaying of the uranium ores by geophysical methods. In both these activities the Physics Department of the Adelaide University rendered much assistance.

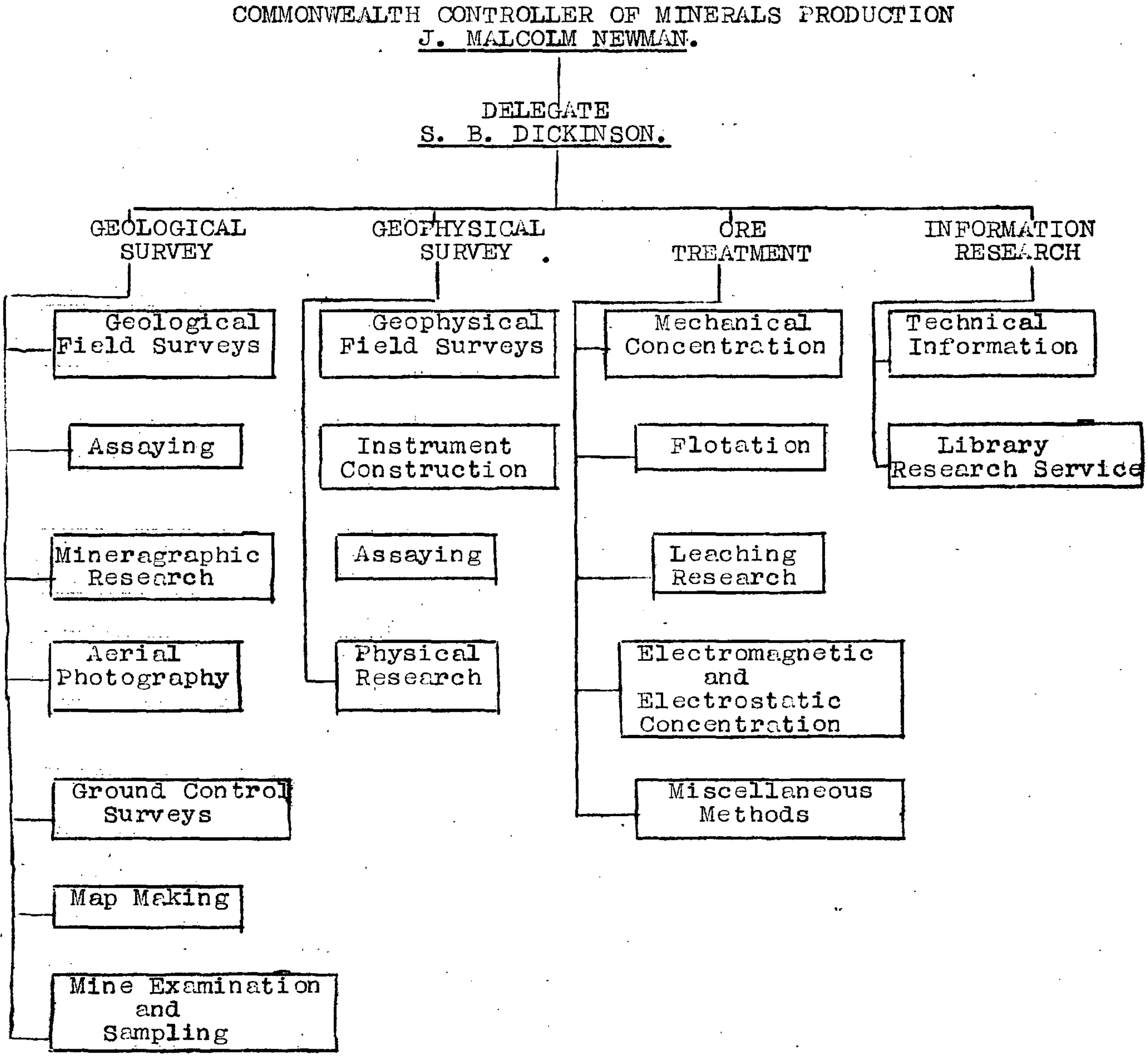
Sir Douglas Mawson, Professor of Geology at the University of Adelaide, furnished many scientific reports and arranged for certain studies, both of which have contributed towards a better understanding of the origin of the uranium deposits.

In addition to the work done by scientists, the Royal Australian Air Force took aerial photographs and provided aircraft for the transport of key personnel to and from the Mount Painter field; the Australian Army prepared photomaps and furnished the manpower for mining operations and certain special equipment; the Engineering and Water Supply Department of the South Australian Government built the access roads and aerodrome; the Zinc Corporation Ltd., Broken Hill, made available mining engineers and plant, and other Commonwealth and State Government Departments all made useful contributions to the investigation.

The following organization charts indicate the chief sub-divisions of the investigations:-

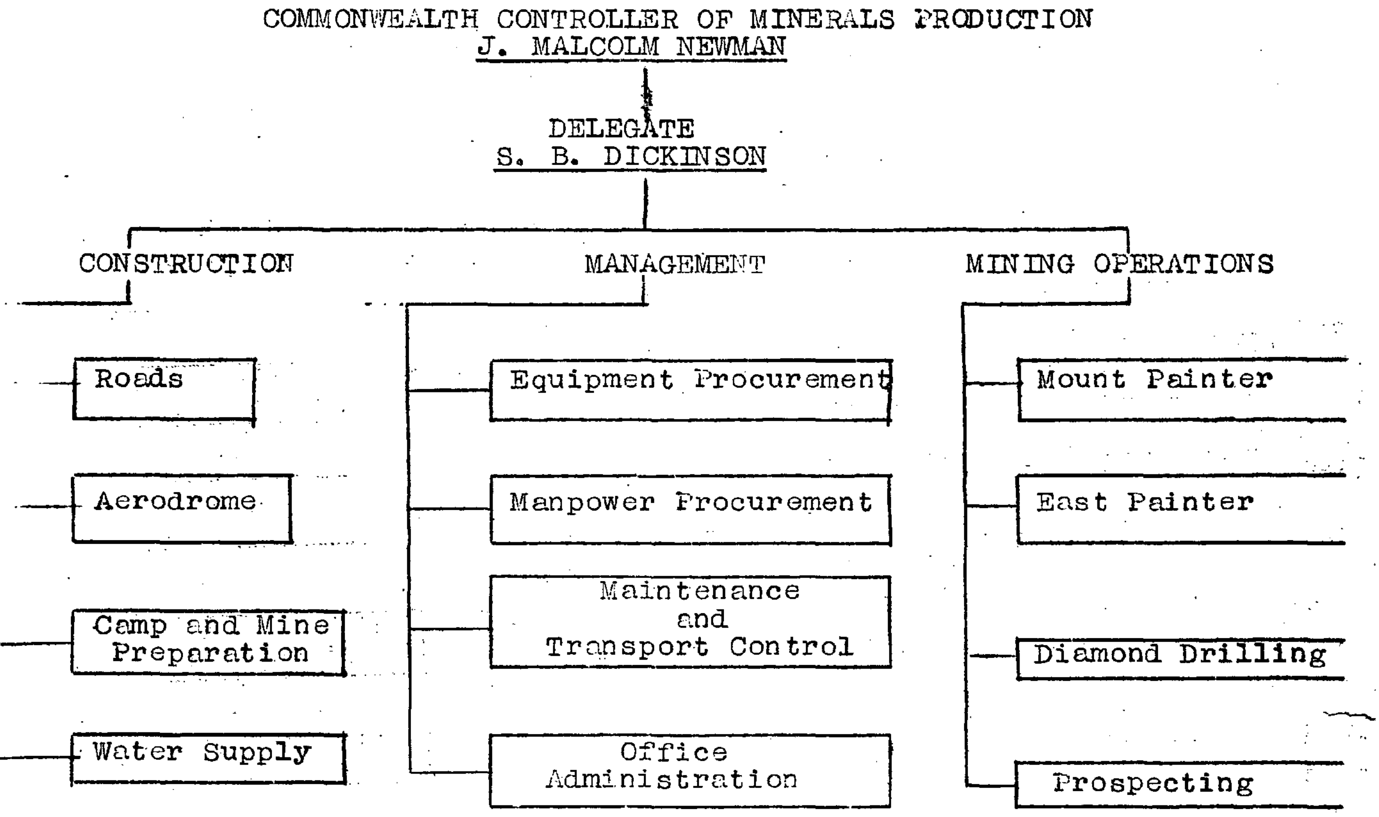
ORGANIZATION OF SCIENTIFIC INVESTIGATIONS.

MOUNT PAINTER PROJECT.



ORGANIZATION OF WORKS PROGRAMME.

MOUNT PAINTER PROJECT.



The several scientific and technical bodies, and the personnel directly and indirectly concerned with preparation of the various reports are listed in a schedule at the beginning of this Bulletin.

III. HISTORICAL REVIEW.

The original discovery of uraniferous minerals near Mount Painter was made in 1910 by W.B. Greenwood, who, since 1898, had been employed and subsidised periodically by the South Australian Government as a prospector for the Mount Painter and Yudnamutana district. One of the specimens sent to the Department of Mines was identified as containing carnotite, other specimens from the same line of outcrop left at the Adelaide University were found by Sir Douglas Mawson to contain torbernite.

These discoveries aroused immediate interest in the field as a possible source of radium and following an examination by Sir Douglas Mawson of the outcrop from which these samples were derived, the Radium Extraction Company of South Australia Ltd., with a nominal capital of £5,000, was incorporated on 28th November, 1910, to explore further and develop the field. Within a few weeks a prospecting party was despatched to commence work on Radium Ridge where the ore was first found.

The party was in charge of H. Fabian, former Government prospector, who had previously made valuable discoveries of gold near Tarcoola. W.B. Greenwood was employed by the Company to prospect an area of eight square miles covered by search licences and A.C. Broughton of the Adelaide University accompanied the party to undertake measurements of radioactivity with an electroscope. Two other companies, the Mount Painter Proprietary Co. Ltd., and the Mount Painter East Prospecting Syndicate, pegged claims but did little work.

During the course of the first six months this Company opened up Nos. 1, 2, 3 and 4 Workings for a few feet in depth, but owing to extremely hard ground Nos. 1 and 2 were abandoned. When operations became handicapped by lack of water, the Government agreed to provide funds for the sinking of wells and this work was pursued concurrently with the exploration. At No. 3 Workings a shaft was sunk 50 feet with a crosscut 25 feet north. At No. 4 a drive was put in 25 feet under the outcrop, and at No. 5 Workings a shallow excavation 7 feet deep was made. The No. 6 Workings found by H. Fabian in May 1911 turned out far the most promising and yielded high grade ore from the first out, almost pure torbernite forming a rich seam, two to three inches wide, on the hanging wall side of a lode formation. As sinking proceeded the ore extended over the full width of the shaft. This was the deposit with which the recent investigations were chiefly concerned. No. 7 deposit was opened up shortly after No. 6. At the end of twelve months operations 15 men were employed and by this time a small supply of water was developed at Mount Painter but no supply had been obtained at Blue Mine Creek, the terminal of the road from whence the camel transport to the field commenced. Ore produced for the year amounted to a few tons of rich "specimen" character some of which was sent overseas. In 1911 the first report on the field by H.Y.L. Brown, Government Geologist (1) was published and describes the features of the Nos. 1-6 Workings. The Company now realised the necessity for concentrating the ore and for extracting the radium and uranium in Australia. In October, 1911, H. Greenway was appointed manager, replacing H. Fabian who asked to be relieved, and the ensuing work comprised chiefly the development and production from the No. 6 Workings. The main shaft was sunk to a depth of 100 feet and a tunnel was commenced to intersect the shaft at a depth of 200 feet. The field was visited by Dr. L.K. Ward (2), Government Geologist, 1912 and his report published in Mining Review 17, p. 27-33, gives a detailed account of the operations to that date. At the time of this visit no work was in progress, both H. Greenway and his successor G. Kemp having resigned and left the field. The development work recommended by Dr. Ward was undertaken shortly after by the Company under the direction of H.G. Stokes. Dr. Ward (3) visited the field again in May 1914 by which date the limits of the rich shoot at No. 6 had been determined and production was being maintained solely by extracting rich ore from around the shaft near the surface. Dr. Ward stated the ore reserves to be 300 tons, containing approximately 0.5% of uranium trioxide, in addition to 700 tons of reject material carrying 0.75-1.0% uranium trioxide.

Operations were interrupted by the outbreak of the first European war as the shipments of ore had been made chiefly to European countries. In 1917 the Radium Extraction Company of South Australia Ltd. went into liquidation after having spent £7,000. Apart from the failure to develop a sufficient tonnage of high grade ore and the difficulty of exploiting the low grade ore, this enterprise was severely hampered from the beginning by the then insuperable natural difficulties in a region of trackless rugged mountains devoid of permanent water.

A keen market for radium led to renewed interest in the Mount Painter field in 1924 when four companies financed by Victorian and South Australian shareholders were formed. In 1926 they were merged into the Australian Radium Corporation N.L., which worked the deposits intermittently until 1934. An experimental treatment plant and refinery (4) was erected at Dry Creek, 6 miles from Adelaide, to which hard-sorted ore and crude concentrates were carried about 450 miles from the field and a mining camp was re-established near the No.6 Workings. Transport to and from Mount Painter presented great difficulties. Loadings were necessarily small with pack camels moving supplies and ore for seven miles between the mines and Blue Mine Creek at the end of the road from Copley 75 miles distant on the Port Augusta-Alice Springs narrow gauge railway. Owing to the existence of this camel section it was impossible to transport heavy machinery to the mine. The largest unit on the field was a 7½ H.P. Ruston-Hornsby oil engine which was carried on a sleigh dragged by camels. It was used to drive a miniature crushing and screening plant which gave unsatisfactory results (5). Lack of water prevented any leaching treatment being established.

Like the earlier venture this attempt was unsuccessful owing to the low grade of the ore, the difficulty of water supply and the extremely costly transportation. The amount of underground development work during the period was extremely small for the time over which the leases were held, and the uncertainty as to the extent and nature of the ore occurrence at No.6 Workings still remained at the cessation of operations in 1934. Dr. L.K. Ward (5) visited the field for the third time in 1926 and reported on the large number of lode formations showing traces of torbernite which indicated the possibility of the discovery of other ore shoots by more intensive prospecting and the large extent of rough unprospected country east and south-east of Mount Painter. In 1929 the Chief Inspector of Mines, Mr. L.J. Winton (6), visited the field and stressed the need for further prospecting and development including an investigation of the possible occurrence of primary radio-active minerals at depth in the following words:-

> "Rich ore has been found in the past and new occurrences of such may be discovered in the future, but the general exposures of uranium minerals are of low grade, and an effective means of concentrating such ore to a profitable grade must be found, bearing in mind also the competition from other known sources of production.
>
> "The extent of the field, the number of uranium mineral exposures throughout it, and the former discovery of rich ore are, however, facts which fully warrant further development and exploration."

In 1934 the leases were abandoned and the field was deserted once more until the commencement of the present operations in July 1944. The early work had located only one small orebody. The inaccessibility of the district and the absence of high-grade ore limited exploratory activity considerably and discouraged prospectors so that a large area of uranium-bearing country still remained virtually unexplored when the British Government's request was received.

References -

(1) H.Y.L. Brown, "Occurrence of Uranium Ores and other Rare Minerals near Mount Painter", Special Bulletin 1911, The Occurrence of Uranium (radio-active) Ores in South Australia, out of print.

(2) L.Keith Ward, "The Mount Painter District", Mining Review 17, pp. 27-33, 1912.

(3) L.Keith Ward, "The Radium Extraction Company's Mine at Mount Painter", Mining Review 20, pp. 36-40, 1914.

(4) L.J. Winton, "The Radium and Rare Metals Industry", Mining Review 43, pp. 63-64, 1926.

(5) L.Keith Ward, "Supplementary Report on the Prospects of the Mount Painter Deposits of Radium-Uranium Ore". Mining Review 45, pp. 57-60, 1926.

(6) L.J. Winton, "The Australian Radium Corporation", Mining Review 50, pp. 63-75.

IV. PAST PRODUCTION.

Available records show that the field produced about 136 tons of selected uranium ore equivalent to approximately 3 tons of uranium trioxide. The output for the 1911-1914 period was sold overseas, whilst the 1924-34 production was treated at Dry Creek, Adelaide, for separation of the radium and uranium contents. The total quantity of radium recovered at this plant was 194 milligrams valued at £2,338 and 1,256 lb. of sodium uranate at £213. The total value of the ore produced since the discovery of the field is estimated at £10,000 which is chiefly the value of the radium content.

URANIUM-RADIUM PRODUCTION - MOUNT PAINTER DISTRICT.

Year	Ore Selected	Grade U308	Radium	Sodium uranate lb.	Value
	Tons	Per cent	mllgrms.		£. s. d.
Prior to 1913	7	2.0			
	2.25	2.02			
	0.25	25.0			
	0.25	25.0			
1913	25.0	2.60			
	20.0	3.24			
1914	0.25	25.0			
1925			7.01	508	172. 17. 0
1927			97.0	412	1,088. 0. 0
1932			72.0	336	1,050. 0. 0
1934			18.0		240. 0. 0
Total	136 tons	estimated			10,000. 0. 0

V. REVIEW OF EXPLORATORY OPERATIONS.

1. INTRODUCTION:

The plans for the examination of the Mount Painter field were formulated between 7th and 13th June, 1944, following an inspection of the No.6 Workings containing the largest known deposit. They were submitted to London on June 21st, 1944 and comprised the following tasks; first, road construction (July to November 1944); second, establishment of camps, mining machinery, and exploration of the most promising deposits (October to December 1944); and third, if results warranted, the erection of a mill to treat 100 tons of ore per day and further developmental work at the mines (January to June 1945), all to be done at a total estimated cost of £96,596.

ORIGINAL ESTIMATE

15th June, 1944.

MOUNT PAINTER PROJECT.

Schedule of estimated expenditure

1944.

	£.	£.
July to November inclusive		
Road construction	10,000	
October to December inclusive		
Establishment of camp and development of No.6 Workings and at outside prospects Labour £7,786, Materials £7,310....................................	15,096	
Plant at No.6 and outside prospects.........	7,237	
Buildings, Mt. Painter Depot and at Eastern Group....................................	3,970	
Subsidiary roads if required................	5,000	
Contingencies....................................	3,697	45,000
January to June inclusive		
Supply and erection of plant to treat 100 tons per day....................................	32,000	
Development at mines during erection of plant, per month, including cost of outside prospecting		
Labour £1,896		
Material 1,370		
6 months at £3,266	19,596	51,596
	TOTAL	£96,596

In this review it is intended to record the main features of the recent investigation which, it is believed, is unique in the history of mining exploration in Australia. From the outset the record of the operations of the early adventurous prospecting enterprises, so far from affording ground for hope, rather indicated that the task was out of all proportion to the results which were aimed at. Nevertheless as the initial geological examination of the lenticular brecciated, ironstone mass at the No.6 Workings, gave promise of the possible occurrence of ore, the task had to be done and done quickly if a vital war purpose was to be served. Full advantage was taken of a number of facilities in regard to men and materials which the country's organization for war work made available. For instance without the assistance of the Royal Australian Air Force in transport and photography and of various Government munition activities, it would have been impossible to complete the programme according to a speedy time schedule.

FIG. 2. Road Construction. Mt. Painter July 1944.

2. MOUNT PAINTER ROAD.

Whilst wartime regimentation simplified considerably the problem of executing a project of this character, the inhospitable terrain, difficult of access in the extreme, presented an ever formidable natural barrier. Rough camel pads existed but were quite useless for the transport of mining and prospecting supplies when time was a prime consideration. It was obvious, therefore, that nothing worthwhile could be accomplished without proper roads giving access to the main deposits and the organisation of strong gangs for road construction equipped with heavy mechanical plant was the first task.

Much time was saved by arranging the assembly of the necessary plant and manpower by anticipating approval from London for proceeding with the exploratory programme. By the time confirmation was received on 11th July, 1944, preliminary surveys and estimates had been completed and actual construction commenced at Balcanoona on the eastern flank of the Flinders Range on 18th July 1944. In 97 days the road was completed to the No.6 Workings and the transport of heavy supplies and mine machinery for exploratory work commenced.

3. FIRST GEOLOGICAL CAMP.

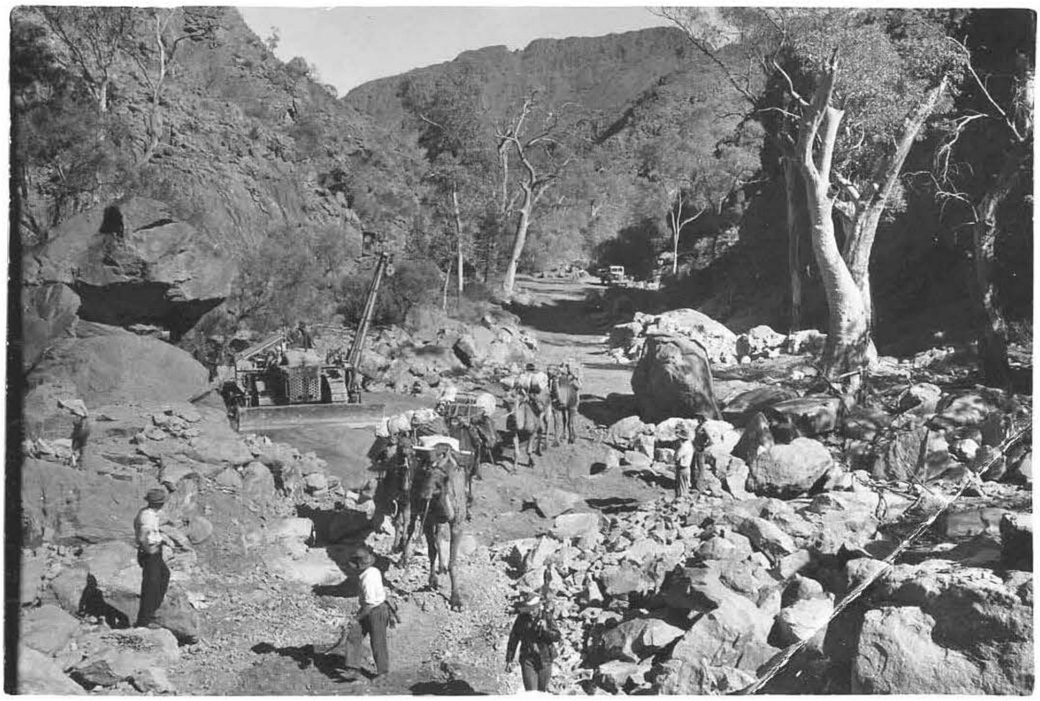


FIG. 3. Camel transport. Mt. Painter.

Even before the Mount Painter road was built a start had been made on the study of scientific problems both in the laboratory and in the field. The first geological survey camp at Mount Painter was established on 22nd June by the use of camels, which continued to bring in supplies until the road was completed and thereafter served isolated prospectors and subsidiary geological camps.

FIG. 4. Camel train. Mt. Painter
Geological Camp. June 1944.

Detailed Surveys were made of the old workings, the more promising ones were sampled and reconnaissance surveys were made in the search for new deposits with the idea of defining a precise programme of testing before the road was completed to enable mining operations to proceed without delay when the right-of-way was open. Geophysical surveys were also made of a few deposits with Geiger-Muller equipment which is responsive to emanations from uranium minerals.

To enable more extensive geological surveys and prospecting to be conducted aerial photographs were taken over about 100 sqaure miles.

The first geological survey camp was dismantled when the road reached Mount Painter on Sept. 23rd 1944.

4. BALCANOONA AERODROME.

During the course of the aerial surveys it became apparent that the aeroplane could provide a solution of the problem of getting emergency supplies and key personnel quickly from Adelaide to the mines. Furthermore it could bring medical assistance to the field in two to three hours, a service which was essential for a camp of 70 men, isolated and nearly 200 miles distant over rough tracks from the nearest resident medical officer.

Accordingly an aerodrome was built at Balcanoona, the nearest suitable area, 25 miles from Mount Painter and regular flights were arranged by the Royal Australian Air Force.

5. MINING OPERATIONS AT NO.6 WORKINGS.

The period during which road construction was proceeding, provided valuable opportunity for the thoughtful planning of mining operations, the assembly of motor transports and equipment at Copley, the nearest railway station, and the securing of experienced miners, prospectors and skilled workers.

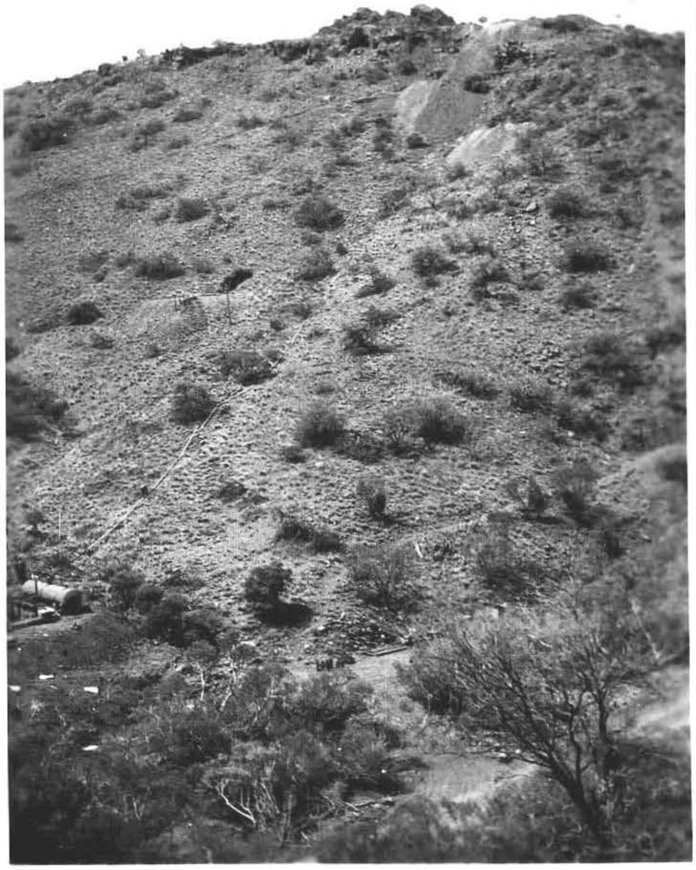


FIG. 5 & 6. Installation air compressors and air line. No.6 Workings. Mount Painter.

With the opening of the road on September 23rd, 1944, the erection of the camp and the preparation of the No.6 Workings were begun and by 31st October, 1944, the preparatory work had progressed sufficiently to permit the commencement of underground driving, sinking and diamond drilling. The geological conditions uncovered from day to day largely determined the course of this work. However, by the end of December the results of operations were rather disappointing. As last possibilities, from January 1st to May 1945, diamond drilling in search of primary minerals in depth and driving for new ore-bodies in the ironstone near the surface were carried out with negative results.

6. SECOND GEOLOGICAL CAMP.

Whilst the search for additional ore at No.6 Workings was progressing, a geological survey camp was established at East Painter where, during earlier reconnaissance work, numerous surface indications were found which gave promise of considerable amounts of disseminated uranium minerals. These occurrences were important enough to warrant testing and, as in the case of Mount Painter, to define a programme of operations detailed geological and geophysical surveys were made, while a second access road was being built for the transport of mining plant. The camp was serviced by camels until the road was completed. From this camp also, geological parties made a regional map of the uranium-bearing belt of rocks, and they utilised aerial photographs for the recording of observations. At the same time ground control surveys were made for accurate map-making at a later date. Geophysical surveys and a limited amount of prospecting work were also undertaken during this camp period.

7. EAST PAINTER ROAD.

As soon as the Mount Painter road was completed, work commenced on the road to East Painter and proved an equally arduous task. Although the distance is only about three miles in a direct line from the No. 6 Workings, approach from that direction was impossible. Access by road was only practicable from the eastern side of the Flinders Range along the East Painter Creek which in places is bounded by almost vertical walls of solid rock 400 to 500 feet in height above the creek bed.

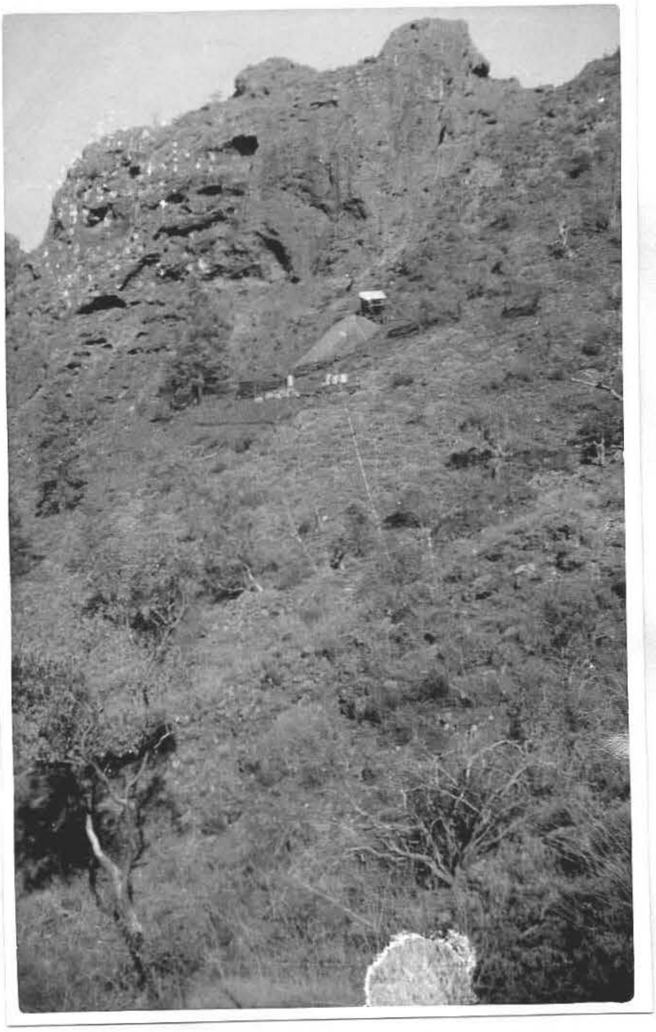
FIG. 7. East Painter Road. December 1944.

In spite of the formidable hazards this second road totalling 14½ miles including 4½ miles through the gorge to East Painter camp and a further 2 miles up the gorge towards the "Smiler Greenwood" prospect, was completed in 68 working days from October 1st, 1944 to December 19th, 1944.

8. MINING OPERATIONS AT EAST PAINTER.

By the time a mining camp had been established centrally to the several East Painter prospects and plant had been transported, the futility of the search at No. 6 Workings became obvious and interest centred now on the East Painter area as being that from which some compensation for the efforts expended was most likely to be gained.

When advice was received from London on 22nd February, 1945, requesting the completion of the exploratory programme and the termination of operations, sufficient work had been done at most prospects to enable the potentialities of the deposits to be assessed.



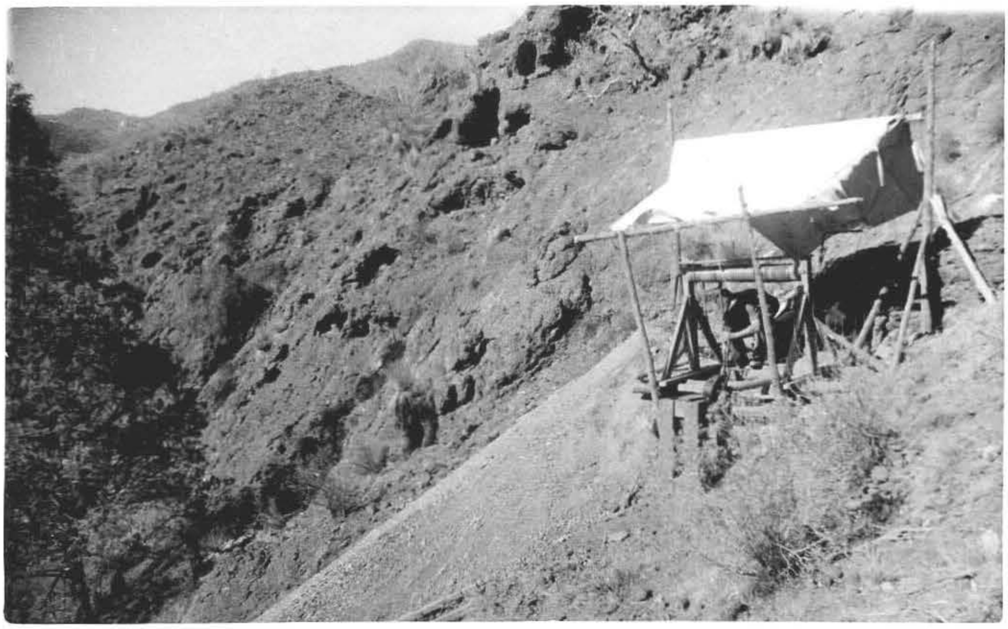


FIG. 8 & 9. No.5 Workings. East Painter.

VI. EXPENDITURE.

The actual expenditure as at the 30th September, 1945, amounted to £62,722. 0. 6.

It is not possible yet to state the precise figure as to the net cost of the project, but it is unlikely that the amount involved will exceed £65,000.

VII. SUMMARY OF RESULTS.

The investigation showed fairly conclusively that both the already known uranium deposits and those newly discovered were small and contained extremely "low grade" material, the uranium content of which rarely exceeded 0.25% uranium trioxide or about 5 lbs. of uranium per ton. This was then fixed as the lowest target to be aimed at as a guide for the mining operations and with a view to subsequent treatment.

The total tonnage of ore so far available in the workings at the several known prospects does not exceed 500 tons containing 0.33% uranium trioxide. An overall recovery of 60% being assumed this would yield on treatment 2,000 lbs. of uranium.

The results of ore-dressing and leaching investigations show that the "uraniferous minerals" of the minimum grade ore fixed can be satisfactorily concentrated by either mechanical or chemical processes for the production of uranium, either as uranous oxalate or fluoride, both of which can be readily transformed into uranium trioxide or uranium metal, the forms in which, when purified, uranium has been used for atomic research and as raw material for the manufacture of atomic bombs. These investigations were conducted only on a laboratory scale, and although being sufficiently informative to enable a treatment plant to be commissioned, had circumstances required it, it would be desirable to supplement them by further research by means of a pilot plant. In these investigations the chemical research work was not carried beyond the stage involving the production of uranous oxalate or fluoride, the stage at which the beneficiation of the ore may be regarded as ending and research on utilization as conveniently commencing.

The results of geological and geophysical surveys generally indicate that the field can only be regarded as a source of relatively small amounts of uranium from "low grade" and near-surface secondary ore and from secondary ore of higher grade occurring in small pockets, similar to that already exploited in the No.6 Workings. Furthermore, it is the general consensus of opinion that there is no great probability of finding primary uranium minerals at depth. The two most commonly-occurring uranium minerals are torbernite and autunite.

Though the work on the No.6 deposit at Mount Painter had disappointing results, the geological surveys led to the discovery of new occurrences at East Painter and it is reasonable to expect further discoveries of a similar character from a continuation of such work.

The Geophysical Surveys were of a far more limited scope than the geological and did not lead to any new discoveries. They were chiefly experimental and much time was devoted by the geophysicists to studies of the application of geophysical methods for assay purposes, but during the course of these studies much valuable information, having a bearing on the nature and age of the uranium minerals, was gained.

The record of the scientific knowledge acquired during the investigations and presented in this Bulletin will be found to be incomplete on a number of important aspects. In particular, the problem of the origin of the deposits cannot be satisfactorily solved before adequate underground exposures are available and further experience is obtained. In the meantime one is obliged to make intelligent guesses in estimating the possibilities and it is on such a basis that the following conclusion is reached in answer to the question which has been asked by the United Kingdom authorities.

VIII. CONCLUSION.

To sum up,

The Mount Painter area may be expected to yield over a period of years from numerous small shallow deposits an annual production of ore containing the equivalent of about one ton of uranium trioxide, the question of cost being problematical.

The field offers scope only for the activities of small working parties and prospectors.

DATED: October 30th, 1945.

PART II: CONSTRUCTION AND MINING OPERATIONS

BY

E. B. JENSEN

INSPECTOR OF MINES AND ENGINEER-IN-CHARGE,
DEPARTMENT OF MINES,
ADELAIDE.

1. PLANS

The following plans accompany this report:--

1. Map of the locality showing roads constructed to the Mt. Painter camp and to the East Painter camp. Plan No. 3170. Sheet No. 23.

2. Plan showing surface layout Mt. Painter Camp. Plan No. 3174. Sheet No. 24.

3. Plan showing surface layout East Painter Camp. Plan No. 3194. Sheet No. 25.

4. Plan showing layout of Balcanoona Aerodrome. Plan No. 3208. Sheet No. 42.

5. Plan of cubicles specially designed for camel transport. Plan No. 3206. Sheet No. 26.

6. Plan of Men's sleeping quarters designed for fabrication in sections for easy transport and erection. Plan No. 3207. Sheet No. 27.

7. Plan of sledge for hauling materials up the hill-side. Plan No. 3200. Sheet No. 28.

8. Diagram showing layout of holes for drag cut in driving. Plan No. 3198. Sheet No. 29.

9. Diagram showing layout of holes for post hole cut in driving. Plan No. 3199. Sheet No. 30.

10. Plan showing location of bore holes drilled for water at Mount Painter and East Painter. Plan No. 3209. Sheet No. 41.

2. ACCESS ROADS TO DEPOSITS.

The location of the Mount Painter disctrict is shown on the map (Plan No. Sheet No.) which accompanies this report. The whole of the locality of which Mount Painter may be taken as the centre is extremely rough and access to the mining areas presented an unusually difficult problem particularly in view of the fact that it was not only necessary to construct road approaches to the proposed camp sites but also to complete all such work rapidly in order that something definite might be known of the possibilities of the field by the end of the year 1944. The time factor was considered to be of vital importance.

The combined local knowledge of the Director of Mines, and the owners of Mount Serle Station and Arkaroola Station suggested that the most suitable entry into the Mt. Painter and East Painter mining areas under all the governing circumstances would be by means of road approaches constructed from the existing bush track extending from the Copley Railway Station to Balcanoona and beyond Wooltana. This track was in use for the weekly mail to outlying stations and was in fair condition. It is shown on the plan of the roads (Plan No. 3170 Sheet No. 23).

A preliminary reconnaissance of this route was made in the first week of June 1944 by a party consisting of the Director of Mines, the writer, the Assistant Government Geologist, the owner of Mount Serle Station and the owner of Arkaroola Station. The party proceeded on foot up the rocky Radium Creek Gorge and examined the locality of the old abandoned mines in the Mt. Painter area. The data collected during this visit made it possible to prepare preliminary estimates of the cost of constructing a track suitable for motor vehicles to carry in the plant and machinery required for mining operations.

Fig. 1 and 2 - Radium Creek Gorge, before road construction, June, 1944.

In order to obtain a check on these estimates another inspection was made between 18/6/45 and 23/6/45 by a party consisting of the Director of Mines, the writer, an Engineer of the Engineering and Water Supply Department and his local Inspector and also an Inspector loaned by the Highway's Commissioner. During the trip the Highways Commissioner's Inspector mentioned that he had never seen a road constructed in country as rough as that in the gorge between Echo Camp and the site of the proposed mining camp.

The estimates prepared by the Engineering and Water Supply Department substantially confirmed the preliminary figures prepared in this office.

Confirmation of instructions to proceed with the construction of the roads was received on 5/7/44. The first road gang moved out from Copley on 11/7/44 to fill in the runnels on the unmade bush track between Balcanoona and Arkaroola and to remove the worst of the boulders and growth between Arkaroola and Bore Hole Camp. When this portion had been made safe for very slow traffic, road-making machinery of a heavy type was brought in and a camp was established at Bore Hole camp on 17/7/44.

From this centre the heavy construction work was pushed very rapidly to Echo Camp and then through the gorge up Radium Creek to the mines. The road head reached the mines on 23/9/44, thus establishing record speed of construction of 29 miles of road in that country.

Fig. 3 and 4 - Radium Creek Gorge, Mt. Painter Road in course of Construction, August, 1944.

At the time the track to Mount Painter was approaching completion, instructions were received to proceed with the construction of a road to give access to the East Painter Field. As this field is only 4 air miles distant from the Mount Painter Field a Licensed Surveyor of the Lands and Survey Department, who was then in the district, was requested to seek a route from Mt. Painter to East

Painter. He reported against the proposal on the ground of cost and time. A similar report was received from the Inspector in charge of the road construction gangs. It was therefore decided to approach East Painter by constructing a track from the mail road north of Wooltana Station across country to the mouth of the gorge and up the gorge to the old camp site known as Greenwood's camp (See Plan No. 3170 Sheet No. 23).

Road making in this locality, (East Painter), commenced on 1/10/44 and was completed by 19/12/44 with the exception of some maintenance surfacing of the rougher portions. The road to East Painter 19½ miles long was completed in 68 working days. This included 4½ miles of road making in the extremely rough country in the gorge leading to Greenwood's Camp, the site chosen for and now known as the East Painter Camp.

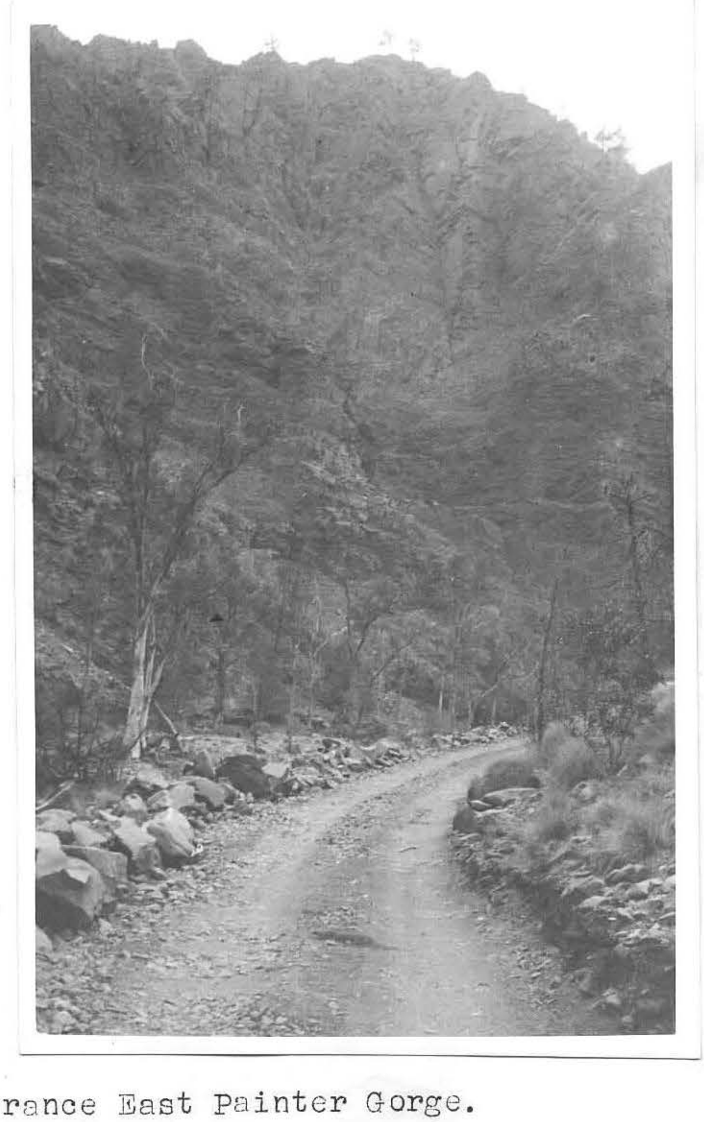


Fig. 5 - Entrance East Painter Gorge.

Fig. 6 and 7 - East Painter Gorge, Road to East Painter Camp.

The following schedule covers the main points of interest:

Road to Mount Painter Camp commenced 11/7/44.
Completed except for maintenance 23/9/44.
Road to East Painter commenced 1/10/44.
Completed except for maintenance 19/12/44.
Average number of men employed, 37.

<u>Plant employed on Road Construction.</u>

1 Bedford utility truck ½ ton.
1 Chevrolet " " ½ "
1 Bedford truck 30 cwt.
1 " " 2 tons
1 Dodge truck (loaned) 3 tons
1 60 H.P. Auto patrol grader
1 R.D.A. Caterpillar tractor.
1 Drawn road grader 7000 lbs.
1 Drawn road roller 4 tons.
1 R.D. 6 Tractor Crane with bulldozer.
1 Portable Broomwade air compressor complete with jackhammers and steel.
1 34' x 17' mess room and kitchen-tents as required.

<u>The costs were:</u>

Construction of road from Balcanoona to Mt. Painter (29 miles)	£4,301	18	8
Construction of road from the turn-off near Wooltana to East Painter (19½ miles)	5,633	8	6
Expenditure on maintenance surfacing of Mt. Painter road	850	0	0
Expenditure on maintenance surfacing of East Painter road	20	0	0
Total expenditure on road making	10,805	7	2

These costs include an amount of £878 for special repairs to heavy equipment damaged owing to the unusually heavy nature of the work in the Mount Painter and East Painter gorges.

The estimated cost of constructing the road, for traffic at 15 miles per hour to Mount Painter together with subsidiary roads to the mine openings was £15,000. This did not include an allowance for constructing a road to East Painter the necessity for which developed later.

The saving in the cost of the work as carried out resulted from:

(a) Omitting the subsidiary roads to the mine openings and substituting an air-hauled sledge to haul the mining plant and stores up the hill-side at No. 6 Workings at Mount Painter.

(b) Omitting for the time being surfacing and regrading of the track for high speeds.

The average safe speeds for loaded motor vehicles over the roads as constructed were:

Between Balcanoona and Arkaroola average	12	miles	per	hour.
" Arkaroola and Echo Camp "	8	"	"	"
" Echo Camp and Mt. Painter Camp	6	"	"	"
From Wooltana turn off to mouth of the East Painter Gorge	12	"	"	"
Through the East Painter Gorge	6	"	"	"

3. AERODROME AT BALCANOONA.

In the interests of quick access to the mines for inspection purposes and as a precautionary measure in case of serious accident or sickness, it was deemed advisable to make provision for landing aeroplanes of the Avro Anson or Oxford type at the nearest suitable point. The Balcanoona site was chosen by the Inspector of Civil Aviation Aerodromes because of its location fairly centrally to the Mount Painter and East Painter camps and its proximity to the Balcanoona Station which is connected to the telephone circuit from Copley and the mining camps.

Fig. 8 - Wooltana Sheep Station, near Balcanoona, on Eastern edge of Flinders Range about 25 miles from Mt. Painter.

The Balcanoona landing ground is situated about two miles east of Balcanoona Station the location of which is shown on the road map (Plan No. 3170 Sheet No. 23) attached to this report. The Flying Doctor at Broken Hill was furnished with a map showing the site of the drome. Supplies of aviation spirit were lodged at Balcanoona Station for emergency use.

One emergency landing was made by an officer of the R.A.A.F. flying from Broken Hill to Mt. Eba on 18/12/44. He was forced down because of engine trouble and remained at Balcanoona until relief reached him.

The flying time from Adelaide to the Balcanoona drome is only three hours including time lost in refuelling at Port Pirie. A visit from Adelaide to the Mount Painter Camp per railway to Copley and thence per motor truck occupied three days to get there and three days to get back owing to the slow railway schedule and the poor condition of the local roads. As this involved too much lost time practically all inspection trips were made per aeroplane after the landing ground had been completed.

Fig. 9 - Balcanoona Aerodrome, November, 1944.

The construction of the drome was commenced on 11/10/44 and completed on 7/11/44 at a cost of £324/9/11. The first plane landed in the first week of November 1944. Subsequently additional expenditure was incurred in improving the ground markings to give better visibility from the air, bringing the total cost to £459/12/9 including the cost of a small shelter shed. The work involved clearing and levelling and preparing the surface for four runways each 1200 yds. long by 50 yds. wide, with additional clearing 67 yds. wide on each side of each runway (Plan No. 3208 Sheet No. 42). The runways were not surfaced with special material and they are not suitable for heavy traffic but they answered the purpose during the progress of the work at Mt. Painter and East Painter. The landing ground was damaged by heavy rains in January and July 1945. The cost of repairs was £7/17/2.

The flights to Balcanoona were serviced by the R.A.A.F. Trainees Section established at Mallala, 40 miles north of Adelaide and the trips were regarded as training flights for pilots, navigators, and radio operators. All flights were arranged with the approval of the Department of the Minister for Air, Melbourne.

4. TELEPHONE COMMUNICATION.

For the control of operations from the office of the Director of Mines, Adelaide and from the point of view of the safety of the mine workers, it was considered necessary to bring the isolated mining camps into telephonic communication with the main telephone system in the State. The only telephone lines in existence east of Copley were privately owned single wire bush telephone lines erected and owned by the various stations in the district. This private system extended east from Copley to Mt. Serle where it bifurcated, one branch going to Balcanoona and Moolawatana and the other branch to Umberatana and Arkaroola (vide the road plant No. 3170 Sheet No. 23).

Permission was courteously given by the station owners to join on to this private system at the point where the track to Mount Painter crosses the bush telephone line in the Wywyana Creek. Arrangements were made to carry a telephone line from that point to the Mount Painter camp a distance of 7½ miles. At a later date the telephone was extended across country from the Mt. Painter camp to the East Painter camp a distance of about 4½ miles.

The bush telephone lines in the locality were in poor condition at the time of commencement of operations at Mount Painter. It was necessary to rehabilitate the system to make communication possible. The Superintendent of Telephones of the Post Master General's Department was most helpful in making available his expert country telephone linesmen and in supplying telephones and equipment not obtainable elsewhere. The cost of the work carried out by the Post Master General's Department was £166/17/7 excluding telephones, but including the provision of a separate line from the Copley Post Office to the Mines Department depot in the Copley Station yard and the complete overhaul of the private lines between Copley and Mt. Painter and between Mt. Serle and Balcanoona. Telephones and accessories cost £47/3/7.

The materials for the extension of the line from the Wywyana junction to Mt. Painter and East Painter were obtained from the Army at a cost of £65/16/8. The cost of repairs etc. at the mine end was £24/2/9. The total cost of establishing telephone communication to the mining camps and maintaining it was, £310/5/-.

At the request of the Premier of South Australia, the private owners readily agreed to place their bush telephone lines at the disposal of the project. It was necessary to establish a switching point at Mount Serle Station and the courtesy and assistance received from Mr. and Mrs. Greenwood at that station are gratefully acknowledged.

5. WATER SUPPLIES.

a. Water Supply at Mt. Painter Camp.

The supply of water for the camps and for mining operations was one of the problems demanding attention at the outset of the project.

About 30 years ago a well (shown on Plan No. 3174 Sheet No. 24) was sunk at the site of the Mount Painter Camp. The supply from this source was small. Fortunately it sufficed for the purposes of the preliminary geological camp, but proved inadequate for the requirements of the construction gang. Moreover as the water level in this well got low, sickness developed in the camp and the drawing of water from the well was discontinued.

An analysis of the water in the well showed the total saline contents to be 80.19 grains per gallon and the total hardness 44.15 English degrees. Full details of the composition of this water are given in Table

The search for water was regarded as urgent and before the road was properly completed, the first plant transported to Mt. Painter was water-boring equipment. The No. 1 bore hole, bored with a percussion plant was located 15 chains downstream from the No. 6 workings at the junction of Mount Gee and Radium Creek. Boring commenced on 15/9/44. The hole proved disappointing and was discontinued on 2/11/44 at a depth of 122 feet. Plan No. 3209 Sheet No. 41 shows the location of the boreholes at Mt. Painter and also at East Painter.

To speed up the search for water a second percussion boring plant was forwarded to the district and commenced boring No. 2 hole on 30/10/44. This hole was located at the junction of Mt. Painter Creek and Radium Creek about 60 chains below No. 6 workings. No water was met with and as the indications were not promising, the hole was discontinued at 81 feet on 11/11/44.

While No. 2 hole was being bored the first percussion plant was shifted upstream to the site of No. 3 hole which was located in the camp area 400 feet downstream from the mess house. Boring commenced on 6/11/44 and the result was promising. Water was struck at 34 feet. The hole was taken to 80 feet and discontinued on 21/11/44.

This bore was cased with 7' of 6" casing and 52' 6" of 5" casing. To obviate engine pumping, a windmill 8' dia. by 20' high was erected with the delivery connected to a nest of four 400 gallon tanks with a 1½" reticulation to the mess kitchen, shower room and stand pipe.

For about four months this bore provided sufficient water for the camp requirements (excluding mining requirements) i.e. a maximum of about 400 gallons per day of 24 hours. The supply gradually fell off until in June 1945 the bore was barely able to provide 200 gallons per day of 24 hours.

An analysis of this water from a sample collected on 19/11/44 showed the total salinity to be 77.58 grains per gallon and the total hardness to be 38.91 English degrees. Details are given in Table 1.

To provide for the requirements of the rock drills and the under-ground diamond drill and also the miners' change house (a total of about 600 gallons per day when all the faces were working) bore hole No. 4 was commenced on 15/11/44 at a point near the engine house. This bore encountered a hard bar at 15 feet. It was discontinued and the plant shifted further upstream to a new location 180 feet from the engine house where a hole known as No. 4A was commenced on 20/11/44. Water was found at 40 feet. The bore was completed at 102 feet on 30/11/44. A small engine and pump were erected on the bore and water pumped as required to the nest of two 400 gallon tanks which acted as the sump for the mine high pressure pump in the engine house.

On 3/12/44 all water-boring operations ceased at Mount Painter.

b. Water Supply at East Painter Camp.

At East Painter camp the old well was dry and no water was available at the time of the establishment of the preliminary geological camp in August 1944. All water required at the camp had to be carried in by camels from the Trig bore, about 9 miles away. This arrangement continued until the road reached the camp on 19/12/44. After that date water was carted per motor lorry from the Trig bore until that supply became unusable in December 1944. Water was then carted from Wooltana station a distance of about 22 miles from the camp until the first week in February 1945 when water became available in No. 2 bore near the mouth of the gorge and distant about 4 miles from the camp.

The first bore, East Painter No. 1, was located about 800 feet upstream from the East Painter camp site. It was commenced on 30/11/44 and discontinued on 12/12/44 at a depth of 98'. No water was met with.

The next bore, East Painter No. 2, was located near the mouth of the gorge. It was commenced on 6/12/44 and discontinued on 2/2/45 at a depth of 171 feet. Water was struck at 120 feet. The water was of good quality for that locality and the bore supplied the camp until the cessation of operation.

An analysis of the water from East Painter No. 2 bore hole gave a total salinity of 46.82 grains per gallon and a total hardness of 23.68 English degrees. Details are given in Table 1.

It is not likely that a large supply of water would be available from this No. 2 bore hole. In the light of the experience gained at Mt. Painter the chances are that the underground supplies are limited and it would be unsound to rely on a continued draw of more than 200 or 300 gallons per day over a few months.

During the sinking of No. 2 bore hole at the mouth of the gorge it was decided to locate No. 3 hole about 100 yards downstream from the East Painter camp. This bore was commenced on 18/12/44 and was discontinued at a depth of 160 feet on 19/2/45. No water was met wit

To make these notes complete it is recorded that an attempt was made to use water from the Buxton Bore which is situated about 3½ miles east of the Lady Buxton Mine and 7½ miles from the East Painter Camp.

It was found that this water had rather an offensive odour and a disagreeable taste. It was only used for mining and slushing purposes. An analysis of the Buxton Bore water gave a total salinity of 183.33 grains per gallon and a total hardness of 38.90 English degrees. Details are given in Table 1.

<u>c. Costs of Water Supplies.</u> The following data and costs in connection with water supply at Mount Painter and East Painter are recorded.

(i) <u>Cost of Bore Holes</u>

The attached schedule, No. 1, gives the cost of all bore holes.

(ii) <u>Cost of Carting Water</u>

Until bore holes Nos. 3 and 4A at Mt. Painter came into production it was necessary to cart a good deal of water for mining and mess purposes. The cost of this carting was as follows:-

COST OF CARTING WATER PER MOTOR LORRY AT MOUNT PAINTER

From	Distance from Mt. Painter	Gallons Carted	Total Cost	Cost per 1000 galls. including 9d. per mile motor lorry costs.
Borehole Camp	6 miles	17,000	£42 13 7	£2 10 2
Arkaroola Station	12 "	1,500	5 13 6	3 15 8
Wooltana Station	28 "	5,000	34 6 5	6 17 3

At East Painter, until water was located in Bore Hole No. 2, all water used was carted either by camels or by motor lorry. The costs were:-

COST OF CARTING WATER PER MOTOR LORRY AT EAST PAINTER.

From	Distance from East Painter	Gallons Carted	Total Cost	Cost per 1000 galls. including 9d. per mile motor lorry costs.
Trig Bore	9 miles	7,000	£27 10 9	£4 2 3
Buxton Bore	8 "	2,000	6 1 10	3 0 11
Wooltana	22 "	15,000	76 11 8	5 2 1½

COST OF CARTING WATER PER CAMELS AT EAST PAINTER

From	Distance From East Painter	Gallons Carted	Total Cost	Cost per 100 galls. including hire of camels
Trig Bore	9 miles	2,500	£56 3 0	£2 4 11
Road Camp	4½ "	900	15 10 0	1 4 5
From Camp to Similar Lease	2½ "	250	4 3 4	1 13 4

Note the hire cost of camels was 30/- per day for a string of 6 camels. The camels were hired from the owner of Arkaroola Station. They were handled by a camel driver released from the Navy on behalf of the Project.

SCHEDULE NO. I

SCHEDULE OF BORE HOLES.

MOUNT PAINTER.

No. of Hole	Commenced	Completed	Depth in feet	Struck Water at Feet	Labour			Camp Allowance			Materials			Holiday Pay			Hire of Plant			Total Cost			Cost per foot
					£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d	
1	24/9/44	2/11/44	122	Dry	100	10	10	20	17	0	23	18	11	11	0	11	12	5	0	168	12	8	27/8
2	27/10/44	11/11/44	85	Dry	40	2	3	10	18	6	8	19	1	7	13	11	8	11	0	76	4	9	17/11
3	3/11/44	26/11/44	80	34	55	4	4	14	0	0	16	7	9	7	4	10	8	0	0	100	16	11	25/2
4	12/11/44	16/11/44	15	Dry	10	1	3	2	11	5	3	6	9	1	7	2	1	10	0	18	16	7	25/1
4A	17/11/44	2/12/44	102	60	38	13	2	10	5	9	11	8	2	9	4	8	10	5	0	79	16	9	15/8

EAST PAINTER

1	27/11/44	12/12/44	98	Dry	32	15	2	7	1	5	11	10	10	8	17	5	9	17	0	70	1	10	14/3
2	3/12/44	7/2/45	171	120	86	3	9	25	1	5	28	2	3	15	9	7	17	3	0	172	0	0	20/1
3	13/12/44	28/2/45	160	Dry	115	5	7	32	15	9	36	12	3	14	9	8	16	1	0	215	4	3	27/-

Casing left in Bore Holes.

Mt.	Painter	Hole	No.	3	52' 6"	of	5"	Casing
"	"	"	"	4A	12' 2"	of	5"	"
East	"	"	"	2	13' 6"	of	6"	"
Mt.	"	"	"	3	7' 0"	of	6"	"

(iii) Cost of Pumping Water at Mount Painter Camp.

On the completion of No. 3 bore near the mess house at Mt. Painter, the hole was equipped with a wind-mill and nest of storage tanks with a capacity of 1600 gallons. All the water required for camp and shower room gravitated through a reticulation to the points where it was needed. A total of about 60,000 gallons was pumped by the wind-mill. No running costs were incurred other than about £5 for repairs to the wind-mill damaged by storms.

On the completion of No. 4A bore near the engine house it was fitted with a 3" bore hole pump operated by a 3 H.P. Ruston Hornsby petrol engine. All water for mining operations and for the miner's change house was then obtained from this bore. The maximum draw on this borehole was about 600 gallons per day for about one month. Thereafter requirements were less and the draw on the bore was reduced gradually to 250 gallons per day toward the close of operations.

All the water required for the underground rock drills and for the underground diamond drill was lifted from the sump in the engine house to tanks near the top of the hill at No. 6 workings by means of an Ajax 3" high pressure bulldozer pump on one bed plate with a 3 H.P. Norwood Bagshaw O.C.V. petrol engine. The rock drills required about 200 gallons of water per shift. The supply to the diamond drill varied according to the nature of the ground penetrated, the maximum requirement was 400 gallons per shift. The cost of this pumping was 2/6 per thousand gallons.

d. Additional Sources of Water

Any future mining and milling operations in the area covered by this report will be faced with a water problem, as the underground supplies located to date are inadequate for mining and milling.

About three miles downstream from Echo Camp there is a permanent water hole in the Arkaroola Creek which when visited by the writer in a dry period in November, 1944, had a supply estimated at half a million gallons. The high water marks indicate that when full this water hole would have a capacity of about 2 million gallons.

Another locality where water might be obtained in some quantity is at Paralana Hot Springs. The Director of Mines visited this area with the writer in December, 1944.

An analysis of the water from Paralana Hot Springs made by the S.A. School of Mines on September 3rd, 1913, is given in Table 1.

LOCALITY	MT. PAINTER WELL 8/6/44	MT. PAINTER - MAIN CAMP WELL 4/11/44	MT. PAINTER - MAIN CAMP WELL 13/11/44	MT. PAINTER - NO. 3 BORE 22/11/44	BORE NEAR ECHO CAMP, ARKAROOLA. 8/6/44	EAST PAINTER - NO. 2 BORE. 14/2/45	ARKAROOLA WELL 8/6/44	MUNYALLINA WELL 7/11/44	MUDLAMUTANA WELL 22/11/44	BUXTON BORE 5/2/45	WOOLTANA H.S. WELL 7/11/44	WERTALOONA H.S. WELL 26/7/44	PARALANA HOT SPRINGS 3/9/43	BALCANOONA STN. NO. 7 BORE 31/5/45	BALCANOONA STN. [illegible]
TOTAL SOLIDS IN SOLUTION. Grains per Gallon.													(2)		
Chlorine, Cl.	40.67	22.06	21.85	21.56	45.49	9.65	37.43	41.02	21.43	86.72	16.00	42.88	22.50	22.23	14.51
Sulphuric acid (radicle), SO_4	16.29	8.44	8.39	7.49	25.64	4.16	64.65	31.32	19.75	16.92	10.70	24.08	10.56	11.48	7.57
Carbonic acid (radicle), CO_3	23.53	21.16	21.30	20.70	22.54	16.38	19.53	32.40	14.40	12.02	15.00	16.26	10.35	19.83	22.09
Nitric acid (radicle), NO_3	Trace	Present	Present	Present	Present	Present	Present	Nil	Present	Nil	Nil	Present		Trace	Nil
Sodium Na	27.94	14.29	13.60	15.54	30.79	9.92	30.78	37.60	15.30	55.69	12.31	26.07	21.53	12.96	12.45
Calcium Ca	7.72	8.93	9.07	7.22	7.86	2.43	7.72	6.15	3.14	6.43	2.64	10.94	3.82	7.72	3.86
Magnesium. Mg.	8.15	5.31	5.53	5.07	10.18	4.28	16.16	11.51	8.19	5.55	6.16	6.97	1.39	7.03	6.92
Total saline matter, grains per gallon	124.30	80.19	79.74	77.58	142.50	46.82	176.27	160.00	82.21	183.33	62.81	127.20	77.72	81.25	67.40
Total saline matter, ounces per gallon	.28	.18	0.18	0.18	.33	0.11	.40	0.37	0.19	0.42	0.14	.29		0.19	0.15
ASSUMED COMPOSITION OF SALTS												(3)	(1)		
Calcium carbonate	19.28	22.30	22.68	18.05	19.63	6.07	19.28	15.38	7.85	16.06	6.60	27.12	9.55	19.28	9.64
Magnesium carbonate	16.82	10.95	10.80	13.87	15.14	14.84	11.20	32.56	13.62	3.36	15.51		4.86	11.62	22.92
Magnesium sulphate	16.33	10.58	10.51	5.30	28.76		64.01	10.49	21.09	21.20	8.36	29.94		14.39	1.53
Magnesium chloride		.04	1.18							1.17		3.61		3.02	
Sodium carbonate						3.85							2.03		
Sodium sulphate	4.82			4.82	3.98	6.15	20.07	33.94	4.32		5.96		15.61		9.39
Sodium chloride	67.05	36.32	34.57	35.54	74.99	15.91	61.71	67.63	35.33	141.54	26.38	66.26	32.67	32.94	23.92
Sodium nitrate	Trace	Present	Present		Present	Present	Present				Nil	Present		Trace	Nil
HARDNESS - Total	52.82	44.15	45.44	38.91	61.52	23.68	85.78	62.74	41.55	38.90	31.94	56.01	15.33	48.21	38.12
" Temporary	39.24	35.30	35.48	34.51	37.61	23.68	32.57	54.02	24.02	20.05	24.99	27.12	15.33	33.07	36.84
" Permanent	13.58	8.85	9.96	4.40	23.91	nil	53.21	8.72	17.53	18.85	6.95	28.89	-	15.14	1.28
" due to calcium	19.28	22.30	22.68	18.05	19.63	6.07	19.28	15.38	7.85	16.06	6.60	27.32	9.55	19.28	9.64
" " " magnesium	33.54	21.85	22.76	20.86	41.89	17.61	66.50	47.36	33.70	22.84	25.34	28.69	5.78	28.93	28.48

(1) Paralana Hot Spring - Assumed Composition of Salts:- (additional)
Potassium Chloride 5.61 Sodium Silicate 7.32
Aluminium Silicate 0.07

(2) Paralana Hot Spring - Total solids in solution:- (additional)
Silicic Acid, Radicle (SiO_3) 4.61 Potassium (K) 2.94 Iron (Fe) (less than 0.01) Aluminium (Al) 0.02

(3) Wertaloona H.S. Well - Assumed Composition of Salts:- (Additional) Calcium sulphate .27.

6. ELECTRIC LIGHTING.

In order to light up the mine for afternoon shift work and to provide illumination for the men's recreation room and for each cubicle, a small electric lighting unit was installed in the engine house with leads to the mine and to the various buildings. The Department of Supply and Shipping in Melbourne supplied to the job a 5 KVA 3-ph. 415/240 V.A.C. petrol driven air cooled generator set.

Owing to war conditions difficulty was experienced in obtaining a suitable electrician to install the wiring. The South Australian Railways were able to make available an experienced electrical mechanic who handled the job rapidly and efficiently.

The generator set was erected in the engine house and leads were taken from there up the hill-side to the magazine at the 185 ft. level and to the necessary points at the 100 ft., 50 ft., and 25 ft., levels and also along the road to the camp and to the various buildings in the camp.

The number of points at which lights were installed was 63. The total cost excluding the prime cost of the generator set (value £300) was £501/3/6. The chief items being cables and wire £269/14/10. Lamps £9/13/9, poles £46/3/11, labour erecting and wiring £125/1/11. On the completion of the wiring the electrical mechanic was not further required. The running costs were then 13/8 per day.

The provision of a good light in each cubicle did much to improve living conditions at the camp. It was found that the men made comparatively little use of the recreation portion of the large mess room in the evenings. They preferred to foregather in small groups in their own cubicles.

7. SANITATION AND HYGIENE.

Every effort was made from the commencement of operations to protect the health of the men. A carefully designed six-hole fly proof latrine was erected at a cost of £41/12/7. This received attention daily from an employee deputed to wash the seat with hot soapy water and to use suitable disinfecting solutions.

To minimise the fly nuisance the mess room and every cubicle in the sleeping quarters was supplied with a sprayer and anti-fly comppund. All meat was placed in the refrigerator as soon as received at the mine, and cut up as required by the cook in a special meat room protected on four sides with fly wire. All refuse from the mess room was incinerated in a constantly burning home made incinerator.

In the early stages of the project all the water required for the small camp was obtained from the old well near the mess room. With the advent of the construction gang the well was unable to cope with the draw. Sickness developed rapidly in the camp and practically every member was affected.

In order to carry on, it was necessary to abandon the well for the time being and to seek another source of safe water until the boring operations located fresh supplies nearer the camp.

The health of the men responded immediately to the new water and from that time onward there was no recurrence of sickness of the nature of diarrhoea or dysentery. Acting on the advice of Dr. Ponsford, Commonwealth Medical Officer in Adelaide, all drinking and cooking water was chlorinated and a good stock of medical requisites recommended by him was established at the camp. First aid kits were provided and these met the needs of those minor accidents which always occur on construction work or in mining operations. There were no serious accidents throughout the job.

The cost of sanitation and chlorination was £65/2/3 for labour and £22/7/8 for materials. The cost of medical attention was £8/6/11 for labour and £37/15/10 for medical supplies including first aid kits.

MT. PAINTER CAMP

8. CONSTRUCTION

a. Introduction.

The construction period which covered the carting in of all equipment and the erection of mine buildings, plant and machinery commenced immediately upon the completion of the road to the Mount Painter camp on 23/9/44 and was completed to the point of the erection of all major buildings, stores, haulage-way and machinery and the installation of air and water mains to working faces by 23/10/44.

This was a period of great activity. The men worked long hours including Saturdays, Sundays and holidays as it was regarded as vital to commence developmental operations at the earliest possible moment.

The chief items of interest were as set out below.

b. Carting.

By the time the road head reached the Mount Painter camp a considerable bulk of stores, plant and equipment had accumulated at the Copley depot awaiting transport to the field. Arrangements had been made in advance to handle this material. Every motor truck available was pressed into service and a contract was also let to H.E. Ding of Unta. The contractor's drivers and the Mount Painter drivers worked long hours and it is gratifying to record that the construction gang was not held up for materials.

The carting from Copley to the Mount Painter camp totalled 318 tons of which the Mount Painter motor lorries handled 260 tons for £1208/4/4 at a cost of 11/8 pence per ton mile. The contractor carted 58 tons 11 cwt. for £251/15/4 at a cost of 10.98 pence per ton mile.

The road surface was rough and the ruling grades were severe. As time went on the traffic over the roads together with the constant attention of a maintenance gang improved the running surface somewhat. The worst grade which occurred near Arkaroola was by-passed eventually by means of a deviation along the creek bed.

The contractor's price for carting was less than the Departmental cost. At a later date the contractor was again invited to quote. He then advised that he could not undertake the work at less than 1/3 per ton mile as he was out of pocket on the previous contract.

Carting to the East Painter camp involved rather easier conditions in that there was not quite the same rush to get the road through, and the construction gang had profited by their experience in the Radium Creek gorge. The result finally was a better graded track with a somewhat better surface.

In the early stages of the East Painter activities a preliminary track via the old Lady Buxton Mine was adopted. The track was very rough and numerous breakages resulted. As soon as the road head reached the East Painter camp a deviation was constructed on very easy grades from the mouth of the East Painter gorge east to the Wooltana-Moolawatana Road. The result was reflected in the carting costs.

The figures are:-

Carting Copley to East Painter via Balcanoona and Wooltana 45 tons for £173/15/4 at a cost of 11.18 d. per ton per mile. Carting out of East Painter to Copley on the completion of the job, 25 tons for £40/13/1 at a cost of 6.48d. per ton per mile.

c. Construction Schedule.

While the road was being built to the site of the propdsed mining operations, all plant, equipment and stores were under order and were steadily coming forward chiefly from Alice Springs, Adelaide and Melbourne.

The opportunity was also taken during this period to decide on the general layout of the works, and to design details of buildings and equipment. A construction schedule based on a time factor was carefully drawn up covering every item of construction at the field. This schedule set out the order of erection, the date items should be in transit - the date they should arrive at the mine and the date on which installation should be complete. An outline of this schedule, included in the sixth progress report, is as follows:-

Schedule of Preliminary Construction Work (Sept. 25, 1944)

The schedule of preliminary construction is as follows. This is to the point at which mining operations should start:-

TIME CONTROL

	In transit	At Mine	Ready for Operating	Remarks
Temporary stores (2)	-	23.9.44	26.9.44	This is the first line of constructio
Men's Quarters	28.9.44	3.10.44	10.10.44	Refers to the first group of quarters
Engine House	22.9.44	6.10.44	16.10.44	Floor and some of the walling will be completed later.
Compressors	by lorry	7.10.44	16.10.44	Large receiver may have to follow but plant can work with small receivers to start with.
Pump and Engines	24.9.44	30.9.44	10.10.44	Engines on way from W. Australia Pumps " " " Victoria
Lighting Set and Wiring Poles etc.	28.9.44	2.10.44	16.10.44	Electrical Mechanic leaves Adelaide on 28.9.44 taking equipment with him. Lighting set from Victoria.
Pipe Lines (Air)	22.9.44	2.10.44	12.10.44	Now on rails from Alice Springs.
Pipe Lines (Water)	22.9.44	2.10.44	12.10.44	No difficulty in coupling these up as soon as first bore strikes water.
Semi-permanent Stores	-	-	about 16.10.44	When preliminary construction is out of the way.
Magazine	3.10.44	7.10.44	12.10.44	Standard cubicle set. Wood floor - masonite lined - improved later.

	In transit	At Mine	Ready for Operating	Remarks
Sheds for Carpenters Smiths & Sampling Change House	3.10.44	10.10.44	16.10.44	Being made of standard cubicle sets by firms in Adelaide.
Sampling Plant Crusher etc.	-	-	-	After 16.10.44
Staff House	-	-	-	As soon as possible after 16.10.44 (Now coming in from Hatches Creek)
Mine Office	28.9.44	3.10.44	10.10.44	Floor and windows later
Water Supply other than by carting	-	-	-	Depends on result of boring (first bore down 20 ft. on 23,9 (second bore being started 30.9
Rock Drills Hoses Accessories	22.9.44	7.10.44	16.10.44	Being sent to Adelaide for overh and testing.
Rock Drills Air and water mixing tanks	-	-	12.10.44	Being specially made in Melbourne.
Mine Preparation	-	-	12.10.44	Chute to make at 100' Level Plate laying and levelling Portal sets of local gum.

Every item on the schedule was completed to time with the exception of the item of preparation of mine faces which was a few days behind schedule.

At the time this work was put in hand, no skilled men were available on the labour market. However, with the assistance and the good will of several Government Department, it was possible to obtain for the short rush construction period all the labourers, carpenters, pipe fitters, mechanics and electricians required. This assistance came chiefly from the Army, the Railways, the Engineering and Water Supply Department, the Munition Factory at Finsbury and the Munition Factory at Salisbury. A very good crew was got together and they worked with remarkable enthusiasm for long hours and over Saturdays and Sundays and holidays.

A description of the chief items of interest follows.

(i) Engine House and Machinery

The principal items of machinery were housed in a portable all-steel engine shed 44' x 24' which was obtained from Alice Springs.

The steel stanchions were erected on concrete blocks and firmly anchored.

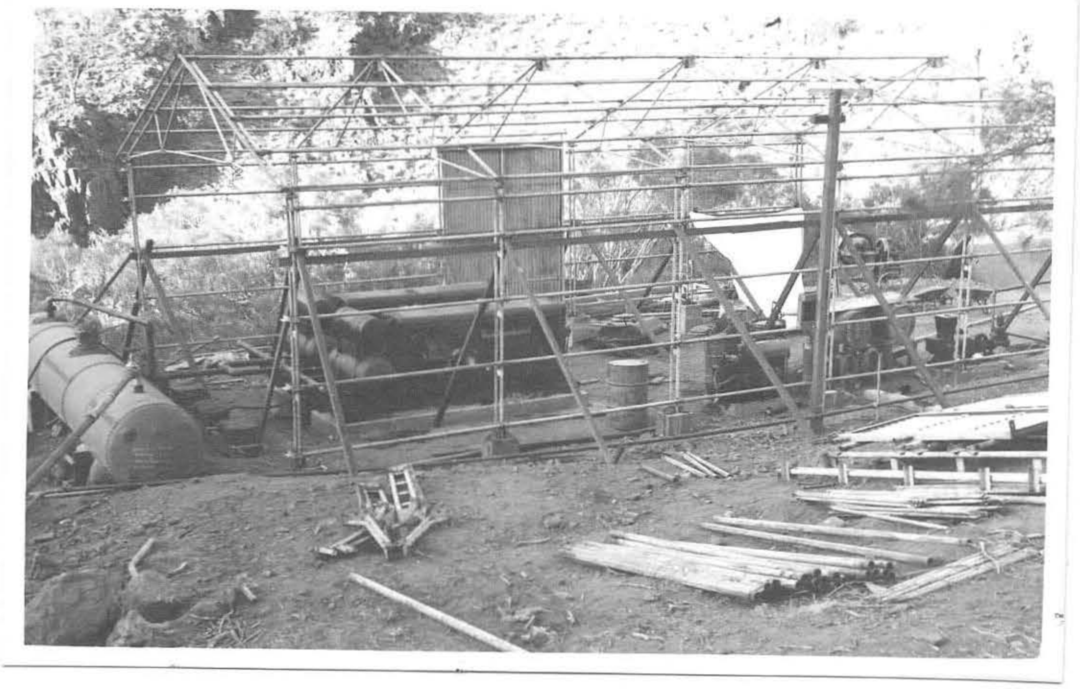


Fig. 10 - Mt. Painter Engine House in course of construction, November, 1944.

The compressors, pump and generator were solidy bedded on concrete foundations.

The costs of erecting the steel shed and the machinery therein were - Labour £190/9/4 - Materials other than the building £22/2/10.

The plant, most of which was loaned to the job, housed in the engine shed, consisted of the following items, the capital values of which were as stated:-

Portable all steel building 44' x 24'	£298	8	6
2 C.P. Air Compressors each 170 c.ft. capacity	2700	0	0
Macson high pressure water pump 3" - capable of lifting 200 ft.	118	0	0
Electric Generator set 5 KVA, 3 ph., 415/240V alternator driven by 8 h.p. Howard engine	300	0	0
Work bench and vices and sundries	15	0	0

The air receiver 22' x 3' 6" complete with valves had a capital value of £146/8/7. This was erected outside the engine house and was coupled up with both air compressors in such a manner that when both compressors were working each compressor took its fair share of the load. After some initial difficulties this arrangement worked satisfactorily. It was a matter of substituting one unloader in lieu of the individual unloaders on each machine.

There was considerable variation in the load on the compressors depending on the intermittent operations of the various machines on the lines, these comprised:-

Air turbine pump to lift water to the top of the hill from the sump at the 100' level.

Air hoist on haulage way.

Air hoists on the winzes.

Jack hammers on the mine faces.

Air operated diamond drill.

Air driven ventilating fans.

As the diamond drill alone require up to 150 cubic feet per minute, there were times when the full capacity of the compressors was insufficient for the load and it was necessary to stagger the periods of the various operations in order to carry on.

(ii) Haulage Way.

The original intention to construct a road approach from the camp site to the mine portals was altered at short notice. This involved a hasty improvisation of an alternative method of getting the mine plant and equipment from the valley level up the hill-side to the mine workings. Air lines were carried up the hill and an air hoist was established there. The steep hill-side was cleared of boulders and a sledge was constructed which was hauled up with the necessary loads. The sledge was guided down with a tail rope. The cost of labour in preparing haulage way was £34/17/2. The capital value of plant including air hoist, ropes and bull wheel was £87/13/-.

On the final design, (see Plan No. 3200 Sheet No. 28), the sledge worked satisfactorily. The cost of the sledge was £8/2/7, of which labour cost £3/9/10 and materials £4/12/9. The chief objections to the use of the sledge were slowness of operation and high consumption of air.

(iii) Air and Water Lines to Mine Faces

One of the first items of construction at the mine end was the laying of air and water mains from the engine house to the mine faces.

The cost of laying the water lines was £205/11/10 of which £35/16/10 was for labour and £169/15/- for pipes, pipe fittings and valves.

The cost of laying the air lines was £270/18/6 of which £82/17/3 was for labour and £188/1/3 for pipes, pipe fittings and valves.

The variations in the mining programme which depended upon the availability of geological data necessitated considerable alterations to air and water lines as faces were closed down and others opened or re-opened. The cost of this dismantling and re-erecting, including relaying portion of the air line with larger pipes was £315/4/- of which labour cost £239/16/3 and materials £75/7/9.

(iv) Buildings

It was considered essential to have lock-up stores for the safe keeping of tools and equipment ahead of the construction gang. This necessitated designing a particularly portable type of structure arranged in section which could be carried in on camel back. The limiting factors in the design were:- no length should exceed 6 feet and no width should be greater than 3 feet. The galvanised iron gable-ended cubicle designed to meet these conditions proved very suitable. Plan No. 3206 Sheet No. 26 sets out the details of the camel cubicle. These structures cost £43/6/6 each on trucks Adelaide

Through the courtesy of the Engineering and Water Supply Department, arrangements were made for the hire of a large mess hall 50' x 20', with a kitchen 16' x 20' attached, complete with portable tables and forms. At one period 75 persons sat down to meals in this mess hall. This structure was loaded on trucks at Robertstown, transported to Copley per rail, thence per contract carting to the mine, where it was re-erected by the construction gang at a cost of £78/4/4 excluding the kitchen.

To cope with the large number of men at one time employed on the mine it was necessary to make some alterations and additions to the kitchen end of the structure and to provide larger stoves. This work, including the erection of the kitchen, cost £83/7/9.

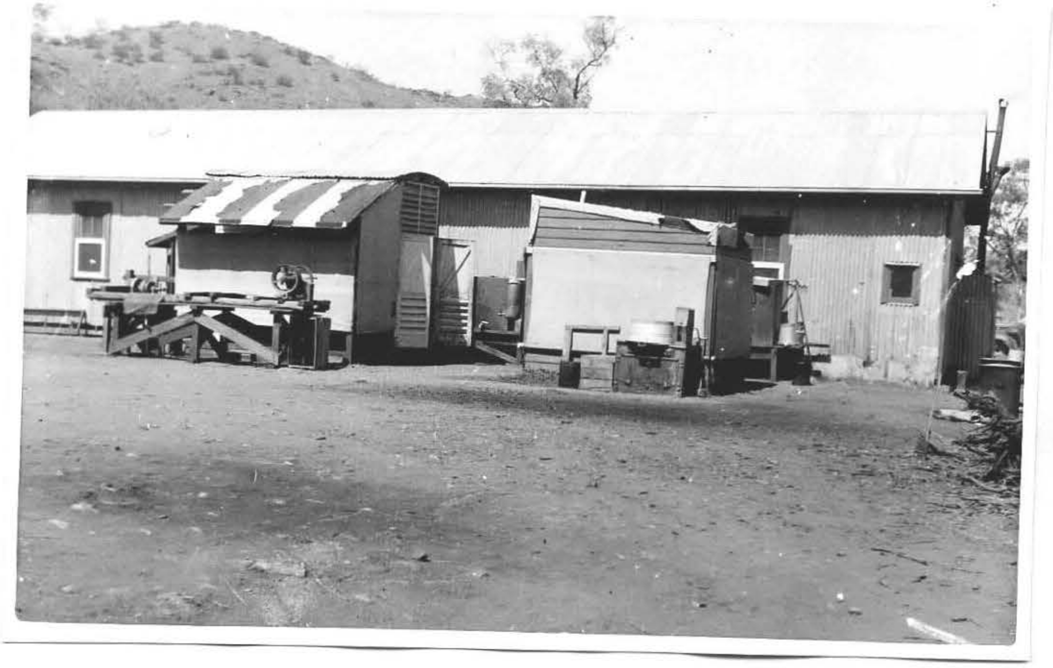


Fig. 11 - Dining Room and Kitchen, Refrigerator House, and Fly-proof Meat Room, Mt. Painter.

The climatic conditions at the mine were severe and it was impossible to handle meat for the constantly varying requirements of the mess without refrigeration. Arrangements were made to obtain a second hand Coldstream refrigerator, capacity 150 cubic feet, from the Director of Mines, Alice Springs. This structure was overhauled by Allied Works Council at Alice Springs at a cost of £150 and forwarded to the job. It was damaged in transit, but was repaired and re-erected in close proximity to the mess room. The cost of erecting the refrigerator and a fly proof room for cutting up meat and also establishing a home-made incinerator was £36/14/11.

The Mount Painter Camp enjoyed the reputation of being one of the best bush camps established in South Australia. This was due chiefly to the care taken in the matter of refrigeration, sanitation and supplying qualified cooks. No complaints were made by the workmen's committe, which constitutes a record in the writer's experience.

It must be borne in mind that the Mount Painter camp was in an out of the way place, too far removed from the nearest bush township for the men to seek relaxation and amusement away from the camp. Throughout the job, the personnel remained on the mine with the exception of occasional short trips to the nearest water hole for bathing.

(v) Sleeping Quarters and other Buildings

Careful thought was given to the design of typical framed sectional structures fabricated in such manner as would enable the component parts or cubicles to be assembled singly or otherwise in rows or back to back or as might be required for the various types of buildings.

Fig.12 - Block of Sleeping Quarters, Mt. Painter, Nov., 1944.

Two rows of five cubicles back to back constituted one block of sleeping quarters containing ten cubicles which gave accommodation for 20 men. There were three of these blocks, two providing beds, each for 20 men, and the third which consisted of 8 cubicles provided beds for 16 men - 56 beds in all. To accommodate the staff, foremen and visitors, the tents which were erected before the road was completed, were utilised. The maximum number of beds in use at the Mount Painter camp at any one time was 75.

Fig.13 - Small Store, Mt. Painter, November, 1944.

A group of 4 cubicles set in the form of a square was adopted for the mine store near No. 6 workings. A group of 4 cubicles in a row formed the miners' change house near the mine store.

Other units were, 1 cubicle for Manager's office inside the mess hall, 2 for latrine, 2 for laundry and shower room and 1 (modified) for blacksmith shop.

Costs were:-

The blocks of 20 bed cubicles cost £233/0/9 each (52' x 18') comprising materials £124/17/-, fabrication £30, erection £78/3/9.
The block of 16 bed cubicles 42' x 18' cost £199/1/1; materials £112/10/1, fabrication £24, erection £62/11/-.
The mine store 23' x 18' cost £139/10/8, materials £58/2/-, fabrication £12, erection £69/8/8.
The miners' change house 42' x 9' cost £131/6/8, materials, £55/18/3, fabrication £12, erection £63/8/5.
The staff laundry and shower room 24' x 11' cost £41/14/10; materials £25/8/9, fabrication £6, erection £10/6/1.
Flyproof latrine cost £47/12/7, materials £19/8/2, fabrication £6, erection £22/4/5.
Blacksmith's shop cost £62/6/4; materials £30/19/-, erection £31/7/4.
Other building were:-
Gable roofed cubicle for camel transport. Cubicle £43/6/6 Materials for benches £3/19/1 erection for use as a store £8/1/6.
Sectional steel store from Alice Springs.
Portion erected was 24' x 12'; materials for benches £13/15/6, cost of erecting store and benches £52/0/7.
Depot at East Painter turn-off.
(Cubicle frame covered with fibreen) cost £24/2/-; materials £8/17/4, fabrication £3, erection £12/4/8.

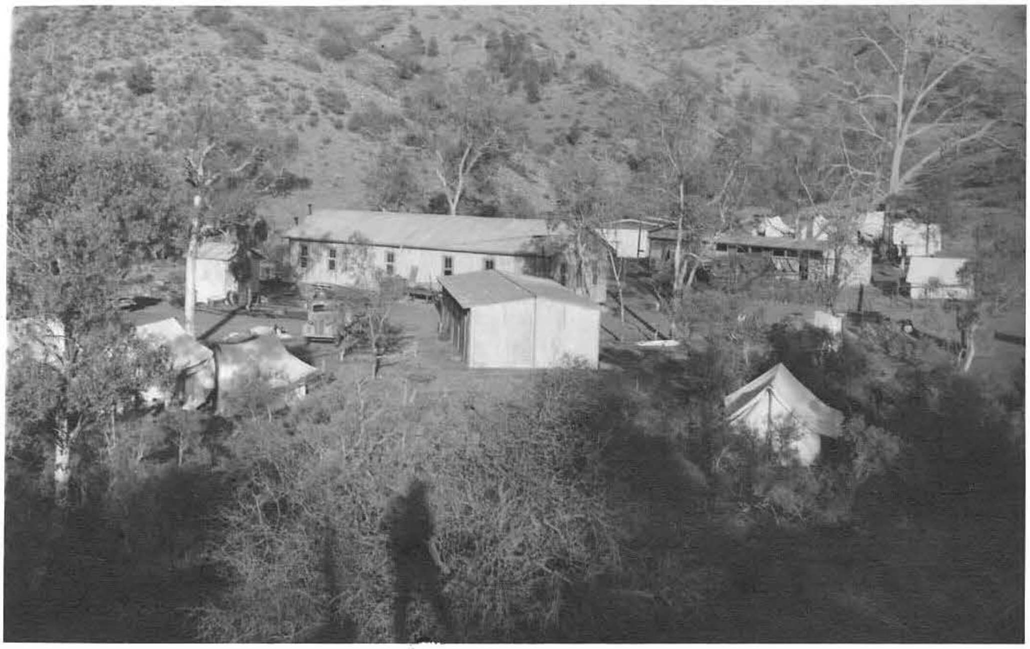


Fig 14 - General View Mt. Painter Camp, November, 1944, showing Kitchen and Mess Building, Sleeping Quarters (3 blocks), Camel Cubicle, and Tents.

9. CONSTRUCTION - EAST PAINTER CAMP.

The first accommodation at East Painter was provided by six tents and three of the portable "camel cubicles" carted in on camel back. Six camels were hired from Mr. Bentley Greenwood of Arkaroola Station at the rate of 5/- per camel per day. A camel driver to look after the camels and to do other work was released from the Navy. The camel team did excellent work in supplying the camp with equipment stores, foodstuffs and water until the road was completed.

When the road head reached the East Painter camp, construction materials were carted in and the buildings mentioned below were erected. Plan No. 3194 Sheet No. 25 shows the lay-out of the camp.

A mess room 34' x 17', with a kitchen 15' x 17' attached was in use at the Engineering and Water Supply Department's road camp at the mouth of the East Painter gorge. This structure was purchased for £50 from the Engineering and Water Supply Department and was carted to the site and erected there by the road gang. The chief items of interest were as follows:-

Three portable gable cubicles (Plan No. 3206 Sheet No. 26) designed in sections for camel transport were erected at a cost of £43/6/6 each plus the cost of preparing the site and erecting the three cubicles which amounted to £71/19/9.

The same practice of using the standard sectional design for sleeping quarters, office and store as at Mount Painter was adopted at East Painter. This gave flexibility of construction and made the structures available for use elsewhere.

Sleeping quarters consisted of two rows of three cubicles bakc to back forming a block of six cubicles and provided accommodation for twelve men. The cost of this block was £163/15/11 of which labour cost £70/5/8 and materials £93/10/3.

The Manager's quarters cost £81/4/-, of which labour cost £41/10/4 and materials £39/13/8.

The mine office cost £66/18/8, labour £36/13/1, materials £30/5/7.

The mine store cost £72/11/10, labour £42/6/3, materials £30/5/7.

A shed for the blacksmith was erected at a total cost of £10/2/11.

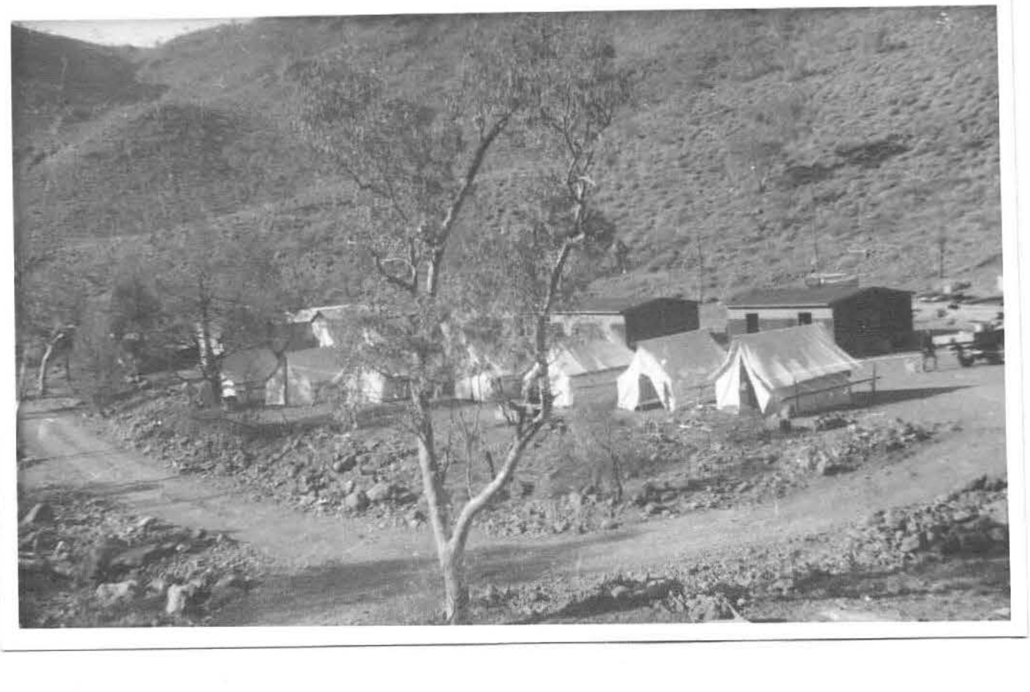


Fig.15 - General View of East Painter Camp.

At East Painter, to speed up the driving of No. 1 adit, a portable Broomwade air compressor which had been used on the road construction through the East Painter gorge, was hired from the Engineering and Water Supply Department. This was installed on the road immediately below No. 1 adit and air and water pipes were laid to the working face. The water required for the S.L.11 Holman hammers was carted to the sump and lifted to a head tank on the hill-side above the adit by means of a single stage air driver sump pump. The cost of installing this portable plant including air and water mains was £59/19/4 of which materials cost £53/12/9, labour £6/6/7.

No other construction work of importance was carried out at East Painter. Prior to the closing down of all operations at East Painter, detailed plans were in preparation for the installation of a gravity ropeway from No. 5 workings to the road head and also for the erection of an air compressor and air and water lines to the working faces on that lode. These proposed works remained in abeyance.

10. PREPARATION OF MINE FACES AT MOUNT PAINTER.

Before the approved mine developmental programme could be commenced, considerable work was involved in preparatory operations. The old workings were narrow and low and badly graded. On the 100 ft. level there was a jump-up of 7 feet half way along the level. This necessitated breaking down the back and installing a chute and involved double trucking for all operations on that level. Before diamond drilling could be started it was necessary to cut chambers for each set up. Before winze sinking could be commenced, chambers had to be cut to give head room for hauling the kibbles. Rails had to be curved and laid and flat sheets installed.

11. DEVELOPMENT AND EXPLORATION, FOOTAGES.

At the inception of the job the programme of development schedule for completion by the end of 1944 (required as a guide for further operations) was:-

Winzing	100 feet.
Driving	100 "
Shaft Sinking	20 "
Diamond Drilling	200 "
Boring for water until located.	

Development commenced on the first week of November, 1944. On 17/12/44 when the miners left the locality on Christmas leave, the footage completed was:-

Mount Painter.

Winzing	96 feet
Driving	77 "
Shaft Sinking	15 "
Rising	17 "
Diamond Drilling	649 "
Boring for Water	404 "

East Painter.

Costeaning	60 feet.
Open cutting	50 "
Shaft sinking	6 "
Boring for water	150 "

Mining and diamond drilling and water boring operations were resumed on 13/1/45. Mining development continued on a reduced scale as and where required by the geological control until 23/5/45. Diamond drilling ceased on 18/8/45.

A summary of the total footage of developmental and exploratory work from November, 1944 to August, 1945 inclusive, is given below.

Mount Painter.

Winzing	126 feet
Driving	460 feet.
Shaft sinking	19 feet.
Rising	17 feet.
Cross Cutting	18 feet.
Diamond drilling	1249 feet.
Boring for water	404 feet.

East Painter.

Driving	78 feet.
Shaft sinking	40 feet.
Cross cutting	12 feet.
Open cutting	50 feet.
Costeaning	140 feet.
Boring for water	429 feet.

12. DESCRIPTION OF MINING OPERATIONS.

a. Introduction

At the time the Mount Painter Project was put in hand, experienced machine miners and diamond drillers were not available. It was found necessary to obtain approval for the release of the required personnel from the Army. The number of men so released was 31. Although outside Army control, provision existed for civil action to be taken in the event of the released men failing to complete their obligation to work on the Mount Painter Project for such period as might be required. No trouble was experienced. The men accepted the conditions and set to work with a will and with the under-lying idea of earning as much money as possible before visiting their homes during the Christmas period.

Broken Hill miners as a class are shrewd, resourceful and skilful. The released men were no exception. They worked hard and continuously without thought of visiting Copley for relaxation, the spur to their efforts being the desire to take advantage of the somewhat generous footage bonus offered by the management.

All mining operations were under geological control under the Director of Mines. The assistant Government Geologist who was associated with the project resided on the mine most of the time for the purpose of recording and reporting to the Director all matters which might have a bearing on the search for ore bodies.

At Mount Painter the programme of development as set out at any one time was subject to modification at short notice in the light of geological disclosures or for other reasons. This made it necessary for the management to adopt a more flexible method than the usual contract system for payment by results. The mine manager recommended a bonus system as being the most suitable under the conditions existing at Mount Painter. The mine manager was authorised to fix bonus rates at his discretion, depending chiefly on the hardness of the ground at the various faces.

The flexibility and suitability of this sytem was demonstrated at Mount Painter where it was necessary not only to provide an incentive to speed up operations but also to avoid the resentment which miners feel at frequent changes in the form and direction of their work.

The South Australian Government's Industrial Officer adopted the A.W.U. (Australian Workers' Union) award for the Mount Painter area, and by reason of its out of the way location, he granted a camp allowance of £2/5/- per week, the men to order and pay for their own food and the freight thereon.

The A.W.U. award rates were:-

Miners £5/11/- for a week of 40 hours
Shaft sinking £5/17/- for a week of 40 hours
Truckers, underground, £5/2/- for a week of 40 hours.
Winch men £5/14/- for a week of 40 hours.
Labourers £4/19/- for a week of 44 hours.

Overtime was paid for in accordance with the A.W.U. award, at the rate of time and a half, gazetted holidays at double time if worked. No overtime rates were paid to men working on a bonus or contract.

The bonus rates per foot set by the mine manager at Mount Painter were:-

50' Level N. Drive - 24/- increased toward the end of the job to 27/- per foot; the extra 3/- in payment for miners doing their own trucking.

50' Level rise - 16/- per foot (preparation of mine faces)

50' Level winze - 16/-, increased successively to 18/-, 20/- and 22/- per foot.

80' Sub-level driving-22/- increased to 24/-

100' Level South Portal Drive -20/- increased to 24/- and 26/-.

100' Level N. drive - 24/- increased to 26/- and 27/- including own trucking toward the end of the job.

100' Level Winze - 16/- increased to 18/- and 22/-

150' Sub-level drive - 24/- per foot.

150' Sub-level crosscut at 27/-

Surface shaft 20/- per foot.

Schedule No. 2 attached to this report gives a complete list of the earnings of all the miners on bonus work for the whole of the time they were so employed at Mount Painter.

SCHEDULE NO. 2

MOUNT PAINTER - NO. 6 WORKINGS.

Schedule showing nett earnings of bonus and contract miners.

rtnight ing	Name	Daywork			Bonus			Camp Allow-ance			Deductions						Nett Earning		
											Mess			Income Tax					
		£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d
/44	Delbridge, T.H.	17	6	10	7	5	0	4	10	0	3	3	0	4	2	6	21	16	4
	Hill, F.H.	17	6	10	7	5	0	4	10	0	3	3	0	8	18	0	17	0	10
	Carlaw, E.G.	18	7	9	8	0	0	4	10	0	3	3	0	10	5	6	17	9	3
	Netherwood, A.C.	18	6	4	8	0	0	4	10	0	3	3	0	6	17	6	20	15	10
	Rawson, R.J.	17	9	7	4	4	0	4	10	0	3	3	0	4	9	6	18	11	1
	Ellis, S.O.L.	17	15	2	4	4	0	4	10	0	3	3	0	8	18	0	14	8	2
	Bailey, L.M.	19	15	10		-		4	10	0	3	3	0	7	12	0	13	10	10
	Perko, C.T.	17	9	7	4	14	6	4	10	0	3	3	0	9	0	6	14	10	7
	Harvey, W.C.	20	15	2		-		4	10	0	3	3	0	5	12	0	16	10	2
	Savage, M.	20	17	11		-		4	10	0	3	3	0	6	4	0	16	-	11
	Campbell, A.	11	11	9				1	18	7	1	1	0	2	10	0	9	19	4
2/12/44	Delbridge, T.H.	20	1	8	9	5	0	4	10	0	2	2	0	8	5	0	23	9	8
	Hill, F.H.	20	3	1	9	5	0	4	10	0	2	2	0	9	17	0	21	19	1
	Carlaw, E.G.	23	10	3	11	8	6	4	10	0	2	2	0	13	8	0	23	18	9
	Netherwood, A.C.	23	13	0	11	8	6	4	10	0	2	2	0	10	19	6	26	10	0
	Ellis, S.O.L.	23	2	10	5	4	0	4	10	0	2	2	0	10	13	0	20	1	10
	Rawson, R.J.	23	3	6	5	4	0	4	10	0	2	2	0	8	7	0	22	8	6
	Bailey, L.M.	23	9	8		-		4	10	0	2	2	0	8	10	6	17	7	2
	Campbell, A.	23	14	6		-		4	10	0	2	2	0	5	16	0	20	6	6
	Ellenson, P.P.	23	15	11		-		4	10	0	2	2	0	8	15	6	17	8	5
	Harvey, W.C.	23	19	5		-		4	10	0	2	2	0	6	4	0	20	3	5
	Savage, M.	23	17	5		-		4	10	0	2	2	0	6	10	0	19	15	5
/44	Carlaw, E.G.	16	10	4	19	18	3	4	10	0	3	0	1	14	3	0	23	15	6
	Netherwood, A.C.	16	8	11	19	18	3	4	10	0	3	0	1	11	12	0	26	5	1
	Bailey, L.M.	19	1	10	17	3	9	4	10	0	3	0	1	14	3	0	23	12	6
	Harvey, W.H.	18	16	7	17	3	9	4	10	0	3	0	1	11	3	6	26	6	9
	Delbridge, T.H.	15	10	10	14	17	0	4	10	0	3	0	1	8	12	6	23	5	3
	Hill, F.H.	19	8	10	3	9	6	4	10	0	3	0	1	7	2	0	17	6	3
	Ellis, S.O.L.	15	16	5	13	13	0	4	10	0	3	0	1	11	3	0	19	16	4
	Rawson, R.J.	15	16	5	13	13	0	4	10	0	3	0	1	8	17	0	22	2	4
	Best, H.	22	18	3		-		4	10	0	3	0	1	8	13	0	15	15	2
	Savage, M.	22	18	0		-		4	10	0	3	0	1	6	2	0	18	5	11
	Campbell, A.	18	8	10	11	7	6	4	10	0	3	0	1	8	1	6	23	4	9
13/12/44	Nil																		
13/1/45	Nil																		
27/1/45	Delbridge, T.H.	15	9	8	13	10	0	4	10	0	1	11	6	8	6	0	23	12	2
	Campbell, A.	14	10	0	13	10	0	4	10	0	1	11	6	7	10	0	23	8	6
	Netherwood, A.C.	15	5	8	13	10	0	4	10	0	1	11	6	8	12	0	23	2	2
	Carlaw, E.G.	15	5	8	13	10	0	4	10	0	1	11	6	9	12	0	22	2	2
	Rawson, R.J.	14	3	0	14	14	0	4	10	0	1	11	6	8	12	0	23	3	6

Schedule showing nett earnings of bonus and contract miners.

Fortnight Ending	Name	Daywork			Bonue			Camp Allow-ance			Deductions Mess			Income Tax			Nett Earnings		
		£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d
27/1/45	Ellendon, P.P.	16	16	8	6	10	0	4	10	0	1	11	6	8	10	6	17	14	8
	Perry, H.E.	15	15	11	2	9	4	4	10	0	1	11	6	6	10	0	14	13	9
	Smith, J.A.	15	19	2	2	0	0	4	10	0	1	11	6	6	6	0	14	11	8
	Ellis, S.O.L.	8	2	4	8	4	0	2	11	5	-	18	0	6	9	0	11	10	9
	Savage, M.	6	2	9		-		2	11	5	-	18	0	1	5	0	6	11	2
10/2/45	Carlaw, E.G.	14	18	5	25	0	0	4	10	0	3	6	6	14	7	0	26	14	11
	Netherwood, A.C.	14	18	5	25	0	0	4	10	0	3	6	6	13	4	6	27	17	5
	Rawson, R.J.	15	10	10	15	18	0	4	10	0	3	6	6	9	12	0	23	0	4
	Ellis, S.O.L.	15	2	6	15	18	0	4	10	0	3	6	6	11	18	0	20	6	0
	Ellendon, P.P.	16	18	7	5	8	0	4	10	0	3	6	6	8	2	0	15	8	1
	Smith, J.A.	19	3	6		-		4	10	0	3	6	6	6	16	0	13	11	0
	Percy, H.E.	18	1	11		-		4	10	0	3	6	6	6	8	0	12	17	5
24/2/45	Carlaw, E.G.	14	9	3	18	0	0	4	10	0	3	6	6	11	2	0	22	10	9
	Netherwood, A.C.	14	9	3	18	0	0	4	10	0	3	6	6	9	19	6	23	13	3
	Ellis, S.O.L.	13	4	3	15	16	6	4	10	0	3	6	6	8	14	6	21	9	9
	Rawson, R.J.	13	4	3	15	16	6	4	10	0	3	6	6	11	0	6	19	3	9
/45	Ellis, S.O.L.	10	12	4	6	15	0	2	5	0	1	14	10	5	10	3	12	7	3
	Rawson, R.J.	10	12	4	6	15	0	2	5	0	1	14	10	4	2	3	13	15	3
	Carlaw, E.G.	10	12	4	9	0	0	2	5	0	1	14	10	5	13	3	14	9	3
	Netherwood, A.C.	10	12	4	9	0	0	2	5	0	1	14	10	5	0	3	15	2	3
3/45	Cooper, D.G.	9	13	7		-		2	5	0	1	4	6	3	9	0	7	5	1
	Hayward, E.W.	9	13	7		-		2	5	0	1	4	6	3	9	0	7	5	1
24/3/45	Percy, H.E.	14	4	5	14	3	6	4	10	0	2	9	0	10	13	0	19	15	11
	Smith, J.A.	14	4	5	14	3	6	4	10	0	2	9	0	10	13	0	19	15	11
	Cooper, D.G.	14	5	10	11	8	0	4	10	0	2	9	0	9	10	6	18	4	4
	Hayward, E.W.	14	5	10	11	8	0	4	10	0	2	9	0	9	10	6	18	4	4
7/4/45	Percy, H.E.	13	6	5	15	6	0	4	10	0	2	9	0	10	15	6	19	17	11
	Smith, J.A.	13	6	5	15	6	0	4	10	0	2	9	0	10	15	6	19	17	11
	Cooper, D.	13	9	2	10	2	6	4	10	0	2	9	0	8	13	0	16	19	8
	Hayward, E.	13	9	2	10	2	6	4	10	0	2	9	0	8	13	0	16	19	8
21/4/45	Percy, H.E.	14	7	3	10	11	6	4	10	0	2	12	6	9	4	0	17	12	3
	Smith, J.A.	14	7	3	10	11	6	4	10	0	2	12	6	9	4	0	17	12	3
	Cooper, D.	14	5	10	11	3	6	4	10	0	2	12	6	9	8	0	17	18	10
	Hayward, E.	14	5	10	11	3	6	4	10	0	2	12	6	9	8	0	17	18	10
/5/45	Cooper, D.	14	11	4	14	10	3	4	10	0	2	12	6	11	0	6	19	18	7
	Hayward, E.	14	11	4	14	10	3	4	10	0	2	12	6	11	0	6	19	18	7
/5/45	Cooper, D.	14	5	10	12	3	0	4	10	0	2	9	2	9	15	6	18	14	2
	Hayward, E.	14	5	10	12	3	0	4	10	0	2	9	2	9	15	6	18	14	2

At East Painter where the conditions were simpler, a footage contract system was adopted, the rates being set by the manager and based on the local conditions.

All mining operations at East Painter whether on the surface or underground, were carried out by hammer and tap methods with the exception of the drive at the No. 1 adit for which a portable Broomwade air compressor and rock drills of the jack-hammer type were installed. At a later stage a larger air compressor and the requisite piping were made available for carrying on the work at No. 5 lode, but this plant was not installed because of the cessation of mining operations at East Painter.

The contract rates at East Painter were:-

Driving No. 1 adit at £2 per foot.
Sinking shaft on No. 5 Lode £3/7/6 increased to £4 per ft.
Driving 40' level on No. 5 Lode £3/7/6 per foot.
Crosscutting 40' level No. 5 level. £3/7/6 per foot.

Schedule No. 3 shows the earnings of all the miners on contract work at East Painter.

SCHEDULE NO. 3

EAST PAINTER

Schedule showing nett earnings of contract miners.

Fortnight Ending	Name	Daywork			Contract			Camp Allowance			Deductions						Nett Earnings		
											Mess			Income Tax					
		£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d	£	s.	d
13/1/45	Flaherty, M.	6	9	1	10	9	10	4	10	0	2	9	0	5	18	0	13	1	11
	Hayward, E.	6	9	1	10	9	10	4	10	0	2	9	0	5	18	0	13	1	11
	O'Donoghue, M.	1	13	4	8	1	5	2	17	10	1	11	6	3	15	0	7	6	1
/45	Flaherty, M.				18	13	7	4	10	0	2	9	0	6	12	0	14	2	7
	Hayward, E.				18	13	7	4	10	0	2	9	0	6	12	0	14	2	7
	Cooper, D.				18	13	7	4	10	0	2	9	0	6	12	0	14	2	7
	Harvey, W.H.				10	4	4	2	5	0	1	4	6	2	9	0	8	15	10
	McInerney, J.	13	12	5				4	10	0	2	9	0	3	12	0	12	1	5
	O'Donoghue, M.	4	5	3	11	4	3	4	10	0	2	9	0	5	6	0	12	4	6
	Bailey, L.M.				10	4	4	2	5	0	1	4	6	3	13	0	7	11	10
10/2/45	Hayward, E.				19	16	7	4	10	0	1	18	6	7	2	0	15	6	1
	Cooper, D.				19	16	7	4	10	0	1	18	6	7	2	0	15	6	1
	Flaherty, M.				19	16	7	4	10	0	1	18	6	7	2	0	15	6	1
	Bailey, L.M.				23	12	2	4	10	0	1	18	6	8	13	0	17	10	8
	Harvey, W.H.				23	12	2	4	10	0	1	18	6	6	2	0	20	1	8
	O'Donoghue, M.	15	15	0				4	10	0	2	1	6	5	8	0	12	15	6
	McInerney, L.	12	5	7				3	10	8	1	13	3	2	16	0	11	7	0
24/2/45	Gangell, A.H.	12	0	0				4	10	0	2	2	0	3	0	0	11	8	0
	Hayward, E.				21	17	9	4	10	0	2	2	0	7	18	0	16	7	9
	Cooper, D.				21	17	9	4	10	0	2	2	0	7	18	0	16	7	9
	Flaherty, M.				21	17	9	4	10	0	2	2	0	7	18	0	16	7	9
	Bailey, L.M.				30	1	1	4	10	0	2	2	0	11	8	0	21	1	1
	Harvey, W.				30	1	1	4	10	0	2	2	0	8	16	0	23	13	1
/3/45	Hayward, E.	4	3	3	4	10	0	2	5	0	1	1	0	3	1	0	6	16	3
	Cooper, D.	4	5	4	4	10	0	2	5	0	1	1	0	3	2	0	6	17	4
	Flaherty, M.	12	9	9	4	10	0	4	10	0	2	2	0	5	18	0	13	9	9

The plant for machine mining was obtained, some new and some second-hand, from the Director of Mines at Alice Springs through the Department of Supply and Shipping, Melbourne. Suitable plant was not available elsewhere at the time.

The rock drills supplied were of the following types:

C.P. 50 drifters
Ingersoll Rand, J.B. 4's.
Holman Silver Bullets
Holman S.L. 11's.

The C.P. 50 drifters were excellent machines but required too much air for the conditions at Mount Painter. They were discarded by the machine miners in favour of the smaller types. The Ingersoll Rand J.B. 4's and the Holman machines all did good work and there was not much to choose between them. Two Holman S.L. 11's. were used at East Painter.

b. Mining Methods.

Most of the miners on the job were from Broken Hill but there was a sprinkling of men with Western Australian experience as well. There was a difference of opinion as to the best type of cut to be adopted in driving. The Western Australian miners adhered to the drag cut with which they were familiar, but the Broken Hill men showed a marked preference for the post hole cut, which is a type of burned out centre cut.

Plans Nos. 3198 and 3199 Sheets Nos. 29 and 30 illustrate both drag cut and post hole cut as used on the field. Careful analysis of the results indicated that the post hole cut had the advantage both as to rate of advance and cost of explosives.

The Western Australian miners using the drag cut operated with a 4 foot horizontal bar with one set up, boring 18 holes in a round. The objective (never quite realised) was to set up, bore, fire and bog one three foot cut per shift. The firing order is shown on Plan No. 3198 Sheet No. 29.

The Broken Hill miners adopted the post hole cut using a 6 ft. vertical bar with one set up drilling a round of 18 holes 6 ft. deep square with the face. Their objective was a 6 ft. advance in a shift and a half. This was seldom if ever realised, but it was not uncommon to get a 6 ft. cut cleaned right out in 2 shifts. In this system the centre cut consisted of 5 holes close together. The centre hole only of this group was loaded. It was loaded right to the collar. The four holes in the form of a square were not loaded. The firing order is shown on Plan No. 3199, Sheet No. 30.

Some of the machines from Alice Springs were fitted with ⅞" chucks and some with 1" chucks but the steel available from that source proved to be much less than anticipated. Difficulty was experienced in obtaining collared and shanked steel of both sizes at short notice. Steel was obtained locally. This was collared and shanked by Stonyfell Quarries Ltd. Screwing when necessary, was done by A. Noble and Sons, Adelaide.

The Supply and Shipping Department in Melbourne was able to forward modern Jack bits type 1 which will fit either ⅞" or 1" steel. These were in sets of three sizes 1⅞", 1¾" and 1⅝". These jack bits were suitable for moderately hard ground but in the brecciated country carrying haematite there was a tendency for the softer materials to clog because of inadequate water pressure. Water for the drills was obtained from tanks near the top of the hill, but this was only about 30 feet above the highest point of operation. Standard Ingersoll Rand pressure tanks were supplied to boost up the water pressure to equal the air pressure if necessary. The men, however, were not familiar with this equipment.

The advent of a first-class blacksmith on the mine improved conditions and the men then adopted the chisel bit, relying on the blacksmith to give them the temper they needed on their steel. The blacksmith's shop was located at the mouth of the 100 ft. level, near enough to all working faces for easy contact with the miners. Under these conditions the chisel bits proved successful and were adopted throughout the mine.

When not fully employed on sharpening drill steel the blacksmith was engaged in making chain ladders, spikes, gads, ladder dogs, etc. for both Mount Painter and East Painter. The total cost of blacksmithing and tool sharpening at Mount Painter was £444/17/1. At East Painter the cost was £64/12/9.

Schedule No. 4 illustrates mining costs at Mt. Painter and also at East Painter.

SCHEDULE NO. 5

MOUNT PAINTER NO. 6 WORKINGS

SCHEDULE SHOWING COST PER FOOT OF MAIN HEADINGS.

Location	Footage in Feet	Miners Wages	Miners Bonuses	Miners Camp Allowance	Stores	Air	Smithing and Repairs	Water	Hire of Plant	Holiday Pay	Total Cost	Cost per foot
		£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d	£ s. d
Winzes below 50' & 100' L.	126	313 13 3	105 13 6	86 5 10	47 10 -	61 12 8	88 6 4	12 0 8	24 7 4	17 0 5	756 10 0	6 0 1
South Portal drive 100' L.	94	126 9 1	101 3 0	36 15 0	44 1 11	45 19 11	65 17 4	8 19 2	18 3 4	12 14 0	460 2 9	4 17 11
North Drive 100' L.	186	235 17 11	229 16 0	74 5 0	109 11 2	91 0 3	130 7 8	17 15 0	35 19 9	25 2 7	949 15 4	5 2 2
North Drive 50' L.	129	179 16 8	177 4 6	55 10 0	82 13 10	63 2 5	90 8 4	12 6 2	24 18 11	17 8 6	703 9 4	5 9 1
Drive 80' Sub.L.	51	38 17 2	60 4 0	12 0 0	24 17 10	24 18 10	35 14 11	4 17 6	10 2 9	6 17 9	218 10 9	4 5 9
Drive 150' Sub. L.	6	4 16 6	7 4 0	1 10 0	2 12 3	2 19 3	4 4 2	- 11 4	1 2 4	- 16 2	25 16 0	4 6 0
Crosscut 150' Sub. L.	12	16 13 0	16 4 0	5 5 0	13 11 3	5 17 5	8 8 3	1 2 8	2 5 6	1 12 4	70 19 5	5 18 3
Preparation of Mine Faces	-	356 17 4	- - -	98 2 5	46 8 2	24 9 7	35 0 11	4 15 6	9 12 9	6 15 1	582 1 9	- - -
EAST PAINTER WORKINGS.												
Drive No. 1 Adit	50	- - -	120 0 0	24 0 0	29 3 6	20 16 0	24 17 3	Machines 11 5 0	67 4 2	19 0 7	316 6 6	6 6 6
Drive 40' L. No. 5 Lode	28	- - -	96 3 9	22 19 0	18 16 7	Nil	13 18 5	Hand Labour 1 0 0	9 12 11	10 13 2	173 3 10	6 3 8
Sinking Shaft No. 5 Lode	40	59 2 10	87 10 0	43 2 6	22 4 3	Nil	19 17 9	Hand Labour 1 0 0	13 15 9	15 4 5	261 17 6	6 10 11
Crosscuts 40' Level No. 5 Lode	12	- - -	40 10 0	9 0 0	5 12 11	Nil	5 19 4	Hand Labour - - -	4 2 8	4 11 3	69 16 2	5 16 4

c. Plate Laying

250 yards of light tram tracks were laid to the various working faces at a cost of £302/5/4 of which labour cost £134/13/10 and materials £167/11/6. The cost per yard of track was 24/2. The rails used were steel, 14 lbs. to the yard spiked to 4" x 1½" Karri sleepers with ⅜" dog spikes.

d. Underground Trucking

The jump up of 7 feet on the 100 feet level added considerably to trucking difficulties. It was necessary to construct a chute at this point and to re-truck from the chute to the dump. To give working room for the chute it was necessary to blast down the back and widen the drive. The grades on the old working levels were very bad. Time did not permit of regrading - all that could be done was to make the most of existing conditions.

Truckers were paid at A.W.U. Award rates of £5/2/- per week of 40 hours with overtime at time and a half. The total cost of trucking was £452/7/7 at Mount Painter. This is equivalent to 14/7 per foot of development trucked.

At East Painter where only one short adit of 50 feet was driven, the trucking was included in the contract rate for driving the No. 1 adit.

e. Surface Prospecting Shaft at Mount Painter.

For prospecting purposes a shaft was sunk on top of the hill to the north of the No. 6 workings. The sinking was done by hammer and tap methods as air was not available. The depth was 19 feet on the underlie. The ground was heavy and required supporting with timbers, a good deal of timber was also required at the collar of the shaft.

The cost including timbering, was £279/7/11 of which labour cost £114/11/8 and stores £164/16/3, the cost being £13/6/1 per foot. This shaft failed to disclose anything of interest and it was discontinued by the Geological control.

f. Diamond Drilling.

A programme of diamond drilling commenced on 29th October, 1944, and continued until 21st August, 1945 under the supervision of the Mechanical and Boring Engineer of the Mines Department and under the direction of the geological control by the Director of Mines through the Assistant Government Geologist.

An air driven Sullivan No. 6 machine was used, producing a core 1⅛" diameter. The bits used were machine cast and made by Boyles Brothers, Melbourne. The diamonds were Congo brilliants. Duffield's patent double tube core barrel was used. This gave a core recovery of 86 per cent over the total footage drilled. Much of the ground passed through was badly fractured and produced broken core.

The foreman diamond driller was paid at the rate of £6/10/- and his assistant at the rate of £5/9/- per week of 40 hours. They also received the camp allowance for the locality of £2/5/- per week. A bonus of 1/- per foot of hole drilled and 1/- per foot of core recovered was also granted as an incentive to quick and careful work. This bonus was divisible by the two men operating the drill.

The conditions at Mount Painter were somewhat difficult as far as diamond drilling was concerned. The bonus earnings per man were less than the bonus earnings of the miners. On the diamond drilling the bonus earnings averaged about 30/- per week per man.

Schedules Nos. 5 and 6 set out the main points of interest.

SCHEDULE NO. 5

DIAMOND DRILLING AT MT. PAINTER - NO. 6 WORKINGS.

Schedule showing location of holes and core recovery.

Location	Type	Hole Number	Depth in Feet	Core Recovery
100' Level	Horizontal	1	201' 0"	166' 4"
100' Level	Depressed	2	112' 9"	91' 9"
100' Level	Horizontal	3	221' 8"	204' 2"
100' Level	Depressed	4	115' 1"	85' 9"
100' Level	Horizontal	5	100' 0"	89' 9"
185' Level	Depressed	6	330' 8"	298' 3"
50' Level	Horizontal	7	167' 11"	139' 11"
			1249' 1"	1075' 11"

SCHEDULE NO. 6

DIAMOND DRILLING AT MT. PAINTER - NO. 6 WORKINGS.

Date Commenced 5/11/44 Date Completed 28/8/45

Size of Core 1$\frac{1}{8}$"

Total Footage drilled	1249 ft. 11 in.				
Total core recovered	1075 ft. 11 in.				
Percentage core recovery	86.1%				
Cost of Wages, day work	£704	16	6	Cost per foot	16/3
Cost of Camp	187	17	6		
Cost of Bonuses	121	7	0		
Cost of stores, including bits	314	19	0	do.	5/-
Cost of air	366	14	9	do.	5/10
Cost of water	56	4	0	do.	11d.
Hire charge on drilling plant	57	4	9	do.	11d.
Repairs to drilling plant and compressor £48/9/2 + £15/0/3	63	9	5	do.	1/-
Preparation for diamond drilling	51	10	4	do.	10d.
1249 ft. 11 in. of drilling cost	£1924	3	3	Total cost per foot	30/9

13. GENERAL CAMP COSTS.

a. Mess Organisation.

The atmosphere of the camp was most satisfactory. This was due in large measure to the care exercised in the matter of providing adequate cooking and mess facilities. It was decided to adopt what is known as "shearers' mess" conditions under which the men are responsible for ordering and paying for their own food-stuffs, the details being left to a committee appointed by the men. The Project supplied the personnel for the cook house free of charge to the men, and also assisted the mess committee by doing everything possible to ensure that the food ordered in Adelaide was promptly collected and forwarded to Copley.

The mess was fortunate in being able to obtain fresh supplies of meat from Arkaroola Station until about the middle of February 1945. The local drought conditions then became so severe that Arkaroola was unable to supply any more sheep fit for killing. Thereafter until the completion of the job, sheep were obtained from Wooltana Station.

The rates of pay for the cooks and cooks' offsiders were fixed by the South Australian Government Industrial Officer in accordance with A.W.U. award rates under which a cook's pay is £5/9/- per week of 44 hours with overtime at the rate of time and a half. After studying the conditions on the job, the Industrial Officer varied the rate to a flat rate of £9/10/- per week, no overtime and at a later date reduced this flat rate to £8 per week, no overtime. The cost of cooks and cooks' offsiders, was £1197/4/2. At East Painter the cost was £324/13/3.

b. Supervision at the Mines.

At the commencement of operations great difficulty was experienced in obtaining a suitably trained man to act as mine manager at Mount Painter. The Manager of the Zinc Corporation at Broken Hill, however, came to the assistance of the Project by making available, young, well trained officers on loan from his own staff. The cost of supervision consisting of Manager and foreman was as follows:-

At Mount Painter	£622/1/1
At East Painter	£143/15/5

Timekeeping costs were:-

At Mount Painter	£404/7/0
At East Painter	£120/5/-

c. Surveying.

The plans of the No. 6 workings at Mount Painter and all geological plans of that locality were prepared by the Assistant Government Geologist. Similar plans of the East Painter area were prepared by the geological and geophysical staff camped at East Painter.

The surface plans showing the layout of works and buildings at Mount Painter and East Painter were prepared by a licensed surveyor from the Lands and Survey Department who also carried out the field work and prepared the plan of the trigonometrical survey (Plan No. 2942) of the whole of the Mount Painter area.

The cost of surveying excluding Geologists' salaries was £444/18/10 of which £402/9/- was incurred by the Lands and Survey Department, chiefly in connection with the trigonometrical survey of the district.

d. Supervision at Copley.

It was necessary to establish at Copley a receiving and forwarding depot to handle the business of the Project into and out of that railway station and to transmit telephoned instructions and messages to and from the mines.

To house the depot clerk the Egnineering and Water Supply Department made available a suitable building 34' x 17'.

The cost of supervision at Copley was £395/16/1.

e. Holidays and Travelling Time.

As mentioned elsewhere a large proportion of the personnel employed on the Project at Mount Painter and East Painter were soldiers released from the Army for this work. On account of its urgency the men were required to proceed to the job immediately upon release. This gave most of them little or no opportunity to visit their homes after their return from the fighting forces.

Under the circumstances the men, who so desired were permitted to visit their homes during the Christmas vacation period. Arrangements were made for them to join a special train from Marree on 20/12/44 and to return by special train leaving Adelaide on 10/1/45. Employees taking advantage of this opportunity were reimbursed the cost of their railway fares upon production of receipts.

The cost of travelling time to and from the mine at the commencement and end of the job and during the vacation period was £248/12/3 for Mt. Painter and £30/0/0 for East Painter.

14. MISCELLANEOUS COSTS.

Other costs not dealt with in detail in the foregoing sections are:-

Item	Labour			Material			Total		
	£	s.	d	£	s.	d	£	s.	d
Mt. Painter Camp Maintenance	263	17	3	162	5	10	426	3	1
East Painter " "	35	17	7	30	11	11	66	9	6
Mt. Painter Sampling	62	0	5	36	8	10	98	9	3
East Painter "	7	6	0	1	12	0	8	18	0
Mt. Painter Prospecting	21	5	0	6	8	11	27	13	11
East Painter "	70	11	5	4	6	9	74	18	2
Mt. Painter Road Repairs	73	14	7	-	-	-	73	14	7
East Painter " "	16	8	3	-	-	-	16	8	3
Mt. Painter Erecting Tents	4	4	10	-	-	-	4	4	10
East Painter " "	13	16	8	-	-	-	13	16	8
Mt. Painter Underground Maintce.	137	1	0	-	-	-	137	1	0
Mt. Painter Dismantling Plant	232	10	1	-	-	-	232	10	1
East Painter " "	43	9	4	-	-	-	43	9	4
Mt. Painter Erecting Saw Bench	8	0	1	-	-	-	8	0	1
Mt. Painter Repairs to Gates and Fences	6	12	0	-	-	-	6	12	0
East Painter Clearing Camp Site	10	12	9	-	-	-	10	12	9
East Painter Costeaning	81	3	3	12	0	3	93	3	6
East Painter Repairs to Telephone	1	6	2	-	-	-	1	6	2
East Painter Open Cutting	73	6	9	12	0	3	85	7	0
East Painter Camel Pad	24	13	1	-	-	-	24	13	1

15. DISPOSAL OF PLANT AND EQUIPMENT.

All field work connected with the Project ceased on 17/9/45 on which date the remaining personnel left Copley per motor lorries with the last loads of materials for Adelaide. Prior to that date all plant and equipment had been carted to Copley for disposal.

Some of the South Australian Government Departments were prepared to purchase part of the surplus plant and stores under the conditions laid down by the Commonwealth Government Disposals Commission. The chief buyers were the Engineering and Water Supply Department, the Mines Department, the Trust Fund for Commonwealth Assistance for Metalliferous Mining.

The unsold balance is stored pending a decision on the extent of future operations. Further purchases are likely to be made by Government Departments.

The final result will be included in the financial statement to be prepared by the Accountant on completion of accounts.

16. ACKNOWLEDGEMENTS.

The Mount Painter Project was initiated at a time when skilled workers and mining equipment were not obtainable through ordinary channels because of war conditions. Its successful establishment at short notice and in almost record time, was due in large part to the co-operation and assistance received from many sources.

Very many acknowledgements are due. A few are mentioned here.

The Department of Supply and Shipping in Melbourne for assistance throughout.

The Engineer-in-Chief of the Engineering and Water Supply Department for constructing the roads and assisting in many other ways.

The General Manager of the Zinc Corporation, Broken Hill for providing trained officers and also equipment.

The Director of Mines, Alice Springs for much equipment.

The Director of Explosives Supply and his managers of the munitions factories at Finsbury and Penfield for personnel and other assistance.

The Railways Commissioner and the Chief Engineer for S.A. Railways for much assistance.

The Royal Australian Air Force Training Section for supplying aeroplanes and conducting many inspection flights during the progress of the work.

The Flying Doctor at Broken Hill for his readiness to fly to Balcanoona if and when required.

The Regional Manager of the Commonwealth Disposals Commission for much help.

The Superintendent of Telephones in the Post Master General's Department for help in the matter of telephone communication with the mines.

The local station owners for placing their telephone lines at the service of the Project.

The Army for releasing men and materials and its helpful attitude generally.

PART III - GEOLOGICAL SURVEY REPORTS.

I. URANIUM MINERALS FROM MOUNT PAINTER*

by

(CAPT.) F. L. STILLWELL, D.Sc., and A.B.EDWARDS, D.Sc., Ph.D., D.I.C.

Mineragraphic Investigations
Council for Scientific and Industrial Research
Geology School, University of Melbourne
Victoria

....

The following observations on the uranium minerals of the Mt. Painter district are based upon investigations of a collection of specimens gathered from the outcrops and workings of the various "prospects" in the Mt. Painter district, supplemented by specimens supplied by the Mines Department of South Australia, and specimens of ore from the No. 6 workings in the museums of the Geology Departments of the Universities of Adelaide and Melbourne.

The various "prospects" on the uranium field include:

Mt. Painter Area

Nos. 1, 2 and 3 workings on Radium Ridge.
No. 4 working on a southerly spur of Mt. Gee.
No. 5 working on a lower slope of Mt. Gee.
No. 6 working on a spur south-east of the camp, and east of Radium Creek.
No. 7 workings and Minerva Heights workings, on a spur west of Radium Creek and west of No. 6 workings.

East Painter Area

"Bentley Greenwood" prospect.
"Smiler Greenwood" prospect.
No. 1 adit, East Painter, north-west of camp, east side of Heighty Creek.
No. 2 prospect, north of camp.
No. 3 prospect, about 500 ft. west of Heighty Creek.
No. 5 prospect and workings, north-west of camp.
Prospect one mile due west of East Painter camp, on an outcrop running for two miles straight towards Mt. Painter (not visited).

The localities of these various prospects are shown in the plans accompanying other reports in this Bulletin.

Previous publications dealing with the mineralogy of the district have been listed elsewhere, and will be referred to here only as required by occasions in the text.

1. OCCURRENCE

The uranium minerals occur in, or near the margin of, an area of granitic rocks, frequently in brecciated zones that often coincide with the outcrops of small bodies of ironstone. The

---

* An account of the nature and occurrence of these minerals was issued as Mineragraphic Report No. 315, of the Council for Scientific and Industrial Research. This present account is a revision and enlargement of Report No. 315, following upon a second visit to Mt. Painter.

granite approaches a pegmatite in composition, and consists essentially of quartz and reddish microcline. Desseminated ferromagnesian minerals are typically absent, but bands and schlieren of biotite rock, or biotite-garnet rock, presumably related to the granite are found at short intervals. Mawson (1944, p.356) records the occurrence of small grains of a black radioactive mineral in the granite at a point about one-quarter of a mile south-west of Smiler Greenwood's prospect.

Near to the Mt. Painter Camp, the sedimentary schists and gneisses marginal to the granite have been extensively granitized, and all stages can be traced in thin sections from biotite schist and gneiss to rocks of apparently igneous character, some resembling felspar porphyries, others pegmatic granite. This transition is clearly seen in the diamond drill cores from the No. 6 workings.

At east Painter, also, the main prospects occur in a region of brecciated and granitized sedimentary schists and gneisses.

Along the spur leading from Radium Creek to the prospects on Radium Ridge, well within the main granite area, there are numerous outcrops of large residual bodies or xenoliths of schist and gneiss, dispersed through the prevailing pegmatitic granite. Similar bodies of schist and gneiss were noted along the ridges between Mt. Gee Creek and the Smiler Greenwood prospect, and also along the slopes and spurs of Mt. Gee. This appears to be a characteristic feature of the granite area, so much so that the impression is gained that the peculiar pegmatitic granite represents an area in which there has been extensive replacement of the schists and gneisses by the magmatic fluids.

A feature of the granitised rocks, apart from the introduction of quartz and felspar, is the alteration of the biotite of the schists which has accompanied the formation of tourmaline and other minerals. The brown biotite is altered first to a paler-coloured, high polarising biotite. It then passes sometimes into a pale chlorite, sometimes to near-colourless mica, distinguishable from muscovite only by a pale yellowish-green pleochroism, and finally to a colourless muscovite. In all cases, the change is accompanied by a discharge of iron oxide, sometimes dispersed as fine particles through the biotite, sometimes aggregated around the margins. When recrystallisation has occurred, blades of hematite appear. The later migration and accumulation of this iron probably contributes to the formation of the ironstone deposits. Sometimes the alteration of the biotite is accompanied by the development of a brownish, highly refracting mineral similar to sphene, and at other times needles of rutile are formed.

At times the crystallization of felspar and quartz do not appear entirely contemporaneous, and quartz has the appearance of invading and enveloping the felspar. When the quartz predominates, silicification of the schists occurs with the formation of a quartzite-like rock.

The ironstone deposits are in no sence gossans derived from the oxidation of sulphides. They were formed after the introduction of the felspar into the granitized sediments because the iron ore minerals are generally found replacing the felspars along cleavages and fractures. Some of the iron ore developed originally as magnetite, and pseudomorphs of hematite after magnetite, often with some residual magnetite in the core, are to be found. Much of this ironstone has a granular texture, which remains after oxidation and weathering, giving rise to massive outcrops of hard ironstone, but where the ironstone contains a considerable proportion of unreplaced and often kaolinized felspar, it is softer, and somewhat greyish. At the No. 6 workings, where the ironstone appears to taper downwards, the thinning or lensing out is accompanied by an increase in the amount of unreplaced felspar.

When this felspar is altered to kaolin or related clays, the ironstone has the appearance of being more severely weathered in depth than at the surface.

The uranium minerals have been found chiefly about the margins of a wide area, which is intensely impregnated with vughy quartz, centring about Mt. Gee. The quartz is of epithermal character, and occurs characteristically as clusters of radiating crystals (Fig. 1), which frequently project into vughs, and have been deposited frequently in columnar fashion on rod-like crystals of gypsum. At the surface the gypsum has generally been redissolved, leaving a prismatic cavity ("nail-hole") in the quartz (fig. 2). In places, as near the base of Mt. Gee, north of the No. 4 prospect, the gypsum is found apparently in place, in the "nail holes". The possibility that the gypsum was introduced as a secondary filling in the holes cannot be altogether excluded, but it seems unlikely, because in some specimens of the "nail-hole" quartz, the prismatic cavities end in arrowhead re-entrants, such as characterise many gypsum crystals (Figs. 3 and 4). Moreover, there can be no doubt that the original mineral must have been a common mineral, in view of the abundance of the "nail holes", and it must also have been relatively soluble under atmospheric conditions.

A little calcite is encrusted on the quartz and gypsum at the locality north of No. 4 prospect, and there has been some tendency to regard it as marking the point of issue of a hot spring. This seems unlikely in view of the fact that the greatest development of the epithermal quartz is above these outcrops, and there is no concentration of the "nail hole" quartz downstream from the so-called spring sites. It seems more probable that these outcrops are simply localities where stream erosion has exposed quartz, from which the gypsum needles about which it crystallized have not been removed.

In places, the gypsum and quartz show alternating deposition, a thin layer or layers of fine "nail hole" quartz encrusting a thicker, coarser-grained layer of similar quartz.

The greatest thickness of this quartz is at Mt. Gee, where it occurs throughout several hundred feet vertical height. Away from Mt. Gee, the epithermal quartz is less strongly developed, and does not appear to extend far below the tops of the present ridges. Much secondary quartz is found in some of the prospects, showing features in thin section identical with undoubted epithermal quartz, while not apparent as such in the hand specimen.

East from Mt. Gee, close to the site of the temporary camp on Mt. Gee Creek, the epithermal quartz occurs intimately intergrown with a small lode of fluorite. Both green and purple fluorite occur, in zoned crystals, the purple fluorite occurring generally in zones in contact with the epithermal quartz. Barite veins also occur in the epithermal quartz belt, at several widely spaced points, such as Radium Ridge, and east of Mt. Gee, on the track to "Smiler's" East Painter prospect. A small amount of barium is present in the No. 6 Workings (Mining Review, Department of Mines, S.A., No. 19, 1914, p. 12).

There is a genetic relationship between much of the hematite occurring in the ironstone outcrops and the epithermal quartz, as is evidenced by their intimate intergrowth. This relationship is clearly seen at the No. 2 Workings, on Radium Ridge, where coarse-grained hematite is associated in part with quartz with a concentric structure enclosing vughs lined with quartz crystals and sometimes containing uranium minerals. Here hematite of two types is found. Intergrown with abundant monazite is hematite which is partly martite. Traces of magnetite in hematite are still to be observed in polished sections, and much of the hematite shows a granular structure suggestive of the form of original magnetite crystals. Associated with this hematite is a considerable amount of bladed or specular hematite, which is intimately intergrown with the epithermal quartz, and in occasional specimens hematite occurs as perfect crystals encrusting quartz crystals in vughs. It seems probable from this that the iron was originally present as magnetite, but was subsequently oxidised to hematite, possibly through the action of the epithermal siliceous solutions that deposited the quartz. Part of the iron was redissolved by these solutions, and reprecipitated as specular hematite.

The fact that the uranium prospects are associated with inronstone outcrops (which also contain monazite) may be explained as due to the fact that these localities originally marked bodies containing magnetite and monazite, probably with primary uranium minerals. These rocks were, in some instances, fractured and brecciated, and then infiltrated by the epithermal quartz solutions which dissolved some of the iron oxides re-precipitating them as hematite and at the same time cementing the breccia into a particularly resistant rock.

2. URANIUM MINERALS.

The uranium minerals so far discovered in the various prospects and lodes are as follows: autunite, torbernite, meta-torbernite, uranophane, gummite or uranium ochre, and an unidentified uranium silicate, and fergusonite. Associated with them are various primary minerals such as hematite, monazite, pyrite and fluorite, and various alteration products and secondary minerals such as psilomelane, goethite, collophane, iron phosphates, limonite and gypsum.

Other uranium minerals recorded as occurring at Mount Painter, but not found during the present investigation, are zeunerite, and carnotite. A specimen labelled zeunerite in the Museum of the Geology Department, Melbourne University, was found to consist of meta-torbernite.

a. Autunite -$CaO.2UO_3.P_2O_5.10H_2O$

Autunite was a prominent constitutent of the rich shoot formerly extracted from the No. 6 workings (fig. 4), but has not been found in the subsequent openings made at this locality, except for a vein about half an inch wide at the south end of the shoote, on the 50 ft. level, opposite the No. 1 winze. It has not been found as yet in any of the other prospects of the district, except for small amounts in vughs at the No. 2 workings on Radium Ridge. The autunite occurred in small seams (Fig. 5) intimately associated with torbernite, colloform manganese and limonitic minerals, as a cement of a breccia. The seams contained numerous vughs and massive lemon-yellow to sulphur-yellow plates, up to 1 cm. across, developed on the walls of the vughs. The autunite has refractive indices between 1.550 and 1.575. It is uniaxial, or very nearly so, and optically negative. In ultra-violet light it fluoresces a vivid emerald green.

An analysis of selected material from the No. 6 workings has been published by Hodge-Smith (1926), and is as follows:-

CaO	6.56
MgO	0.26
U_3O_8	58.85
P_2O_5	14.80
H_2O	19.60
	100.07
Sp.Gr.	3.198

This corresponds to the formula $CaO.2UO_3.P_2O_5.10H_2O$.

The Mt. Painter autunite appears to be isomorphous with the torbernite found there, and compositions intermediate between the two can be found. The presence of copper in the mineral is indicated by the appearance of a greenish tint, which becomes more intense as the copper content rises. This transition has been observed in a single platy crystal, the core consisting of yellow autunite, passing to cupriferous autunite at the margin, and has been confirmed by micro-chemical tests for calcium and copper. The transition is marked by an increase in the refractive index of the autunite. Rinne (1901) describes a similar relationship.

It is noticeable at Mt. Painter that the autunite is always slightly earlier than the associated cupriferous autunite or torbernite, the latter occurring in the centre of vughs, while the autunite encrusts their margins.

Torbernite ($CuO.2UO_3.P_2O_5.12H_2O$) and Meta-Torbernite ($CuO.P_2O_5.8H_2O$)

Torbernite, or its dehydration product, meta-torbernite, is the most prevalent of the uranium minerals in the Mt. Painter prospects, being present in all of them. Commonly it is the most abundant, and sometimes the only, uranium mineral, but in the No. 2 workings on Radium Ridge, it is present only as occasional flakes. An analysis of a hand-picked sample from the No. 6 working made by T. W. Dalwood, of the South Australian Mines Department, is as follows:-

SiO_2	1.24%
Al_2O_3	0.38
Fe_2O_3	0.38
CaO	0.06
UO_3	58.02
CuO	8.09
P_2O_5	14.91
H_2O	17.04
	100.12

This analysis corresponds to a formula $CuO.2UO_3.P_2O_5 + 9\frac{1}{2}H_2O$, and indicates that the sample contained torbernite and meta-torbernite in the proportions of 3 to 5.

The two forms of torbernite are generally found in close associating in hand specimens. Neither of them normally fluoresces, so that where fluorescence is detected it indicates the presence of calcium or some other uranium salts (Meixner, 1930) on the surface of the torbernite. A distinction between the two varieties of torbernite can only be made under the microscope. The torbernite has refractive indices between 1.58 to 1.59, and is strongly pleochroic from green to colourless. It has a good cleavage parallel to the platy faces of the crystals, and cleavage flakes appear almost isotropic, although they give a uniaxial figure of negative character. Flakes at right angles to the cleavage show blue polarization colours.

The meta-torbernite has a higher refractive index, between 1.62 and 1.63, and cleavage fragments are practically isotropic. It also has a slightly higher specific gravity. The meta-torbernite shows a weak uniaxial figure on cleavage plates, and is optically positive. Polarisation colours of fragments at righ-angles to the cleavage are a distinctive purple, a little bluer in colour than the colour of a gypsum plate.

The relationship of torbernite to meta-torbernite has been studied by Rinne (1901), Buchholz (1903) and Hallimond (1920).

Rinne showed that torbernite readily transforms to meta-torbernite, losing 4 molecules of water, on heating in air to 60°C.-65°C. Buchholz showed that the change could be brought about at room temperatures by keeping torbernite in a dessicator with concentrated sulphuric acid as a dehydrating reagent, for about 60 hours. Again the loss was equivalent to 4 molecules of water. Hallimond demonstrated that the torbernite could be converted to meta-torbernite by heating to above 75°C. when immersed in water. The change appears to be irreversible.

Mt. Painter torbernite, when stood on a water-bath for one hour, immersed in distilled water, was found to have altered to meta-torbernite. When placed in a desiccator, over sulphuric acid, the change was completed overnight (20 hours) at ordinary room temperatures.

These facts suggest that (1) the torbernite was formed at low temperatures, less than 60°C., and (2) that in the hot arid region where it occurs it may undergo dehydration to meta-torbernite in surface rocks. Some of the torbernite appears to be in an intermediate stage of transition to meta-torbernite, and shows anomalously high refractive indices. There is also the possibility that some of the meta-torbernite was originally deposited as such by solutions at temperatures above 65°-75°C.

The torbernite and meta-torbernite commonly occur as flakes along the joint planes and occasionally as small veins. It also occurs in open spaces or vughs lined with goethite and psilomelane. Where it occurs in cavities, the torbernite plates often grow across the open space but do not completely fill it, and smaller plates grow outwards from the main platy crystals, forming a cellular structure (Fig. 6).

In new exposures in the No. 6 workings, it was noticed that plates of torbernite commonly occurred as a "paint" on the surface of fragments of silicified gneiss (quartzite) embedded in the hematite breccia. They also occurred along fractures in the siliceous rock, but rarely in the matrix of the breccia. Apparently the surfaces of contact were a factor in influencing deposition.

In the No. 3, No. 7 and Minerva Heights working, the meta-torbernite occurs in vughs lined with epithermal quartz. An isolated instance of such an association is stated by R. G. Thomas to have been found on the slopes of Mt. Gee, and specimens from two prospects in the East Painter area, one on a bold outcrop of epithermal quartz about 1 mile west of East Painter Camp, and the other about three-quarters of a mile north of Bentley Greenwood's prospect, show a similar association. In the specimens from the former of these two prospects the quartz shows "nail-holes".

In some specimens from the Bentley Greenwood prospect, the torbernite occurs in small areas 2 to 3 mm. in width, dispersed through the ironstone. These small areas frequently show euhedral outlines, though consisting of aggregates of meta-torbernite, and they may represent an earlier mineral, possibly pyrite, that has been replaced. One section of ore from this prospect contains a vein, about 2 mm. wide, of scondary quartz, intergrown in part with meta-torbernite, radiating prisms of goethite and a little manganese oxide. Another section from this ore contains meta-torbernite filling fractures through relatively fresh microcline.

The occurrence of flakes of torbernite more or less filling small pits, 2 to 3 mm. across, is also characteristic of ore from the No. 1 and No. 2 workings at East Painter. The pits are dispersed through massive hematite ironstone and are sometimes filled with limonite, sometimes with torbernite and or lepidocrocite, and sometimes with a boxwork of secondary minerals pseudomorphous after the torbernite. The original filling of the pits would appear to have been pyrite. At the No. 2 workings, East Painter, a specimen was obtained in which the torbernite crystals occurred in the small vughs of a jasperoid quartz vein, about 3 mm. wide, traversing such pitted ironstone.

A thin section from the No. 2 workings on Radium Ridge (Mt. Painter Camp area) also shows meta-torbernite intergrown with secondary quartz, indicating more or less contemporaneous deposition.

c. Uranophane - $CaO.2UO_3.2SiO_3.7H_2O$

Uranophane has been found at the No. 2 and No. 6 workings. At the No. 2 Workings, it is present occasionally as rosettes of yellow radiating prisms, seldom more than 1 mm. long, resting upon the crystalline faces of quartz projecting into vughs. The prisms are mostly bright yellow, but are sometimes brownish yellow, particularly at their tips.

Under the microscope, the thin acicular crystals are pleochroic from bright yellow to colourless. Some prisms show low anomalous colours. The crystals are biaxial and negative, and have an extinction angle of 4°-5° in the prism axis. The lowest refractive index is approximately 1.645. These characters identify the mineral as uranophane; and when tested with HCl, the mineral dissolves readily in the cold, forming a solution which yields a strong positive microchemical test for calcium. The mineral also fluoresces green, and appears to be slightly radioactive.

In thin sections of ore from the No. 2 workings, a number of rosettes and bundles of exceedingly thin, yellow, pleochroic (?) needles up to 0.1 mm. long, have been observed embedded in quartz of evidently secondary origin. These needles are presumed to be uranophane and, if so, indicate its contemporaneity with the secondary quartz.

At the No. 6 Workings, the uranophane occurs as an apparent alteration product of weathered torbernite, as fine needles impregnating the bleached torbernite, and as rosettes and clusters of minute needles coating adjacent joints and cleavage planes. This specimen is strongly radioactive, and the uranophane fluoresces a vivid green.

d. Hydrous Uranium Silicate.

A specimen of quartz and hematite preserved in the Museum of the South Australian Mines Department from the No. 2 Workings at Radium Ridge contains a yellow mineral which, though labelled (?) gummite, is a yellow uranium silicate.

This mineral is yellow and non-flourescent, and forms radiating fibrous masses in vughs that are lined with projecting quartz crystals. When prised free, the mineral is distinctly fibrous and can be teased like asbestos fibre. Mounted in oils, it appears sulphur yellow and is weakly pleochroic yellow to pale yellow. Its polarization colours are low, being chiefly blues. The refractive index is close to 1.635, being slightly less in one direction. It has straight extinction, but the optical character could not be determined. It dissolves with difficulty in acids and leaves an insoluble residue of silica. The solution gives negative tests for lead and copper. Iron appears to be present. A spectroscopic examination at the Maribyrnong Defence Laboratory determined the major elements to be uranium and silica, with strong traces of lead and barium.

The mineral somewhat resembles uranophane, but cannot at present be assigned to any known species of uranium silicate. It differs from uranophane in its relative insolubility and in its slightly different refractive index.

e. Gummite (chiefly uranium oxide)

At the No. 2 Workings, Radium Ridge, small areas of a yellow ochreous mineral, which fluoresces green and is moderately radioactive, are found coating cavities. The mineral is crystalline, occurring as minute plates and prisms. Mounted in oils, the prisms show pleochroism from lemon yellow to nearly colourless, and are birefringent with low polarisation colours. Extinction on the occasional prisms appears to be straight. Crystal plates are practically isotropic, but are too small to yield an interference figure. The refractive index of the plates is greater than 1.740, and the birefrigence is very low.

The mineral is readily soluble in cold nitric acid without effervescence, and yields good positive microchemical tests for uranium, lead and phosphorus, but gives no test for calcium, copper or sulphate. It was thought at first that the lead test was strong enough to indicate the presence of lead as a major constituent of the mineral, and the mineral was provisionally reported as a lead uranium phosphate, possibly dewindite (Mineragraphic Report No. 315).

Subsequently a hand-picked sample, amounting to about 0.4 grams, was analysed by Mr. F. D. Drews, of the Melbourne Ore Dressing Laboratory, with the results shown below in Column No. 1.

	1.	2.	3.	4.
UO_3	79.9	91.9	-	77.95
UO_2*	-	-	90.7	-
PbO	2.4	2.8	st.tr.	5.28
P_2O_5	0.9	1.0	st.tr.	n.d.
H_2O^+	3.9	4.5	4.9	8.90
Insol. etc.	13.1	-	3.9	8.73
	100.2	100.2	99.5	100.86

* Uranium calculated as UO_2 to give a reasonable total.

1. Gummite, No. 2 Workings, Radium Ridge, Mt. Painter.
2. Same, recalculated free of insoluble.
3. Gummite or uranium ochre, No. 2 Workings, Radium Ridge.
4. Gummite, from Spruce Pine, North Carolina, for comparison (Ross, Henderson and Posnjak, 1931).

This analysis indicates that the analysed mineral is probably related to the mineral gummite, which seems to have a somewhat variable composition. The phosphorus percentage is sufficient to suggest that the analysed specimen is a mixture. In the absence of calcium and copper, it may be assumed that the impurity is not autunite or torbernite. The phosphorus is either combined with uranium as a uranyl phosphate like phosphuranylite $(UO_2)_3.P_2O_8.6H_2O$, or with lead and uranium, as a lead uranium phosphate, like dewindite $3PbO.5UO_3.2P_2O_5.12H_2O$. Dewindite from Chinkolobwe in the Belgian Congo contains 24.85% PbO and 10.14% P_2O_5, (Schoep, 1925), which is approximately the proportions in which PbO and P_2O_5 occur in the mineral under discussion. There is thus some suggestion that the analysed mineral is a mixture of uranium oxide and dewindite.

This view is strengthened by the fact that a second sample of about 0.4 grams, analysed by Mr. Drews, showed only traces of lead and phosphorous, although both could be detected with microchemical tests. The analysis of this sample is shown in Column 3 above. The abundance of uranium oxide in the sample was such that it was necessary to calculate it as UO_2 rather than UO_3 to obtain a total approximating to 100%.

If it be assumed that the phosphous is combined only with the uranium, then the marked differences in lead content of the two samples is difficult to explain, excpet by assuming that they were formed at widly differing times.

A closely related uranium oxide mineral containing lead and phosphorus in detectable amounts occurred in small amount at the No. 6 Workings, and was referred to by W. S. Chapman as gummite. Specimens from the Mines Department Museum, South Australia, show that it occurred as clots of a yellow ochreous character, without definite crystal form, and intimately associated with torbernite and autunite. Some of the torbernite occurs in small rings within the gummite.

This mineral differs from the gummite of Radium Ridge in that it fails to fluoresce, and is not readily soluble in nitric acid, when it yields good positive tests for lead and phosphorus. Mounted in oils it resembles the gummite in that it shows faint pleochroism in yellows, and has a low birefringence, with a refractive index above 1.740. It lacks the tendency towards crystal form shown by the Radium Ridge mineral. Unfortunately, it is too intimately associated with torbernite to obtain a sufficient quantity of the pure mineral for quantitative chemical analysis. The lead may be a breakdown product or an impurity. It may be remarked that no trace of lead could be obtained by microchemical tests on the adjacent torbernite and autunite.

Fergusonite

Fergusonite (so-called) has been found in specimens from the No. 2 Workings, where it occurs as scattered blebs, averaging about 5 to 6 mm. across, of a dark, moderately hard mineral, with a brilliant vitreous to submetallic lustre on fractured surfaces. Internal reflections give it a suggestion of reddish-brown colour. Its occurrence has also been reported at the Exploration lode, north-east of Mt. Painter (Mining Review, S.A., No. 45, 1927).

The identity of this mineral is based on the following chemical analysis by Chapman, quoted by H.Y.L. Brown (1911), column 1. A second analysis has been made by Kleeman (Mawson, 1944) column 2.

	1	2
SiO_2	6.26	-
TiO_2	-	0.43
Al_2O_3	2.99	0.46
Iron Oxide (calc. as FeO)	19.41	21.75
MnO	0.48	0.26
MgO	0.13	Sl.tr.
CaO	4.32	1.50
ThO_2	tr.	Sl.tr.
Ca_2O_3	1.76	0.29
La_2O_3 etc.	-	0.12
Y_2O_3 etc.	7.08	3.75
Nb_2O_5) Ta_2O_5)	48.08	51.11
U_3O_8	1.20	8.64
PbO	-	0.66
SnO_2	tr.	tr.
P_2O_5	-	-
H_2O^+)	6.44	4.57
H_2O^-)		3.22
	98.15	98.76

1. Fergusonite from Radium Ridge. Analysed by W.S. Chapman. Sample selected as free as possible from associated siliceous matter.

2. Fergusonite from Radium Ridge. Analysed by A.W. Kleeman. Re-calculated free from SiO_2.

The silica and alumina in Chapman's analysis indicate the presence of impurities in the analysed material. The analysis shows marked differences from fergusonite proper (as described in Dana system of Mineralogy), because there is too much iron and too little yttrium, erbium and cerium. It does not agree either with the analyses of other hydrous niobates (and tantalates) such as yttrotantalite or samarskite, which contain more iron than is present in the Mt. Painter mineral. Kleeman's analysis shows considerably less yttrium and much more uranium, but is otherwise similar. The difficulties attending such an analysis are great, and until further chemical data are available, the mineral cannot be determined accurately. Meanwhile, the practice of referring to it as fergusonite is continued.

The identity of the fergusonite cannot be established by its appearance in either thin section or polished section, and depends primarily on its appearance in the hand specimen. Neither in thin or polished section does it appear homogeneous. In thin section it consists of central isolated opaque areas surrounded by reddish-brown translucent areas. In polished sections it takes a good polish, and the central portions are greyish-white in colour, while the translucent areas of the thin section are, by comparison, darker grey. The darker grey areas sometimes follow cracks or surround inclusions, and are so intimately associated with the "islands" of lighter grey color, that they are suggestive of an alteration phase. Brownish-red internal reflections can be seen which are much more intense in the marginal areas. It is clearly anisotropic, and it is unattached by standard etching reagents, though it can be etched by cold HF, or by a mixture of HF and H_2SO_4. The reflective power of the fergusonite is low, but is a little higher than that of magnetite. Occasional inclusions of hematite, magnetite and quartz have been observed in the fergusonite. It is not strongly radioactive, and does not fluoresce.

3. ASSOCIATED MINERALS.

The pegmatitic granite or its modifications which contain the uranium minerals consists essentially of quartz and microcline, sometimes associated with pale mica. Other pegmatitic minerals include monazite, tourmaline, fluorite, occasional apatite and rutile. Psilomelane is associated with the abundant hematite as well as a little magnetite, pyrite and occasional traces of chalcopyrite. Goethite is sometimes prominently associated with psilomelane.

Within the ironstained material associated with the hematite outcrops, microcline frequently shows considerable replacement by iron oxide with a probable separation of quartz. It is uncertain, however, how far this is due to simple weathering.

Mica is more prominent in the Smiler Greenwood and Bentley Greenwood prospects at East Painter than in the other prospects nearer the main camp. This mica is pleochroic from pale green to colourless and is distinguished from torbernite by its bright polarisation colours. A spectroscopic examination of a crushed sample, rich in mica, failed to reveal more than a trace of vanadium.

Monazite generally occurs as occasional crystals up to 1 mm. long scattered sparsely through all the prospects and in pegmatitic veins associated with them. It is, however, abundantly developed in places at the No.2 workings, where it occurs associated with martite and fergusonite. Here it is pinkish-brown in colour from the presence of iron oxide along fractures and cleavage, and some hand specimens consist of perhaps 30% monazite, in association with subordinate quartz, scattered fergusonite and abundant hematite. An analysis of the monazite from this locality has been made by R.G. Thomas, and is quoted by Mawson (1944).

Tourmaline and apatite occur in some abundance in several thin sections of the granitized gneiss of the diamond drill cores obtained from the No. 6 workings. The tourmaline is pleochroic from greyish-brown to pink, and tends to form intergrowths with the quartz, having evidently been introduced during the process of granitization (pegmatitization). Grains of apatite are sometimes 1 mm. long.

Small grains of purple fluorite have been observed in specimens from the East Painter prospects and also in ore from No. 3 and No. 7 workings. It is reported by G.A. Greenwood to be a frequent associate of torbernite in the various prospects throughout the district.

Rutile has been identified somewhat doubtfully from sections of the Bentley Greenwood prospect in minute yellowish-green crystals, but observations on a polished surface appear to confirm it.

In one specimen from the No. 3 workings, torbernite is associated with a zeolitic mineral, which has low birefrigence and a refractive index between 1.525 and 1.530. It dissolves in acid and yields a strong positive test for calcium, suggesting that it is a lime zeolite such as thomsonite.

In many instances, the abundant hematite contains traces of residual magnetite. Magnetite inclusions have appeared, however, in more than traces in one section from the No. 5 workings where magnetite sometimes forms more than half the hematite crystal, even though such crystals have the bladed form of hematite. Minute crystals of pyrite and, less often, chalcopyrite, also occur as inclusions in hematite.

Pyrite also occurs in scattered crystals, up to 3 mm. in width, throughout the hematite at the No. 4 workings. It is sometimes coated with an orange-yellow encrustation consisting of lepidocrocite, the dimorphous form of goethite. In some specimens, the pyrite is completely leached and the cavities are filled with lepidocrocite, sometimes associated with traces of gypsum. Well-formed pseudomorphs of limonite after pyrite are plentiful in specimens from the Smiler Greenwood prospect at East Painter.

Psilomelane occurs at most of the prospects, but is most strongly developed at the No. 6 workings and No. 7 workings (Fig.6.) At these localities and at Minerva Heights it forms botryoidal encrustations and seams cementing together fragments of breccia. A polished section of the manganese oxide at the No. 6 workings shows fragments of quartz, hematite and limonite comented by seams of psilomelane, which shows a banded colloform structure parallel to the margin of the rock fragments. The marginal layers of psilomelane contain numerous thin, columnar, anisotropic prisms of pyrolusite, set at right-angles to the walls of the vein. Succeeding bands are less dehydrated, containing only minute prisms of pyrolusite. The central portion of one seam is occupied by a layer of green torbernite.

Goethite or Needle-ironstone ($Fe_2O_3.H_2O$)has been detected in the Nos. 2,6,7 Minerva Heights at Mt.Painter camp, and at the Bentley Greenwood and No.5 prospects at East Painter. The best specimens come from the No. 7 workings, where goethite forms part of the cement in a zone of brecciated granite or pegmatite (Fig. 7). It occurs as a layer, 2-3 mm. wide, of fine columnar, dark-brown prisms on the surface of the brecciated fragments, and is encrusted with a layer of about the same thickness of psilomelane. The occurrence at Minerva Heights is similar. At the outcrop of the No.

5 prospect, East Painter, torbernite occurs in cavities lined with stalactitic goethite, in a quartzite breccia.

Goethite is easily distinguished from the psilomelane by its brown colour. Under the microscope it is a transparent brown, weakly pleochroic from deep brown to yellow-brown, with straight extinction. It has a refractive index well above 1.740. When crushed, it yields an ochreous yellow powder, soluble in hot HCl. The solution gives an overwhelmingly strong test for iron, a weak test for phosphate, and a negative test for manganese.

Goethite has been observed in thin sections from the No. 6 workings as small clusters of radiating brownish-yellow prisms which project from areas of hematite and manganese oxide. A similar occurrence was observed in thin sections from the No. 2 workings, where the goethite prisms fringe areas of hematite intergrown with quartz. In Bentley Greenwood's prospect, the goethite has been detected in a vein of secondary quartz through granitic rock, and numerous small prisms of goethite project from the walls of the vein into the quartz, which is associated with meta-torbernite and manganese oxide. In the No. 5 workings at East Painter, the goethite forms small stalactites projecting into vughs.

Lepidocrocite (Rubinglimmer) ($Fe_2O_3 \cdot H_2O$), a dimorphous form of goethite, occurs as an alteration product of pyrite at the No. 4 workings. It occurs as an orange-yellow powder with adamantine lustre, consisting of minute crystalline plates which are transparent, have a refractive index well above 1.74, and are doubly refracting. Its colour and appearance simulate that of the uranium ochreous minerals, and it has been distinguished by its form and by a partial chemical analysis on 0.02 gm., which gave 85.1% Fe_2O_3. Similar lepidocrocite occurs at the No. 1 and No. 2 prospects at East Painter, where it is sometimes coated with torbernite.

4. WEATHERING OF TORBERNITE.

Torbernite is frequently resistant to weathering and may occur in the soil as "floaters" in the neighbourhood of outcrops. Nevertheless, specimens of torbernite obtained within a few feet of the surface from a wall of the open cut at the No. 6 workings, show that under the conditions of weathering prevailing at that locality, copper and uranium are being leached out of the torbernite, leaving a residue of calcium phosphate (? collophane). The uranium is being redeposited in the immediate vicinity of the torbernite crystals as the calcium uranium silicate uranophane. The copper is partly reprecipitated as copper oxide in limonite associated with the uranophane and is partly removed in solution from the immediate vicinity.

The torbernite occurs as rectangular platy crystals, which occasionally have dimensions as large as 5.0 x 10.0 mm. The leaching out of the copper and uranium proceeds from the margin inwards, giving rise to a white marginal zone of calcium phosphate enclosing an irregular core of partly altered or unaltered torbernite (Fig. 8).

The leached white margins of the plates are commonly impregnated with microscopic needles of uranophane, distinguishable under the microscope at high powers by their form and optical properties. Occasional fragments of the white rims were obtained, however, which appeared to be almost free from impurities. The material so obtained was colourless and isotropic, with a refractive index of 1.575 ± 0.002. It is slowly soluble in nitric and hydrochloric acids, and yields strong positive tests for calcium and phosphorus, and weaker tests for copper and iron. These tests, in conjunction with the optical properties, suggest that the material is the calcium phosphate, collophane. This identification must be accepted with reserve, since the calcium may have been derived from needles of uranophane included in the material tested, while the phosphorus might conceivably come from included or completely altered torbernite.

Surrounding the torbernite crystals are areas up to several centrimetres across, which are coated with films of brown, cupriferous limonite and minute rosettes of acicular needles of uranophane. The uranophane is readily identified by its needle-like form, its sulphur-yellow colour, strong pleochroism and anisotropism, and refractive index. It gives a strong positive test for calcium, and fluoresces a yellowish-green under ultra-voilet light. Much of the uranophane has developed on the surface of the cupriferous limonite. This limonite, when partly dissolved in acids, yields moderately strong tests for copper and iron. In places, the fragments tested showed colloform banding.

It seems likely that the alteration of the torbernite was caused by surface solutions containing calcium sulphate and silica. The copper and uranium of the torbernite appear to have been replaced by calcium and moved away in the form of copper and uranium sulphates. The phosphate of the torbernite remained in place. The uranium was reprecipitated in conjunction with calcium and silica as uranophane almost immediately, and some of the copper was thrown down as oxide intermixed with iron oxide. Part of the copper may, however, have escaped in solution, probably as copper sulphate. The absence of copper carbonates or other carbonate minerals indicates that the waters causing the leaching were free from carbonate, and there are no adjacent rocks which could be regarded as a likely source of chlorine. Sulphates might be expected, however, in view of the liklihood of the previous existence of abundant gypsum associated with the epithermal quartz deposit centred about Mt. Gee, and from the presence of occasional pyrite crystals in the uranium ores. The unusual nature of this alteration may be due partly to the presence of lime in the torbernite and partly due to its situation. Water slowly percolating down the wall or joint plane would subject this torbernite to a more sustained attack than torbernite exposed at the surface or in the soil of this arid region.

At Smiler Greenwood's prospect in East Painter, the torbernite is largely replaced by a brownish mineral that encrusts the torbernite and the surrounding rock, possibly filling spaces from which torbernite has been leached. The torbernite is chiefly meta-torbernite, its peculiar blue polarisation colours indicating only partial dehydration. The mineral replacing it gives strong positive tests for phosphorus and iron, but no copper or uranium, and only a little calcium.

5. BOXWORK STRUCTURES.

At a number of the outcrops, particularly in the East Painter section of the field, the occurrence of torbernite is associated with distinctive boxwork structures (Fig. 9). The torbernite in these prospects commonly occurs as a filling in small cavities, presumably voids formed by the solution of pyrite crystals. The torbernite frequently occurs in the form of a boxwork, and in many instances, where the torbernite has been leached away, a boxwork structure remains which appears to be diagnostic of the former presence of torbernite.

These boxworks are formed of a variety of mineral substances, including calcium phosphate, iron phasphate, gypsum, with or without siliceous material, and sometimes limonitic material. It is not certain whether these substances represent a weathering product of the original torbernite, or were deposited as coatings on plates of torbernite, arranged in a boxwork, and subsequently, leached out, leaving the more insoluble coating substance to form a pseudomorphic boxwork structure after the original torbernite. Thus, in the boxworks figured (Fig. 9), some flakes of torbernite remain, sandwiched between the plates that form the boxwork, suggesting the second mode of origin. Where the boxwork consists of a phosphate mineral, there is a suggestion that the phosphate is residual from original torbernite, the copper and uranium of which have departed in the solutions that brought in the iron or lime of the phosphate mineral.

In the boxwork figured, the cavities between the boxwork plates have been filled with lepidocrocite, and in a number of other specimens, chiefly from the No. 2 workings at East Painter, the cavities containing torbernite are also encrusted with "pustular" lepidocrocite. In some instances, the torbernite plates wrap round a small body of limonite or lepidocrocite. Other cavities are coated with a film of bluish substance which appears to be vivianite. In one specimen which contains no boxworks and only a trace of torbernite, there are a number of limonite pseudomorphs after small idiomorphic crystals of pyrite, and where these pseudomorphs have fallen out of the surface the resulting cavities closely resemble those in which the torbernite occurs.

Crystals of pyrite, about 1 to 2 mm. across, only partly replaced by ledidocrocite, were found in another specimen. Numerous cavities were also present from which the pyrite had been completely leached, leaving only a encrustation of lepidocrocite. A few of these cavities contained a little boxwork suggestive of the former presence of torbernite, while in two other specimens of massive hematite from the same locality (No. 2 prospect, East Painter), a number of cavities whose form suggests the previous presence of pyrite crystals were lined or partly filled with torbernite.

It is apparent from these specimens that they previously contained rather more torbernite than is now visible, and the presence of boxworks without torbernite can be regarded as evidence of previous presence of torbernite. Cavities with only an encrustation of lepidocrocite (limonite) occurring as minute globules or pustules indicate only the former presence of pyrite.

6. ORIGIN OF THE URANIUM MINERALS.

With only one exception (possibly two), the uranium-bearing minerals so far discovered at Mt. Painter are undoubtedly secondary minerals. The evidence for this is partly their mode of occurrence, and partly their character.

The torbernite occurs at almost every prospect as thin plates along joint planes. It occurs in vughs and open spaces that are coated with encrustations, sometimes stalactitic, of goethite and psilomelane (Figs. 6, 7). It is sometimes intergrown with psilomelane, and is occasionally found in vughs encrusting projecting crystals of quartz. Elsewhere it is found as a "paint" of thin flakes lining fractures and joints.

Autunite, which is more restricted in its distribution than torbernite, occurs at the No. 6 workings in veins associated with psilomelane; it also occurs in small amounts in vughs at the No. 2 workings, Radium Ridge.

Uranophane is found encrusting joint planes as an alteration product of torbernite, and possibly autunite, at the No. 6 workings, and at the No. 2 workings, Radium Ridge, it rests on projecting quartz crystals in vughs.

The gummite at the No. 2 workings also occurs as encrustation in cavities, and that at the No. 6 workings is intimately associated with torbernite.

It is evident, moreover, that these uranium minerals must have been deposited at low temperature from watery solutions. Not only are they all hydrated minerals, but torbernite is a low temperature mineral that inverts a meta-torbernite at 60° to 65°C. The dehydration is readily brought about at room temperature (20°C.) in a dessicator over sulphuric acid, and it takes place under water at 75°C. In view of the ease with which the change can be brought about, it is thought that the frequent occurrence of meta-torbernite, in exposed or surface specimens from Mt. Painter, is due to dehydration in the dry hot climate that prevails there in the summer.

The fact that much of the torbernite remains unchanged, particularly in specimens from the No. 6 underground workings, indicates that the minerals were deposited from cold or only warm water solutions. The fact that the torbernite crystals only partly fill some cavities, while extending right across them, is further proof of the watery character of the solutions from which it was deposited, as is the stalactitic character on some of the associated goethite.

All these facts establish that the uranium minerals are secondary minerals, deposited from supergene solutions. Certain features of the deposits suggest, however, that they are not altogether the simple oxidation products of primary uranium minerals.

The occurrence of the torbernite in vughs in epithermal quartz proves that some, if not all, of it was deposited after or during the deposition of the epithermal quartz which is so abundant in the district. At the Bentley Greenwood prospect, and at the No. 2 prospect at East Painter, specimens were obtained which were traversed by veinlets of secondary quartz containing goethite, and meta-torbernite. Another specimen from the Bentley Greenwood prospect in thin section showed meta-torbernite filling fractures through relatively fresh microcline. These occurrences suggest that the torbernite and the epithermal quartz were deposited from the same solutions. The occurrence of minute rosettes of presumed uranophane embedded in quartz at the No. 2 workings, Radium Ridge, suggests a similar relation for this mineral.

A second feature opposed to the interpretation of the torbernite as a simple product of oxidation is the fact that torbernite, though a relatively insoluble mineral, does not resist attack by supergene waters, and in one instance at least such attack has led to the separation of the three chief constituents of the torbernite, namely, uranium, copper and phosphorus. The occurrence of torbernite weathering in this way at the No. 6 workings may be exceptional, but it suggests that surface solutions could not be expected to bring about a concentration of these elements, such as was necessary to form an ore shoot, unless they were previously of a different composition from those now in existence.

Thirdly, it is a feature of the specimens of the rich No. 6 ore, preserved in the various museum collections examined, that the autunite was always deposited before the torbernite. If the deposition was from surface waters only, it might be expected that some variation in the proportions of lime and copper associated with the phosphorus and uranium would occur from time to time, leading to occasional reversals of this state of affairs, but, so far as can be ascertained, this did not occur.

Fourthly, the impression is gained that the uranium prospects bear a spatial relationship to the wide area of epithermal quartz centring about Mt. Gee, and have a pronounced tendency to occur in favourable structures at points around the margin of this area.

These various features suggest as an alternative explanation of the origin of these secondary uranium deposits that the uranium was leached from primary uranium minerals associated with the pegmatitic granite by the epithermal solutions that gave rise to the wide- spread "nailhole" quartz, and that as these solutions migrated outwards from their point of origin, depositing their silica content, they transported the uranium to favourable structures about the margin of the area of quartz deposition.

7. PRIMARY MINERALS.

The only uranium-bearing mineral yet discovered on the field that is clearly a primary mineral is the so-called fergusonite at the No.2 workings, Radium Ridge. Tantalum minerals of this general character are primary products of pegmatites, and so is the abundant monazite with which the fergusonite is associated. Whether the hematite with them is also primary is uncertain. Some of it probably is, because some of it contains minute inclusions of magnetite, and has a crystal form suggestive of original magnetite grains. It is thus a normal alteration of magnetite, or it may have been converted to hematite by the action of the epithermal solutions that gave rise to the vughy quartz, so closely associated with it at the No.2 workings. Some of the hematite was recrystallized when the quartz was introduced because fine crystals of hematite form encrustations in the centre of some of the small vughs in the quartz.

It seems unlikely, however, that the fergusonite is the only or the chief primary uranium mineral, because it appears so stable, and so little altered in the presence of the secondary uranium silicates and phosphates. Moreover, there are oxidised patches in the No.2 ore which are strongly radioactive, and which consist of limonite and ochreous minerals, including gummite, which evidently represent the breakdown products of a pre-existing mineral. Further, Mawson (1944) records the presence of small grains of a strongly radioactive black mineral dispersed through the pegmatitic granite, and Broughton (1925) records the occurrence at one locality of radioactive ilmenite containing up to 3.7% U_3O_8, though this could not be confirmed during the present survey.

Another reason for thinking that there are, or were, as yet undiscovered primary uranium minerals such as uraninite is the varying proportion of lead in the secondary uranium minerals. If this lead is simply the breakdown product of the uranium in these secondary minerals, formed since their formation, then the lead-uranium ratio should be constant for all the minerals if they were formed contemporaneously. Actually the lead-uranium ratios vary within wide limits.

The one primary mineral, the fergusonite, from Kleeman's analysis (quoted on p.14) contains 8.64% U_3O_8 and 0.66% PbO. A further partial analysis by Kleeman (pers. comm.) on another carefully purified sample, gave 10.6% U_3O_8 and 0.72% PbO. Assuming that the lead is all derived from the radioactive disintegration of the uranium, this suggests that the primary mineralisation occurred during the early Palaeozoic or the late Pre-Cambrian.

Lead is not readily detectable in the torbernite and autunite by microchemical means, despite the sensitivity of such tests, which are capable of detecting lead in concentrations as small as 0.0027 gm. Pb per 100 c.c., i.e., about 3 micrograms of Pb in a drop of 0.1 c.c. volume. When, however, samples of 0.25 gm. of torbernite and autunite from the No.6 Workings were dissolved in 1 c.c. of conc. HNO_3, evaporate to dryness and taken up in 1 c.c. water, they yielded a faint positive test for lead iodide, but on diluting to 2 c.c. no positive test could be obtained. This indicates that the Pb content of the torbernite and autunite is of the order of 0.01%. So low a lead content indicates that these minerals are of late Cainozoic age.

Quantitative determinations of the lead content of two selected specimens of torbernite, one with associated autunite from the No.6 Workings at Mt. Painter, the other from East Painter, were made by T.W. Dalwood, with the following results:-

Mt. Painter specimen - Uranium 44.75%, lead not detected (less than 0.0001%);

East Painter specimen - Uranium 46.76%, lead 0.045%.

These results indicate not only that the torbernite is geologically young, but also that the torbernite in different deposits and even within a particular deposit has been formed at different times.

An analysed sample of "gummite" from the No.2 Workings at Radium Ridge, carrying 92% UO_3, contained 2.6% Pb, whereas another analysed specimen from the same locality, which contained 90% UO_2, yielded only a trace of Pb, though sufficient to detect in the course of an ordinary microchemical test on a few grains of the substance. Gummite from the No.6 Workings gives an equally strong test for lead when a few minute grains are dissolved, whereas a comparable test with torbernite intimately associated with it gives a negative result.

The high lead content of the one analysed specimen of gummite from the No.2 Workings may be caused by intermixed lead uranium phosphate, dewindite. Even so, dewindite itself is a secondary mineral, which could not have been derived from the apparently lead-free autunite and torbernite, and must therefore have been derived either from a primary uranium mineral, or the lead must have come from some other lead mineral, such as galena, of which no trace has been found.

8. CONCLUSION.

The conclusion arrived at is, therefore, that during the early Palaeozoic or the Pre-Cambrian one or more primary uranium minerals, such as uraninite, was introduced into the area, along with fergusonite and monazite, at the time of the intrusion of the pegmatitic granite, partly as a dissemination through the pegmatitic granite, and partly perhaps in local concentrations.

Subsequently, these primary minerals were subjected to weathering, and reconcentration as secondary minerals, chiefly phosphates, either by surface waters or possibly in conjunction with the introduction of the siliceous calcium sulphate waters that gave rise to the low-temperature, epithermal gypsum-quartz deposits centering on Mt. Gee. The phosphorus was probably obtained from the same general source in view of the presence of primary phosphate minerals, monazite and apatite, in the pegmatitic granite, and in the granitized schists. Some barium, fluorine and manganese were presumably present in the waters, and were deposited locally as barite and fluorite, generally associated with the epithermal quartz, the fluorite often in small amounts with the phosphates. The copper may have been derived from the chalcopyrite associated with the pre-existing magnetite, and where locally the uranium failed, the copper was deposited as turquoise.

The siliceous solutions may represent the final residuum of the pegmatitic magma or they may be much younger solutions, developing after a long period of erosion had lowered the surface of the land to about the level of the tops of the present ridges. In the first case, the association of the epithermal quartz with the secondary uranium minerals is fortuitous, and the minerals are of simple supergene origin. If, however, the secondary uranium minerals were re-precipitated from the siliceous solutions, the solutions must be young on account of the low lead content of the torbernite and autunite. This is in keeping with the epithermal character of the quartz which otherwise could scarcely have escaped erosion in the long period since the early Palaeozoic or the Pre-Cambrian.

The variable lead-uranium ratio shown by different specimens of torbernite indicates that the torbernite in different deposits, and even within a single deposit, has been formed at different times, so that at most, only a part of the torbernite could have been deposited from the epithermal solutions. The remainder of the torbernite (and perhaps all of it) must be attributed to the work of supergene solutions.

The localised concentration of the secondary uranium minerals suggests that they were not moved far from their primary source; otherwise a wider dissemination through the area of epithermal quartz would be expected. The possibility cannot be overlooked that the present ore-bodies are below the base of the original primary ore. It is perhaps significant that the only rich ore-body so far discovered consisted dominantly of autunite, whereas all other prospects are characterised by the more widely disseminated torbernite.

9. LIST OF REFERENCES.

Broughton, A.C. 1925. Radioactive Ilmenite, near Mount Painter, Northern Flinders Range. Trans. Roy. Soc. S.Aust., vol. 49, p.101;

Broughton, A.C. 1926. Ibid., vol. 50, p.315.

Brown, H.Y.L. 1911. Occurrence of Uranium and other Rare Minerals near Mt. Painter in the Flinders Range of South Australia. Mines Dept., S.Aust.

Buchholz, Y. 1903. Der Wassergehalt des Kupferuranite, Centralblatt Min., pp. 362-365.

Hallimond, A.F. 1920. Meta-torbernite I, Its Physical Properties and Relation to Torbernite, Mineral Mag., vol. 19, pp. 43-47.

Hodge-Smith, T. 1926. Mineralogical Notes, No.2, Rec. Aust. Mus., vol. 15, p.74.

Mawson, D. 1944. The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia. Trans. Roy. Soc. S.Aust., vol. 68, pp. 348-356.

Meixner, H. 1939. Fluoreszenz untersuchungen in sekundären Uran-mineralien. Naturwiss, 1939, vol. 27, p.454; Chemie der Erde, 1940, 12, 433-50; Min. Abstracts, vol. 7, 1940, p.529.

Rinne, F. 1901. Kupferuranit und seine Entwasserungsprodukte (Meta-kupferuranit). Centralblatt Min. pp. 618-626.

Ross, C.S., E.P. Henderson, and E. Posjnak, 1931. Clarkeite; A New Uranium Mineral. Amer.Min., vol. 16, pp. 213-220.

Schoep, A. 1925. Nouvelles Recherches sur la Dewindite. Bull. Soc. Franc. Min., vol.48, pp. 77-85. Min.Abstracts, vol. 3. p.6.

10. LIST OF ILLUSTRATIONS.

Fig.1. Cluster of "nail-hole" quartz growths showing diamond-shaped cross sections of "nail-holes". Radium Ridge. X 2/3.

Fig.2. "Nail-hole" quartz, showing radial growth about central prismatic cavity, with "arrowhead" re-entrant termination suggestive of gypsum prisms. Radium Ridge. Natural size.

Fig.3. Typical "nail-hole" quartz cluster showing diamond-shaped cross sections of "nail-holes". Radium Ridge. X ¾.

Fig.4. Side view of above specimen. X ¾.

Fig.5A. Seam of autunite showing autunite crystals projecting into a vugh. Sample from original shoot, No.6 Workings. Natural size.

Fig.5B. Top surface of above. X 2/3.

Fig.6. Platy crystals of torbernite projecting in a vugh lined with psilomelane. A "paint" of torbernite crystals occur on a joint surface in the upper half of the photograph. No.6 workings. X 1½.

Fig.7. Goethite which occurs intimately associated with torbernite. No.7 Workings. X 8.

Fig.8. Weathered torbernite showing white rim of calcium phosphate surrounding a core of little altered torbernite. From an exposed wall in the No.6 surface workings.

Fig.9. Boxwork structure after torbernite. The torbernite occurred as plates across a void. These plates are replaced by a non-phosphate mineral, while lepidocrocite and limonite have been deposited in the interstices. X 10.

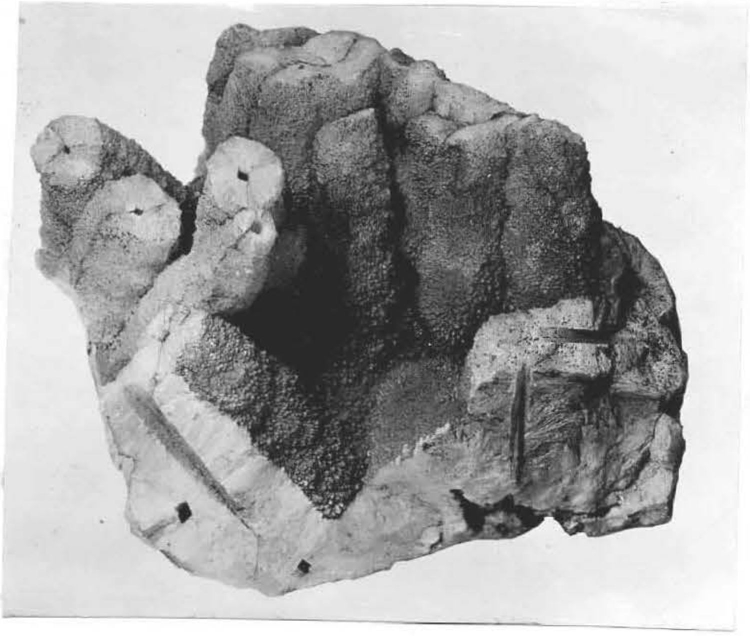


Fig. 1 Cluster of "nail-hole" quartz growths, showing diamond-shaped cross sections of "nail-holes". Radium Ridge. X 2/3

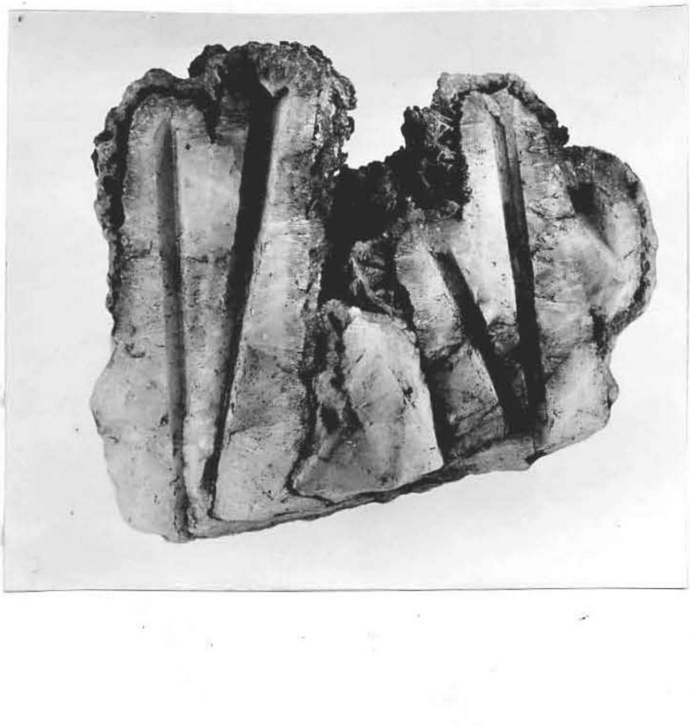


Fig. 2. "Nail-hole" quartz, showing radial growth about central prismatic cavity, with "arrowhead" re-entrant termination suggestive of gypsum prisms. Radium Ridge. Natural size.

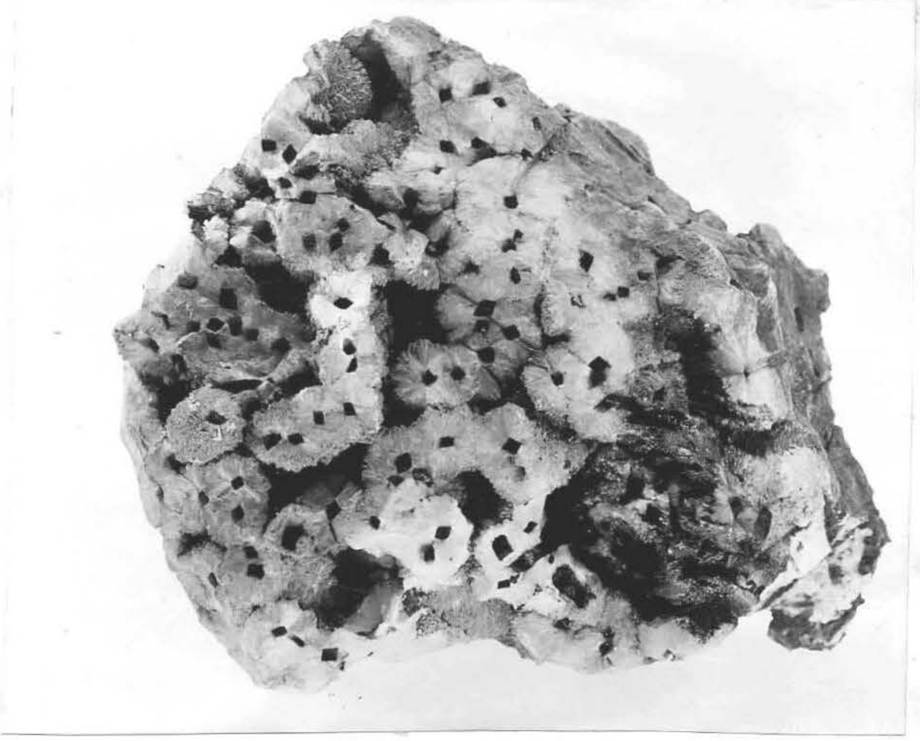


Fig. 3. Typical "nail-hole" quartz cluster showing diamond-shaped cross sections of "nailholes". Radium Ridge. X $\frac{3}{4}$

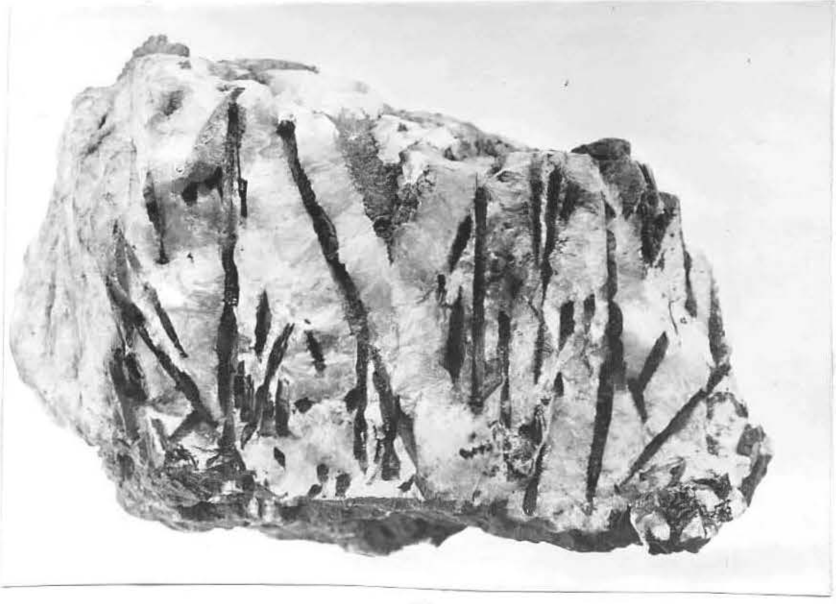


Fig. 4. Side view of above specimen. X $\frac{3}{4}$

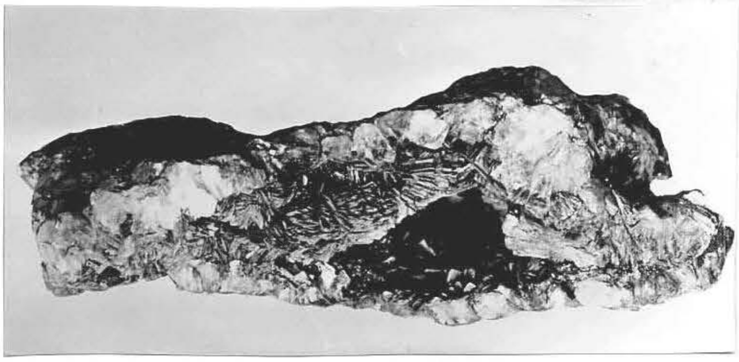


Fig. 5A Seam of autunite, showing crystals of autunite projecting into a vugh. Sample from original shoot, No. 6 workings. Natural size.

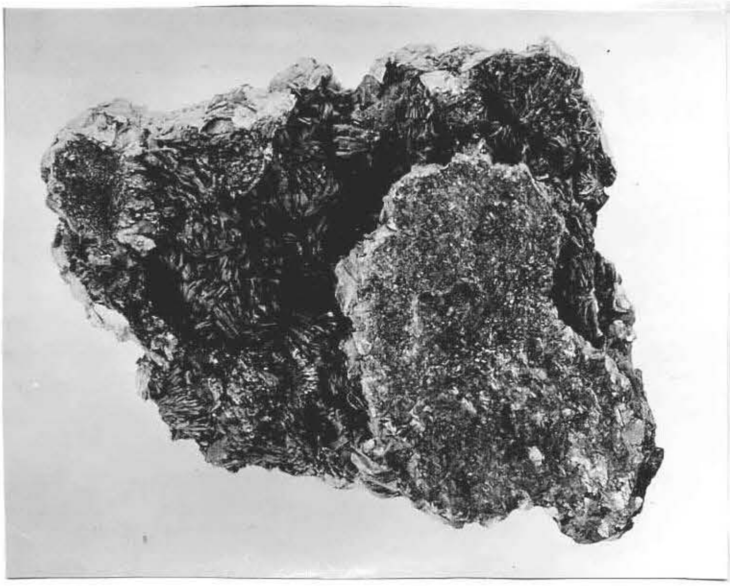


Fig. 5B. Top surface of above. X 2/3

Fig. 6 Platy crystals of torbernite projecting into a vugh lined with psilomelane (opening).
A "paint" of torbernite crystals occurs on a joint surface in the upper half of the photo. X $1\frac{1}{2}$.
Specimen from No. 6 workings.

Fig. 7. Goethite which occurs intimately associated with torbernite. No. 7 workings. X 8.

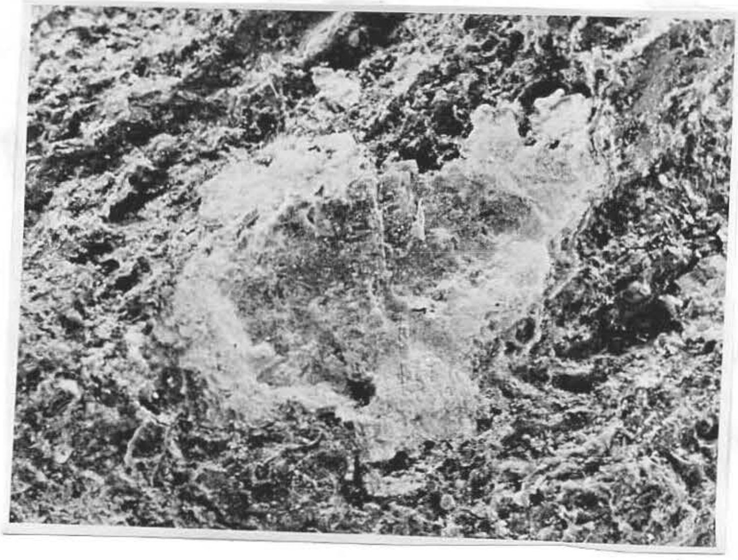


Fig. 8. Weathered torbernite showing white rim of calcium phosphate surrounding a core of little altered torbernite. No. 6 workings at surface. X 8.

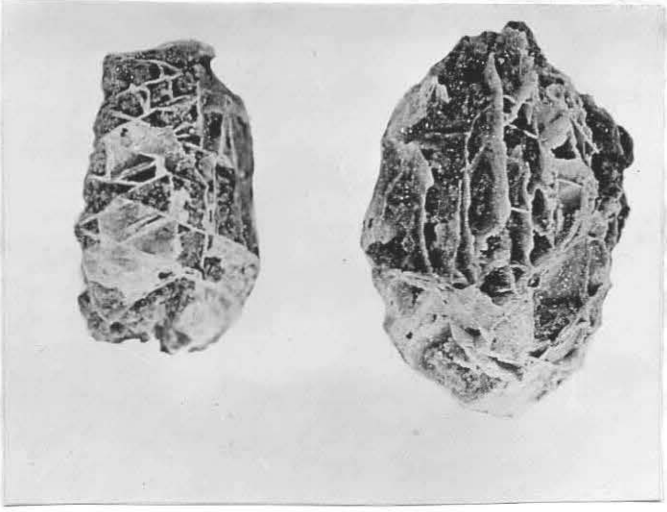


Fig. 9. Boxwork structure after torbernite. The torbernite occurred as plates across a void. These plates are now replaced by an iron phosphate mineral, while lepidocrocite and limonite have been deposited in the interstices. X 10.

11. RECONNAISSANCE GEOLOGICAL SURVEY OF THE MOUNT PAINTER AREA

by

Reg. C. Sprigg

Geological Survey of South Australia
Department of Mines, Adelaide.

---

1. PLANS.

The following plans accompany this report:-

Mount Painter Area Landform Analysis.	Plan 3210,	Sheet No.43.
River Capture by Yudnamutana Creek.	Plan 3287,	Sheet No.44.
Mt. Painter Area Geological Reconnaissance Maps.	Plan 3288,	Sheet No.45a.
	Plan 3289,	Sheet No.45b.
	Plan 3290,	Sheet No.45c.
	Plan 3291,	Sheet No.45d.
N-S Section Across 45a and c.	Plan 3292,	Sheet No.46a.
N.E. Flinders Ranges; sketch Geological Reconnaissance Map and N-S Section.	Plan 3294,	Sheet No.47.

2. INTRODUCTION.

Prior to 1944 the Mount Painter locality had not been examined systematically. The terrain is extremely rugged and until recently was only traversed on foot with pack camels.

Copper, uranium, corundum, gold and bismuth had each been mined on a small scale within the area and the records of the nature and extent of the various deposits were very fragmentary. The several geologists who had visited the region had collected as much information concerning the rock series, mineral occurrences and geological structure as time and the limited transport facilities would allow.

The 1944-45 search for uranium minerals made it necessary to undertake a more detailed methodical geological survey over an area of approximately 140 square miles.

3. THE ORGANIZATION AND SCOPE OF THE WORK.

Field work was carried out between October 19 and December 18, 1944.

From aerial photographs taken by the R.A.A.F., Port Pirie a provisional photomap in 4 sheets was compiled by the Army Survey Directorate, Melbourne, on a scale of approximately 1,200 feet to one inch as a basis for mapping and location of roads. It was prepared without reference to ground survey stations by the Australian Army Survey Directorate. The photomap and prints of the photographs were used for guidance in the field and for the recording of geological data.

Two parties, under the direction of C. J. Sullivan and R. C. Sprigg respectively, were formed and worked independently in the field. Sullivan, assisted by D. E. Gardner, undertook the geological mapping of the eastern half of the area and Sprigg, assisted by H. J. Ward, mapped the western portion.

In addition to the reconnaissance work Sullivan and Gardner carried out detailed mapping using enlarged aerial photographs (approximate scale = 600 feet to an inch) of an area of 3½ square miles in the vicinity of Greenwood's Camp where a number of large ironstone outcrops were situated.

The objectives of the reconnaissance work were threefold:

1. To delineate the area containing known uraniferous deposits and those likely to contain other deposits.
2. To map any large scale geological features which might form loci for possible uranium deposits and to make reasonably certain that, in so far as reconnaissance work permitted, no large uranium deposits were overlooked.
3. To note the general geological features of the area.

The results of the reconnaissance survey are presented as four geological plans (with section) on a scale of 20 chains to one inch (Plans 3288, 3289, 3290, 3291, 3292, Sheets 45a, 45b, 45c, 45d, 46a). The plans were prepared by Mr. A. C. Booth of the C'wealth Dept. of Interior whose report on the compilation methods used appears elsewhere in this bulletin. Geological information and interpretations were marked on aerial photographs and copied together with the topographic detail by Mr. Booth on to the final plans. Considerably more detail was collected than is reproduced on the plans. A number of geological boundaries were difficult to delineate on account of the complexity of mineral associations and variety of rock types, but these difficulties are described fully in the text.

To relate the more complex geological features of the Mount Painter region with those of the surrounding areas lying chiefly to the south and west, a geological reconnaissance sketch plan (Plan 3294) was prepared from geological traverses at widespread intervals. Additional information was obtained from aerial photographs taken at 20,000 feet altitude by the Royal Australian Air Force Trimetrograph Survey Corps.

4. CLIMATE, RAINFALL, VEGETATION AND FAUNA.

The climate is semi-arid with a wide annual range in temperature and a low rainfall less than 10 inches. Pastoralists generally rely on winter rains for good stock-raising results but summer monsoonal disturbances are not uncommon. Such a disturbance occurred in February, 1944, when 16-18 inches were recorded at Mt. Fitton in 24 hours and it caused great damage to fences and to stock.

Local landholders rely on bores for stock water. Adequate supplies of groundwater have been developed chiefly in the Pre-Cambrian sedimentary series flanking the Mount Painter complex but difficulty has been experienced in developing supplies in the granitic rocks at Mount Painter.

The vegetation is sparce. On the plains to the east of Mount Painter the principal vegetation for stock consists of Mitchell grass (Astrebla sp), stipa, salt bush (Atriplex Vesicarium) and blue bush (Kochia Sedifolia). The major watercourses are lined with large red gum trees (Eucalyptus canaldulensis). Away from the watercourses, timber is sparse and generally unsuitable for mining purposes. The eucalypts have stunted mallee habit and the pines (Callitris sp.) irregularly distributed over most of the hilly region also have a stunted appearance. Certain vegetation is limited to distinctive rock horizons, for example, Eucalyptus gillei which is found chiefly on soils overlying calcareous beds, and is thus a useful guide in mapping. Quondong (Eucaria acuminata) trees occur along some of the principal creeks of the area and provide food for birds and marsupials.

The Euro (macropus robustus) is plentiful in the ranges, but wallabies are scarce. On the plains the large red and blue kangaroos are abundant. Rabbits were very scarce during the survey period, but have been known to reach plague dimensions in the past.

Until recently wild camels and donkeys (both introduced) were plentiful in the ranges. The last wild camel was captured during the survey and brought into service at Mount Painter. Bird-life is generally restricted on account of the absence of permanent waters.

5. PHYSIOGRAPHY.

The Mount Painter area occurs in the Flinders Ranges which have a general north-south trend and are bounded on the east by the Lake Frome, and on the west by the Lake Torrens-Lake Eyre depressions. The terrain is deeply dissected and portion of the eastern escarpment of the range is included in the mapped area. Immense outwash plains extend eastwards from this escarpment to Lake Frome which is a remnant of a former but much more extensive lake. The lake is now little more than an extensive salt pan with its bed only 160 feet above sea level and several stranded terraces indicate its former margin much nearer to the Ranges. In and adjacent to the Ranges drainage lines are well defined.

Through rugged gorges the great "gum creeks" meander out onto the plains where they pass out into systems of distributaries near the edge of the Lake Frome. The widths of many of these creeks range up to half a mile and after heavy rains they frequently are unable to cope with the huge volumes of water accumulated in them from the rapid run-off from the ranges. There are no perennial streams in the region, even the large waterholes along the main watercourses dry up after long dry spells. The terrain in the immediate vicinity of Mount Painter is deeply dissected and, being situated near the "divide" between the Lake Frome and Lake Eyre drainage basins, marks an advanced stage in the rapid erosion of the elevated peneplain which initially formed the greater proportion of the Ranges.

The Freeling Heights north of Mt. Painter and including the highest eminence (3,120 feet) in the northern Flinders Ranges, are notable for a flat summit plateau which is probably a remnant of this former very extensive erosion surface or peneplain (Mawson 1923).

Photo. No. 2176

FIG. 1.

It stands at about 3,000 feet above sea level and conforms generally with the summit levels of the Yudnamutana Range, Mount Pitts (2,804 ft.), Mount Painter (2,850 ft.) and Yudnamutana Hill. Further south, beyond the area mapped, portion of the Flinders Ranges known as the Gammon Ranges and other resistant masses of quartzite suggest that this former plateau level continued some distance south of the Mount Painter area.

The 'divide' between the respective watersheds of the Lakes Frome and Eyre basins is located within a few miles of the eastern escarpment but relatively far from the northern limits of the uplifted block on the Lake Eyre aspect, and as a consequence the gradients of the east-flowing streams are relatively steep and stream erosion is powerful. The principal water courses draining eastward comprise the Yudnamutana, Arkaroola and Hot Springs Creeks and in them rapids and rock bars are prominent features. The headwaters of the Yudnamutana Creek are clearly encroaching on the more mature watercourses draining towards Lake Eyre and the westerly migration of the divide is commencing (see Plan 3287, Sheet No.44).

The chief watercourses draining towards Lake Eyre from the Mt. Painter area are the Blue Mine, Yerelina, and Tindalpina Creeks. In contrast the gradients of these streams are much flatter than those draining towards Lake Frome even near their sources. In a few places there are rocky bars, such as near the source of the Yerelina Creek adjacent to the Yudnamutana mine, but these are exceptional. The general terrain west of the Mount Painter area likewise presents a much more subdued form.

In addition to broad topographic features already described there are several minor features which have an important bearing on understanding of the geological history of the region and possibly the age of the uranium deposits. The massive thick quartzite which is co-extensive with the Yudnamutana and Mt. Pitts Ranges is U-shaped with convexity directed westwards. (See Plan 3294, Sheet No.47). These ranges enclose an "amphitheatre-like" basin which is in the process of dissection, but in which the tops of the present hills have a strikingly even surface. These even hill tops situated hundreds of feet below the general elevation of these ranges suggest a second period of peneplanation during the development of the present topographic features. It is interesting to note that the enclosing ranges are composed of quartzite and the "amphitheatre" area is chiefly red granite. East of the Freeling Heights a somewhat analagous subsidiary plateau occurs some hundreds of feet below the level of the Freeling Heights erosion surface which is in quartzites. This lower erosion surface, chiefly on granitic rocks, is separated from the latter by a pronounced escarpment.

South of the Mount Painter area there is evidence of another and still lower erosion surface but its description is not included in this report. In view of a possible bearing of the topographic history on the origin of the uranium deposits an analysis has been made of the survey plans by following the procedure described by W.H. Maze (1944) which has been used with some success to accentuate ancient erosion surfaces.

A. LANDFORM ANALYSIS FROM THE TOPOGRAPHIC MAPS.

In the preparation of the maps of the Mount area the Cartographers of the Department of the Interior interpreted form lines at contour intervals of about 250 ft. These form lines were used in the application of methods of landform analysis outlined and modified by Maze (1944).

The method adopted involved the drawing of a grid (40 chain sides) on the four base-maps. The highest form line value on these base maps for each square in turn was then recorded on a special plan and contours interpolated at 250 feet intervals (see Plan 3210, Sheet 43). Using this plan, the Strip "high point" profiles, the altimetric frequency curve and the "summation" profile were constructed, the latter being an additional method introduced by the author (1945).

The strip "high point" profiles are summary profiles, N-S and E.W, covering 3 grid squares (one and a half miles) in width. The altimetric frequency curve represents a direct count of the respective altitudes recorded in the grid squares. The "summation" curve records the frequency of occurrences of the respective altitude intervals progressively summed and plotted against the altitude in feet.
It is to be noted that nearly all areas above 2,750 feet are underlain by quartzites (see geological maps Plans 3288 - 3291) except near the headwaters of the Yudnamutana Creek which has broken through the quartzite ranges and is actively eroding the relatively soft sediments to the west. It is thus actively capturing westerly-trending watercourses and has created a new local base level.

The strip "high point" profiles indicate 2 major erosion surfaces. The first is above the 2,750 feet level; it represents the "peneplain" or "base" surface of the Freeling Heights and includes the summit levels of Yudnamutana Range and Mts. Pitts and MacDonald. A second erosion surface lies at about 2,000 feet.

Analysis by means of the altimetric frequency curve has indicated a considerable development of the 2,000 feet erosion surface. The summation profibe demonstrates the considerable vertical spread of this erosion surface. Such spread may be due in part to later dissection.

6. THE GEOLOGY.

A. Previous Investigations.

The Mount Painter field embraces a geological and mineralogical province which has not received adequate detailed attention. This is a direct outcome of its remoteness and general inaccessibility. In spite of the difficulties Sir Douglas Mawson has recorded many of its interesting features, and has collected numerous specimens.
Some of the Mineral deposits partly opened up have been examined in more detail and official reports have been presented by Dr. L.K. Ward, Dr. R.L. Jack and Mr. H.Y.L. Brown, chiefly on mine and prospecting workings.

On the evidence of Mawson, the central igneous and metamorphic belt about Mount Painter and north to Mulowurtina previously has been correlated with the Archaeozoic Era. Proterozoic and later sediments were considered to overlay the Archaeozoic core complex unconformably.

B. General.

The 1944 geological reconnaissance survey now indicates that the rocks of the Mount Painter Complex and the surrounding sedimentary series are all to be referred to the Proterozoic and early Palaeozoic eras. Sediments and metasediments within the area surveyed are all recognisable members of the so-called Adelaide Series. Furthermore, there is evidence to suggest that the intrusive red (stressed) and white (unstressed) granites may be correlated with other stressed and unstressed granites intruding the Adelaide Series of sediments elsewhere in South Australia.

The sedimentary series have been folded on a fairly simple plan and both sediments and igneous intrusive rocks have undergone faulting.

The geosynclinal occurrence of Adelaide Series, Cambrian and perhaps Ordovician sediments within South Australia will be referred to as the Flinders geosyncline in distinction from the MacDonnell geosyncline of Central Australia.

C. The Sedimentary Sequence.

The Sedimentary Rocks of the Mount Painter area are all Proterozoic in age. The lowest recognisable horizon is the Thick Quartzite which is an important marker horizon. Where recognised elsewhere in South Australia Mawson (1941) considers that it conveniently represents the base of the Middle Adelaide Series. This massive and resistant quartzite formation 6,000 feet in thickness is responsible for much of the bold scenery of the region. "Erratic" quartzite boulders occur infrequently in some horizons of the quartzite and many measure one foot or more in length. They provide striking correlative evidence in that Madigan (1932) has recorded similar intraformational boulders in the No.2 Quartzite or Thick Quartzite equivalent of the Western Mac Donnell Ranges. In stratigraphical succession there follows a calcareous series which in part is identifiable with the "Blue Metal Series" of dolomitic limestones and magnesite beds occurring at Copley and as far south as Adelaide; near the base of this series there is an arkosic conglomerate of variable thickness. The Sturtian tillite and associated fluvioglacial horizons overlie the calcareous sediments. These glacigene sediments form another marker horizon used in this survey. Very minor fluvioglacial bands occur much higher in the succeeding series of calcareous slates, but they cannot be mistaken for the main tillitic horizon. Dr. Ward (1916, page 8) reported "typical tillite" on the western slope of Mt. Gee. Photographs taken at the time of his visit show a tillite formation traversed by epithermal quartz veins. A search was made for this occurrence, during reconnaissance mapping, but only floaters of tillite were found in Mt. Gee Creek opposite No.5 workings; undoubted tillite must occur in the vicinity. A thick series of calcareous slates and limestones overlie the tillite. Outside the area mapped they pass into the purple series of slates and quartzites with minor limestones horizons. The purple series underlies the Pound sandstone-quartzite of the Gammon Range and this quartzite in turn underlies Cambrian Archaeocyathinae limestones.

No major unconformity has been discovered within the area covered, but to the south and west evidence of thinning and "overlapping of beds" have been recognised.

A stratigraphical sequence has been compiled from the mapping of the area photographed and from information collected while running several traverses south out into relatively undisturbed country bordering the Mount Painter region. It is consistent with results of Mawson's (1941) work on the middle and upper Proterozoic sedimentary series near Copley.

D. Stratigraphical Sequence.

(Thicknesses approximate only).

7,000(+) ft.	The "Pound" Sandstone quartzite of the Gammon Range and elsewhere. Underlies Archaeocyathinae limestones and may represent the base of the Cambrian or close of the Proterozoic sedimentation.
12,000 ft.	Purple Slates Series. Slates and minor limestones and quartzites. Laminated Calcareous Slates and limestones. Equivalent of Tayley Hill Series near Adelaide. Slates contain very minor fluvioglacial bands near Mt. Saturday.
1,000(±) ft.	Sturtian tillite and fluvioglacial Series. This formation is of variable thickness, ranging from 1,000 feet north-west of Mandarin Hill, to 7,000 feet, near Tindalpena where thickening may be due partly to folding. The overlying calcareous slates in the latter locality also show similar thickening. Striated and faceted erratics of very diverse rock types are abundant. Mawson (1911) considers that many erratics are identical with rock types of the central Mt. Painter complex but this may be coincidental, and, in general, does not hold.
11,000 ft.	Calcareous Slates and massive limestones. The limestones in part are dolomitic, magnesitic, tremolitic, and actinolitic. In the lower sections quartzites and an arkosic quartzite of varying thickness figure prominently. Worm-castings (?) and ripple marked surfaces occur in the quartzite near Bolla Bollana.
6,000 ft.	Thick Quartzite formation. Remarkably even-grained sediment except where large interformational boulders occur in the main mass of the rock. Laminations are apparent and ripple marked surfaces are not uncommon.

E. The Igneous Rocks.

Igneous intrusive and extrusive (?) rocks of the Mt. Painter complex will be dealt with in two groups - the acidic and the basic.

Acid igneous rocks outcrop extensively within the Mt. Painter area, and can be subdivided into two categories, the red granites and the white or leuco-granites. It is improbable that these two granites are comagmatic, they both have intruded the Proterozoic Series of sediments.

Within the red granite belt processes of assimilation and/or granitisation are strongly in evidence. All phases from relatively slightly metamorphosed sediments through remnant structure to "normal" granites with little or no directional textures have been recognised. In some areas the hybrid nature of the granitic mass is strikingly apparent. The red granites generally are sheared and fractured and, in outcrop, rounded surfaces and torr-shaped masses characteristic of unstressed granite are absent.

The occurrence of white granites has features clearly distinguishable from those associated with the red granites. The sediments intruded by the white granites are confined entirely to the Thick Quartzite horizon at the base of the sequence. In the case of the white granites in the late stages of intrusion magmatic stoping probably proceeded on a considerable scale. Aerial photos taken from 20,000 feet indicate dykes and sills extending into quartzite rocks along various lines of weakness near the peripbery of the white granite mass. The white granites are more homogeneous in character than the red granites. In weathered outcrop they have typical rounded outlines (Torr structures, etc.) which are outward manifestations of the unstressed character and the presence of few directional textures.

The red granites are believed to be the older of the two. They have undergone a period of stress which the white granites seem to have post-dated or with which the intrusion of the white granites may be contemporary. The association of many signs of granitisation with the red granite "intrusion" suggests a deep seated origin for it, before erosion had removed much of the sedimentary rock. The genetic relationship of the two granite masses require further study before any definite conclusions can be drawn. This study would certainly involve further field work for which the reconnaissance maps and accurate survey plans will provide an excellent basis. The abundance of rock exposures make the Mt. Painter region ideal for such a petrological and structural study.

Concerning the basic igneous suite of rocks the more important occurrences of these have been shown on the small scale sketch reconnaissance map, (Plan 3294), but detailed descriptions of respective areas of outcrop cannot be given. General notes on occurrences are incorporated herein but readers are referred to publications by Mawson (1923, 1926) for additional information on these rocks.

(i) The Red Granite.

Macroscopically the red granite is essentially a medium-grained intergrowth of quartz and pink felspar. For the most part the granite contains only a little mica with schorl and iron minerals commonly present but in small and varying amounts.

The red granite magma is intrusive into the Proterozoic sediments and in the central region particularly west of the "Arm Chair" outcrop relationships between the red granite and the Thick Quartzite suggest a laccolith form, but there has also been some assimilation.

In other localities granitic rocks appear to have been formed in situ, and there is widespread evidence of the granites of vestigeal sedimentary structures which appear to have retained their initial orientations during the granitisation process. The metamorphosed Thick Quartzite formation striking east from Mt. Pitts exhibits increasing metamorphism resulting from processes of granitisation as one proceeds in an easterly direction. The quartzite near Mt. Pitts is micaceous and is intruded by a few pink felspar pegmatites. To the east it becomes gneissic and in many places granitic. Many phases are apparent between gneiss, granite-gneiss and red granite. In this area whatever rock is present the strike of gneissosity (and bedding where sedimentary structures are well-preserved) remains sensibly constant veering a little to the north of east. The dip of structure lines is steep and in most cases in a northerly direction. Inclusions of less altered quartzite are not infrequent. One of these occurs near the junction of Mt. Gee Creek and Radium Creek.

Along the same general line of strike acid gneisses occur in the East Painter area (north of Gardner Gully).

Near the Arkaroola Waterhole massive quartzites inclusions are plainly distinguishable within the granitic rock, but a little to the north difficulty is experienced in distinguishing them in the granitic mass. North of East Painter Gorge also massive quartzites grade imperceptibly into granitic rock. It is not possible to draw boundaries in these areas. For this reason also north of the "Arm Chair" much difficulty was experienced in drawing even a very approximate boundary between obvious quartzites and granitic rock. In addition Pink felspar pegmatites intrude into the quartzites in this locality and render the fixation of finite geological boundaries impossible. For the presentation of a regional map a compromise was decided upon, namely, an arbitrary distinction was drawn between a zone of red granite and definite quartzite sediments, by the definition of a rather ill-defined zone of red granite and quartzite. These boundaries are marked clearly on the geological plans. (Plans 3288 - 3291).

A study of the distribution of the red granite in relation to rock of sedimentary origin makes it apparent that the quartzite resisted granitisation and/or assimilation more strongly than certain other sedimentary types. The Thick Quartzite horizon extends east from Mt. Pitts for some distance into the red granite before it loses its identity. Immediately south of the outcrop of this quartzite and at a relatively higher stratigraphical horizon (e.g. near the Nooldoo Nooldoona Waterhole) metamorphism and intrusion have reached a more advanced stage. Original sedimentary rocks have been converted to a rock which in the field is now referred to as a Quartz-porphyritic-gneissic granite. The gneissic structure conforms with the bedding in adjacent metasediments. Whilst processes of granitisation have altered selectively various sedimentary types their influence has not extended more than about 2,000 feet stratigraphically above the Thick Quartzite formation.

From the central zone in the neighbourhood of Mts. Pitts and Painter, Mawson (1923) refers to "dense felsitic and granitic rocks recalling the Pre-Cambrian porphyry and granite of the Gawler Ranges and Moonta; also a considerable development of syenite-porphyry, quartz felspar porphyry, felspar-porphyry, and acidic gneisses sometimes exhibiting augen structure". Two rock specimens collected in Radium (Sphene) Creek he described in detail. The first was a zircon-biotite-syenite in which large white felspars frequently three-quarters of an inch in length exhibited a gneissic parallelism. The other specimen was a "somewhat altered quartz-felspar-porphyry".

In the two specimens there were many signs of strain and crushing; the porphyritic individuals of the syenite had been granualted about the edges. These brief descriptions support the field evidence that practically all rocks of the red granite belt have been affected by slight to moderate dynamic metamorphism. In many cases the rock appears unstressed in the hand specimen but in thin section the effects of directed pressure are evident. Major crush zones are of frequent occurrence within the red granite and it is within these that most uranium mineralisation is located. The red granite is typically a "stressed" one and wide zones of shearing, trending NE-SW, are prominent east of Mount Painter.

The red colouration of the granite is probably due to the presence of iron oxide within the felspars. Specular hematite is widely distributed in red granitic rocks. It is particularly conspicuous in the thin light red pegmatite stringers near Yudnamutana. Ward and Jack (1915) attributed the occurrence of such primary oxide material in these lodes and pegmatites to an "unquestionable deficiency of sulphur" (p. 11).

In numerous localities the red granite is traversed by dykes of light red pegmatite. In these the felspar crystals are in many cases unusually coarse, and may range up to one foot or more in diameter. These pegmatites are generally devoid of important mineralisation, but books of muscovite and pockets of schorl are not uncommon.

Microscopically the red granite is comparatively well-mineralised. Specularite, magnetite, ilmenite, sphene, rutile, apatite and monazite are common accessory minerals but from an economic standpoint, however, no large concentrations of any of these primary minerals have been discovered. Several primary minerals of the rare earth suite have been found about Mt. Painter. Monazite occurs prominently in No. 2 workings on Radium Ridge and has been identified with uranium minerals in several other workings, but analyses by Mr. R.G. Thomas (Mawson 1944) indicate that thorium content is very low. According to Mawson (personal communication) allanite (cerium epidote) occurs in the Hot Springs Creek, a short distance upstream from the Paralana Hot Springs. The allanite occurs as small crystals and grains within gneissic red granite. With the exception of a radio-active ilmenite described by Broughton (1925) (actual occurrence unknown), Fergusonite is the only primary uranium mineral yet located. It is found at No. 2 Workings, on Mt. Gee, and at the "Smiler Greenwood" deposit.

(ii) The Leucogranites. (White granites).

A large mass of white granite outcrops in the north-eastern portion of the area. It forms a batholith extending about six miles in north-south and three miles in an east-west direction. The granite has intruded the Thick Quartzite horizon in what appears to have possibly been an anticlinal structure in the sediments. Within the batholith large xenoliths of quartzite are plentiful and near its periphery there is abundant evidence of magmatic stoping. Wedge-like dykes of granite project into the quartzite along planes of weakness.

The leucogranites are typically unstressed but are traversed by two prominent joint systems which trend approximately N 90-100°E and N 150-170°E. In outcrop "torr" shaped boulders are prevalent. There is no evidence to suggest that the granite was introduced prior to folding of the sediments. The regional cleavage of the sedimentary and metamorphic rocks to the west is not reflected within the granite.

Like the red granite, the leucogranite is essentially a binary granite comprising quartz and felspar, but No uranium minerals were located within its boundaries.

Within the central Mount Painter red granite complex there are several dykes of white pegmatite or pegmatitic granite which, although pegmatitic in nature of occurrence, texture and mineral associations, are referred to the leucogranite suite. These white pegmatitic dykes strike in a more or less east-west direction and follow a set of fractures in the red granites. One very persistent dyke traversing the relatively low undulating red granite country north of Mt. Pitts continues a practically unbroken course in an east-north-easterly direction for more than two miles where it approaches closely to more massive occurrences of pegmatites and pegmatitic granite of the same group. These massive pegmatitic granites extend east and west and form the "Armchair" and the Mount Ward Range. The dykes on their western extremities curve southwards to parallel the course of the "two-mile" dyke, previously mentioned.

The principal constituents of the pegmatites have a coarsely crystalline habit and in some places there are notable concentrations of tourmaline and mica. Books of muscovite were found measuring more than four inches in length, but the average size is considerably less. The white pegmatites in a number of places grade into the light red pegmatites. To what extent these two generations of pegmatite are intergrown and to what extent recrystallization of the original pegmatite took place was not investigated.

A third group of leucogranites is present outside the Mount Painter area proper. This group may be coextensive with the foregoing pegmatites. Mawson and Dalwitz (1945) have described it in considerable detail. The Giant's Head, the Needles and Tourmaline Hill are typical occurrences and lie west of the area mapped. Their positions are marked on the reconnaissance sketch plan (Plan 3294). Other outcrops known as the Pinnacles and Sitting Bull are situated on the extreme south boundary of the area. Mawson and Dalwitz (p. 48) consider that the intrusions are "in the nature of cupola summits above the general plutonite mass of a large scale granitic intrusion" into the thick Proterozoic and Cambrian sediments. "They represent the summits of active upward migration of magnetic gases and solution."

The following quotations, taken directly from pages 47 and 48 of Mawson and Dalwitz's paper, summarize some important features of the intrusion:-

> "The igneous rocks of these cupolas are essentially leuco-cratic, quartz-orthoclase (orthoclase microcline and microperthite) - albite (sodaclase) rocks. Rarely if ever is the plagioclase as calcic as oligoclase; in very many the sodaclase is in excess of orthoclase. Femic minerals are absent to rare..... There is .....

clear evidence of large scale migration of soda, silica, chlorine, boron and potash into the surrounding rocks enriching them in such minerals as albite, scapolite, mica and tourmaline."

"Albite is almost ubiquitous in both igneous and metamorphic rocks".

"For the most part these cupola rocks of Umberatana differ in texture from normal granites. They bear evidence of having crystallised from magmatic liquids abnormally rich in volatiles resulting in pegmatitic, aplitic and even spongy textured crystallizations with abundance of liquid and gaseous inclusions in the feldspar and quartz. The summits of the cupola masses are crystallizations of this kind, but a transition in texture towards more normal granite is noticeable in more deeply exposed areas."

"Accessory minerals in the leucogranite are very irregularly distributed but in some places there are notable concentrations of tourmaline, apatite, sphene and garnet."

(iii) The Basic Igneous Rocks.

Rocks of this suite are very variable in mineral composition texture and nature of field occurrence. Both field and laboratory studies of them have not been extensive and many important problems associated yet remain to be solved. Mawson (1923 & 1926) has collected much interesting information concerning these rocks particularly from the Wooltana region, but also from the Mt. Painter area.

The rocks range from coarse dolerites to hyalopilitic basalts. Many of them are highly vesicular and considerably metamorphosed. To a large extent outcrops of basic igneous rock are restricted to the south eastern portion of the area mapped, but isolated intrusions occur near Yudnamutana, the Daly mine and on the western extremity of Radium Ridge.

There is still much uncertainty as to what extent certain occurrences of rocks of this suite represent igneous intrusion or extrusion.

South from the Lady Buxton Copper Mine metamorphosed basic igneous rock are interbedded with metamorphosed sediments. In many cases it is undoubtedly intrusive as there is evidence of the overlying beds having been dislocated by the underlying igneous rock. In other places amygdaloidal textures are indicative of an extrusive origin. Mawson has noted scoriaceous lava textures near Mt. Jacob which also suggest lava outflows rather than sill-like intrusions at depth. Sullivan has recorded possible ash beds near Humanity's Seat. Mawson also has noted possible ash beds near Mt. Jacob and purple shales which were presumed though not proved to be transformed ash beds. A narrow top band of igneous rock of denser character Mawson has taken to represent the original chilled surface of a flow. Vesicles, if present, are in many cases filled with zeolites (stilbite) and calcite, and some show pronounced directional features.

The Wooltana Basic igneous belt is the southward continuation of the extensive area of basic igneous rocks, south of Humanity's Seat, which outcrop in the Mount Painter area. From near Wooltana Mawson (1926) has described normal and amygdaloidal melaphyres, an olivine diabase and an ophitic olivine diabase.

At Yudnamutana Mawson (1923) describes an altered diabase which he considers to be associated with the genesis of the copper lodes. He records other much altered basic igneous rocks occurring near the Daly Mine.

A related series of volcanic plugs ranging up to 16 chains in diameter and intruding the red granites have been recorded. The largest plug is situated less than one mile south of Humanity's Seat. Others have been found adjacent and about 100 chains to the west of Echo Camp. Another occurrence is located on the western extremity of Radium Ridge. There are several others of lesser importance. A further group probably genetically related to the foregoing lie in the wide shatter zone of the Echo Camp Fault. Examples of these are located near the Pinnacles pegmatite. The evidence clearly assigns to these plugs a post-red granite age and an age post-dating the period of faulting which has affected the red granites. In general these basic igneous rocks are relatively slightly altered. In places they are intersected by irregular milky quartz reefs, but they appear not to have undergone dynamic or regional metamorphism to any degree. On the other hand the sill-like and/or flow-like basic igneous rocks ofthe Wooltana Series, to the north of Arkaroola Station have undergone considerable alteration by such processes. The basic intrusives of the Yudnamutana region likewise have been very considerably metamorphosed.

From the foregoing descriptions the evidence points to two distinct periods of basic igneous activity. The one, pre-tillitic, when Proterozoic pre-glacial sediments were impregnated by sills and dykes, with extrusion of the same basic magma as vesicular lava flow. The deposition of ash beds was also a feature of this initial period. During the second period doleritic plugs were intruded into the red granites and the crush zones formed by post-red granite faulting.

For purposes of comparison it is convenient to note briefly several adjacent occurrence of volcanic rocks. At Blinman (70 miles to the south-west of Mt. Painter) volcanic rocks of a probably post-tillite age (Howchin 1922 , and Benson 1909) occur near Oraparinna (16 miles south of Blinman); similar rocks have been tentatively assigned a much later age. Mawson (1938 - 1939), reports volcanic agglomerates near the base of the Pound Quartzite, the formation which immediately precedes the Cambrian Archaeocyathinae limestones.

F. METAMORPHIC ROCKS.

Practically all rocks of sedimentary origin and most of the basic igneous rocks of the area have undergone some degree of metamorphism. Whilst regional metamorphism is general the intensity of the alteration is most marked nearer the red granite mass. Volatiles which emanated from the red granitic magma reacted metasomatically with constituents of the country rock and produced a variety of new minerals. Where high temperatures were present in the metamorphic aureole the original sediments have been altered by dehydration and decarbonation processes. Many other processes have also operated in bringing about the alteration of earlier sediments and basic igneous rocks.

In general metamorphism displays marked zoning in many areas and there is a distinct relationship of these zones with the boundaries of the red granite belt. Variations in sedimentary type and faulting have modified locally this general zoning. The major metamorphic grades readily recognised in the field are as follows:-

Quartzite
↓
Sericitic (muscovitic?) quartzite
↓
Quartz-mica-schist
↓
acid gneiss
↓
granitic gneiss

Slate
↓
Spotted Slate
↓
Knotted Slates and Schists
↓
Andalusite Schist
↓
Tourmaline and Corundum-sillimanite-cordierite Schists.

Limestone
↓
Marble
↓
Scapolite Marble
↓
Tremolite-Actinolite Marble
↓
Brucite-vesuvianite-actinolite-epidote-marble
↓
Amphibolite.

The basic igneous rocks of the Wooltana belt and Yudnamutana have undergone saussuritisation, serpentinization, epidotisation and other processes of alteration to produce melaphyres and diabases.

Dynamic metamorphism has played a lesser part than thermal metamorphism. Cleavage has been induced in many rocks, particularly in slates. About Mt. MacDonald and Freeling Heights micaceous quartzites show strong regional cleavage which is particularly striking on the aerial photographs. The regional cleavage in the north-western area strikes approximately N120°E and dips south at 70-80°.

Within the red granite belt dynamic metamorphism has produced prominent crush structures and it is in and adjacent to these crush zones that most of the known uranium deposits have been found. Thin sections of some of these red granitic rocks, examined under the microscope show prominent "crush" borders to mineral grains and crystals. Pronounced shadow extinction of the quartz grains is also a feature of such sections.

South and east of Mount Painter a definite schistosity is apparent in the red granitic rock, and near Minerva Heights directed pressure has produced metamorphic structures reminiscent of organ-pipe gneiss.

G. GEOLOGICAL STRUCTURES.

(i) Folding.

In South Australia, the Proterozoic, the Cambrian and perhaps the Ordovician period were notable for the accumulation of a vast geosyncline of sediments. With minor exceptions, sedimentation proceeded without very significant disturbance. In the description which follows, this accumulation will be referred to as the Flinders geosyncline in contra-distinction to the MacDonnel geosyncline of central Australia. These two geosynclines probably were continuous as the "Central Geosyncline" of W.H. Brian (1932 p. 20).

In both of these 'basins' sedimentation was terminated by an orogenic process which probably occurred contemporaneously in the two areas, possibly in the late or post-Ordovician period as Ordovician sediments conformably overlie Cambrian and Proterozoic formations in the MacDonnell Ranges where the three rock series have taken part in the folding movements.

So far no undoubted Ordovician sediments have been recognised in South Australia. Mawson (1939) has recorded a thick cross-bedded sandstone overlying upper Cambrian fossiliferous limestones in the Grindstone Range area of the Flinders Ranges. This may be the equivalent of similar red sandstone-quartzites of Ordovician age which occur in the MacDonnell Ranges.

Consideration of fold axes of the MacDonnell and Flinders Ranges favours contemporaneity of orogenic cycles in both regions, although Mawson (1942) personally favours a late Cambrian age of folding in the Flinders Ranges. The writer, however, does not know of any direct evidence to contradict the theory of contemporaneity.

Within the Mt. Painter area mapped in detail, the major folded structures are not readily apparent. For this reason a broad scale reconnaissance sketch map was prepared covering a larger area. (Plan 3294, Sheet 47). This plan reveals the broad folded structures which are similar to those found elsewhere in the northern Flinders Ranges. The folding is generally of a broad open character with the major axes of folding trending in an east-west direction, whilst nearly at right angles a second set of fold axes more widely spaced can be interpreted. In effect a series of periclinal (dome and basin) structures has been produced, elongated in an east-west direction.

The study of the orientation of the fold axes is considered to be important in deciding the age of the folding of the sediments now constituting the Flinders and MacDonnell geosynclines.

In the Flinders geosyncline there is a definite change in relative importance of the respective sets of fold axes. In the South near Adelaide, the folds essentially have a meridianal trend with minor changes in pitch attributable to folding along ill-defined east-west axes. Somewhat similar relations continue to hold until near Hawker where east-west fold element noticeably becomes more prominent. At Wilpena Pound and Blinman, the east-west and north-south elements are about equally developed and have produced the remarkable Pound Structures (Mawson 1942) for which the area is notable. Further north near Mt. Serle, folds trending in an east-west direction are more prominent, but north-south axial planes can be readily interpreted. Similar conditions continue with little modification to the northern extremities of the Flinders Range.

In the MacDonnell Ranges the folding likewise has a dominant east-west trend, but distinct changes of pitch have been noted (Madigan 1932, Mawson 1925, David 1932, Geological Map of Australia).

In the Mount Painter area the folding can be conveniently referred to two sets of fold axes, and it appears likely ~~that~~ the intrusion of the red granite was chiefly confined to an anticlinal structure and during emplacement probably accentuated the original dome-shaped area of weakness. Other prominent structures include an east-west major syncline passing through the centre of the Yudnamutana area and a complementary anticline occurring between Mt. Pitts and the Yudnamutana Range. The "competent" Thick Quartzite has been steeply folded and intruded by red granite. Other folds lying just outside the area are shown on the reconnaissance map (Plan 3294). These include the north, an anticline passing about two miles north of Tindalpina Hut, and in the south a cyncline in the vicinity of Arkaroola Homestead. Pitch data based on cleavage and bedding reachings observed in the Daly-Yudnamutana area indicate the presence of anticlinal and synclinal axes along which the dominant east-west folding can be pictured as having been warped. The reconnaissance sketch map (Plan 3294) illustrates the synclinal axis of this feature, but to the east, intrusion of granite and faulting prevent interpretation of the complete complementary anticlinal axis.

By and large the main structural feature of the Mount Painter area is the dome structure in which granite now largely replaces the older sediments.

Minor fold structures were not studied in much detail in the field and most of the trend lines have been interpolated from scattered field observations used in conjunction with the aerial photographs. Some of these require further elucidation, particularly east of the Paralana fault but elsewhere they are not likely to modify the general structural interpretation of the area.

(ii) Faulting.

Faulting in the Mount Painter area is considered in two groups. The eastern and south-eastern or Paralana group and the north-western or Mt. Saturday group. In both groups hade is small.

(a) The Paralana Group:

The Paralana fault is the most conspicuous and extensive one on the area. It is one along which there is evidence of movement having occurred even in Tertiary times. It is marked by a well defined crush zone along which signs of mineralisation (notably, uranium, copper, quartz and manganese) are not uncommon. The fault has been traced beyond the Ammonia Cave, south of Mt. Warren Hastings.

Several spur faults have been noted branching off from the main Paralana fault. One of these, the Echo Camp fault is the most prominent and is notable for tremendous accumulations of white quartz. Adjacent to Echo Camp a reef-like mass of solid milky quartz, 6 to 8 chains in width, is present and further west pegmatites and narrower quartz reefs mark this fault zone.

Near the "big loop" in the Arkaroola Creek, the actual trace of the Paralana fault is difficult to follow as the Arkaroola has in fact eroded a wide fracture and crushed zone in actintrolitic marbles. From this locality the fault curves south and then south-east and within its crush zone a number of pegmatites and doleritic plugs are present. The pegmatites occurrences include the Sitting Bull and the Pinnacles previously mentioned. The crush zone isstill well developed at the Sitting Bull, the eastern

Extending north from the Mouth of the Yudnamutana Gorge, the Paralana fault forms the main escarpment of the Flinders Range. To the south a low "front of range" escarpment is formed by another fault which evidently converges with the Paralana Fault near the Hot Springs. The junction cannot be observed and cross-faulting between the two major faults has let down a roughly triangular block in which Eyrian (?) sediments have been preserved. This southern escarpment fault probably delineates the edge of the Lake Frome Plain to within less than a mile of East Painter Creek. From there once more it converges towards the Paralana fault. It appears to die out about two miles south of Humanity's seat.

Along the escarpment leading south from the Lady Buxton copper mine, no defining fault was located, but there are two cross faults a little to the north of the mine.

Sullivan has indicated a possible fault adjacent to an actinolitic marble bed occurring south of East Painter Gorge, but further field work is required in this area to fully resolve the structure and stratigraphy.

Faulting of the Paralana group is post-red granite and post Wooltana Basic igneous. It is pre-leucogranite intrusions and preceded the intrusion of the younger basic igneous rocks as indicated by the nature of the occurrences of pegmatites at Sitting Bull and the Pinnacles and of the "unstressed" basic igneous rocks of the same locality.

(b) The Mt. Saturday Radial Faults.

This series consists of a number of hinge dip faults arranged in a radial pattern about a centre approximately one mile south-west of Mt. Saturday. The throw for these faults is generally small and the faults die out before the central focal point is reached. In addition strike faults occur which are not strictly members of the radial series, but otherwise are similar in character to the radial group.

Folding appears to have determined the fault pattern of this group very closely. Downthrow is normally to the north and in consequence the "competent" Thick Quartzite of the Yudnamutana Range has been forced up in relation to the incompetent beds to the north. It is evident also that these pivotal faults relieved stresses during the final stages of granite intrusion.

The faults have not been observed striking into the centre of the red granite belt, but near the Commonwealth mine they occur in quartzites heavily intruded by pegmatites, also within and without the infaulted block of actinolitic marbles, thus indicating that some at least of this faulting preceded the pegmatitic intrusion.

H. EPITHERMAL ACTIVITY.

The locality of Mt. Gee is the centre of widespread accumulations of epithermal quartz. Most of the quartz is distributed in geode form and amethystine quartz has been found in many cavities. Ghost structure is very common in many of the quartz crystals and drusy quartz masses frequently encase fluorite and gypsum crystals. In some instances Torbernite flakes occur within the geodes and at other times intergrown with the quartz. Specularite shows similar relations to the quartz. Other deposits of the epithermal quartz are situated on the western spur of Mt. Painter and in valley bottoms two miles south, and south-east of the main camp and at the Rock holes.

These formations appear to be of considerably vertical extent and to have a pipe-like form. On Mt. Gee the geode material extends six or seven hundred feet vertically. Near Sunshine Pound it occurs at the bottom of valleys more than 1,000 feet below the summit of Mt. Gee.

Originally, the cavities (now partly filled with deposits of epithermal quartz and other minerals) must have been remarkably open structures. For this reason they must have been formed relatively near the surface, at least within the zone of fracture. Assuming that the epithermal activity post-dated the formation of the 3,000 feet peneplain, a remnant of which has been preserved on Freeling Heights, the present upper limit of epithermal quartz on Mt. Gee would have been within 800 feet of the surface of the ancient peneplain.

Evidence from mineralogical studies suggests a late period of epithermal activity and this is consistent with geological and topographical data. As pointed out by Stilwell (this Bulletin), Torbernite occurs intergrown with epithermal quartz and the low values of the lead-uranium ratio for the torbernite indicate formation in the late Tertiary Period.

In the writer's mind, however, there is still a tendency to associate the epithermal quartz with the decadent phases of the hydrothermal activity of the Paleozoic era. In support of this contention it is to be noted that geodic quartz has been formed only within the red granite area, and is absent from the fault fracture zones which, one would believe would offer excellent loci for its deposition. It is to be noted that the Paralana Hot Spring which is still active is situated on a fault, but that the silica deposited, is in no way similar to that of Mt. Gee. Helium gas in minute quantities is being evolved from the Spring waters (Mawson, 1927).

Whilst the age of the epithermal activity need not necessarily be correlated with a period of peneplanation or still-stand as evidenced by the 3,000, 2,000 or 1,000 feet erosion surfaces, it is appropriate to study this possible relationship in view of the uncertainty as to the precise origin of the uranium deposits. The age of these erosion surfaces are thus considered somewhat in detail.

I. THE AGE OF THE EROSION SURFACES.

In the physiographic section of this report reference was made to two erosion surfaces occurring very approximately at elevations above sea level of 3,000 and 2,000 feet respectively.

The 3,000 feet base surface is well preserved in the area known as the Freeling Heights and is undoubtedly most ancient and its perfection indicates a very protracted period of still-stand. The 2,000 feet erosion surface has been deeply eroded, but its former existence is postulated from the land-form analysis (Plan 3210). There is no evidence of any Mesozoic or Tertiary sediments having been deposited on either of these surfaces, and thus there are no stratigraphical clues to their ages.

It is reasonable to picture the Mt. Gee and related epithermal deposits to have been formed when the Freeling peneplain was developed and before it was dissected to any marked degree as epithermal quartz deposits are found near the summit of Mt. Painter (2,850 ft.), an eminence which is considered to be a remnant of the Freeling surface and a monadnock on the Yudnamutana surface.

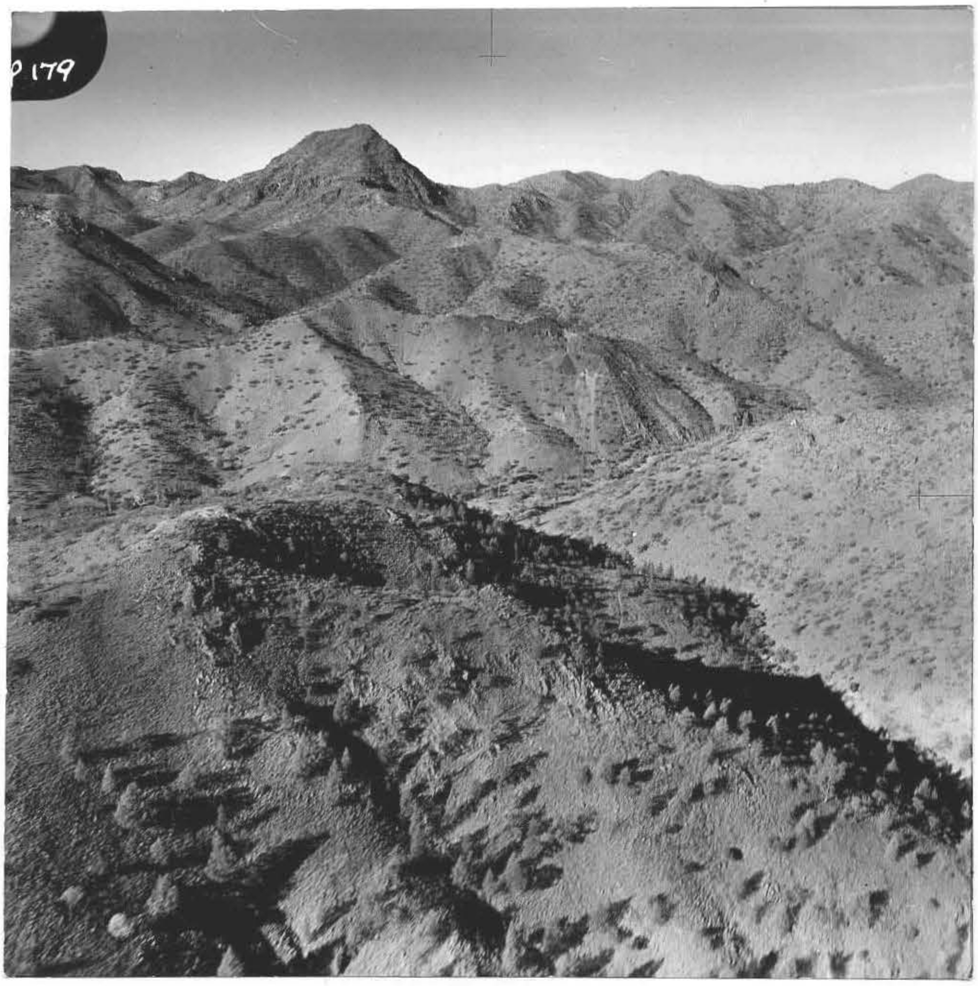


FIG. 2. Mount Painter, Monadnock on the Yudnamutana Surface.

It will be seen from discussions elsewhere in this Bulletin that the secondary uranium mineral deposits were not necessarily formed during the period of epithermal activity, but may have been later. Thus, they may be related to the 2,000 feet erosion surface when the water table was much higher (several hundred feet).

A U/Pb ratio determination made on the torbernite from the No.6 Workings indicates formation in the late Tertiary Period, but no determination has been made for the torbernite found intergrown with epithermal quartz which, on known physiographic and stratigraphical evidence, one would assign a much older age of formation, possibly early Mesozoic or late Palaeozoic. This problem, however, requires further study before any definite conclusions can be drawn.

J. GEOLOGICAL HISTORY OF THE AREA.

During the course of field investigations certain observations were made which have a bearing on the sequence of various important tectonic events, igneous intrusions and periods of mineralisation, and an attempt has been made herein to prepare a tentative sequence of the major geological events in the area mapped. It is outlined as a working hypothesis which will be bound to be modified as more detailed work is carried out.

1. Proterozoic and early Palaeozoic sediments accumulated in the region during a long quiescent period. There appear to be no major breaks in sedimentation although minor overlaps are present. The sedimentary types range from tillites and arkoses through sandstones and quartzite to slates, calcareous slates and limestones.

Prior to Sturtian glaciation (Late Proterozoic) much of the Wooltana basic igneous rocks were intruded (Sills and dykes) or extruded (lava flows and ash beds). The Proterozoic ice age was followed by generally warmer conditions culminating in the aridity of the late Cambrian and Ordovician periods.

Most of the sediments are devoid of fossils but worm burrows occur at several horizons low in the Proterozoic series.

During the Cambrian Period definite marine incursions were responsible for deposition of highly fossiliferous (Archaeocyathinae) limestones. Accumulation of sediment probably continued well into the Ordovician Period.

2. The sedimentary cycle was concluded by the foundering of the region at a time variously estimated as late Cambrian (Mawson ≠) or late Ordovician (Sprigg 1944), during which the geocyclinal structure with open folds developed.

≠ Mawson (1911, 1923) considers that the red granite area of Mt. Painter is an early or mid pre-Cambrian complex and that upon this eroded basement there may be a second ancient sedimentary series which in turn is overlain unconformably by Adelaide Series and Cambrian Sediments. The whole series underwent folding and intrusion by leucogranites (Mawson 1923, 1944) in the late Cambrian.

3. Intrusion of red granite, and granitisation accompanied the folding. It is not known to what extent the granite intrusion enhanced the degree of folding at Mt. Painter. The Thick Quartzite was probably domed into a laccolith structure by the invading granite, but corundum and biotite schists included within the central igneous mass attest to a considerable degree of assimilation and stoping of underlying argillaceous horizons.

4. Late in the period of red granite 'intrusion' there was a period of pegmatisation. The pegmatitic mother liquors apparently introduced the Fergusonite(?) and Monazite of Radium Ridge.

5. About Freeling Heights a white granite invaded the Thick Quartzite. Magmatic stoping occurred on a large scale, and intrusion was most active along an anticlinal structure.

The red granite is older than the white granite. It is crushed and sheared and frequently has been gneissified. This gneissification is not to be confused with earlier sedimentary structures not wholly destroyed in the processes of (red) granitisation. The white granite is unstressed.

Accompanying granite intrusion there was concomitant regional metamorphism of overlying sediments. Calcareous horizons overlying the Thick Quartzite were decarbonised producing massive actinolite-tremolite.

The major faults of the area are post red granite. The Mt. Saturday Radial faults are compressional faults with small hade and wide crush zones. Occurring largely in slates and lime tones they apparently relieved pressure after intensified folding of the relatively "competent" Thick Quartzite formation during red granite 'intrusion'.

The relationship of the white granite batholithic intrusion to faulting has nowhere been observed.

Post-dating the major faulting a second phase of basic intrusion (doleritic) took place. Volcanic necks and plugs have penetrated the red granite, the older basic igneous series and metamorphosed sediments.

The red granite near the "Arm Chair" has been intruded by white light red pegmatites along lines of weakness. A second group of leucogranite pegmatites has intruded the crush zones of the Paralana series of faults. The two groups of pegmatites may be genetically related. Elsewhere along fault zones massive reefs of milky quartzite outcrop prominently, and within the red granite area massive milky quartz intersects one of the newer basic igneous plugs. The quartz may represent a late pegmatite phase.

Mineralisation is closely related to the red granite 'intrusion'. Deposits of copper, gold, bismuth, antimony and tungsten minerals occur chiefly about the margins of the red granite mass and practically all such deposits are associated with zones of crushing and faulting. Notwithstanding the association of metallic mineralisation with intrusion few ore minerals have been located within the red granite. Uranium minerals are restricted to the red granite and traces of copper have been noted there in several instances.

6. The cycle of erosion that may have been introduced with the buckling of the geosynclinal sediments, continued throughout the long period of acid and basic intrusion and primary mineralisation. When epithermal processes were active the Mt. Painter area probably had been reduced to a base surface. Much uncertainty still exists concerning the age of the Mt. Gee geodic quartz deposits. Lead/uranium ratios made for age determinations indicates torbernite formation in the late Tertiary period, but other evidence suggests that the possibility of a greater age, possibly late Palaeozoic, cannot be overlooked.

Two erosion surfaces have been identified provisionallly at 3,000 and 2,000 feet. The 3,000ft. level is regarded as most ancient. The formation of the oxidised copper ores of Umberatana and the secondary uranium mineralisation and epithermal activity about Mt. Painter indicate significant correlations with the two erosion surfaces. The only stratigraphical evidence available favours an age for these secondary mineralization processes in the late Palaeozoic or early Mesozoic periods. However, if certain premises are correct* uranium age determinations indicate that at least some of the secondary uranium mineralization occurred in late Tertiary.

---

*The lead uranium ratios have been made on secondary ores from leached zones of the "ironstone" lodes. The lead content of such ores is generally low and varies very significantly. It is to be noted that experience has shown that such ores frequently give quite unreliable information on which to base age determinations. Reference has been made in the "International Critical Tables of Numerical Data, physics, chemistry and technology", (Vol. 1, p.382, 1926) to certain of these difficulties as follows:

> "Low lead ratios have little significance on account of the ease with which certain minerals abstract lead from circulating natural waters. The atomic weight of lead should be determined wherever possible in order to make certain that the lead is of radio-active origin. In general only primary minerals are suitable for age determinations."

REFERENCES.

1909 BENSON, W.N. "The Basic Rocks of Blinman, South Australia, with notes on associated or allied rocks". Trans. Roy. Soc. S.A. 33.

1932 BRIAN, W.H. "Early Palaeozoic Earth Movements in Australia." A.N.Z.A.S.S. 21 Sydney.

1925 BROUGHTON, A.C. "Further Notes on Radio-active ilmenite near Mt. Painter". Trans. Roy. Soc. S.A. 49.

1922 HOWCHIN, W. "A Geological Traverse of the Flinders Range from the Parachilna Gorge to the Lake Frome Plains." Trans. Roy. Soc. 46.

1932 MADIGAN, C.T. "The Geology of the Western MacDonnell Ranges". Quart. Journ. Geol. Soc. 88.

1911 MAWSON, D. "Pre-Cambrian Areas in the North-eastern Portion of South Australia, and the Barrier, New South Wales". A.A.A.Soc. Sydney.

1923 MAWSON, D. "Igneous Rocks of the Mt. Painter Belt". Trans. Roy. Soc. S.A. Vol. 47.

1926 MAWSON, D. "The Wooltana Basic Igneous Belt". Trans. Roy. Soc. S.A. Vol. 50.

1927 MAWSON, D. "The Paralana Hot Springs". Trans. Roy. Soc. S.A. 51.

1942 MAWSON, D. "The Structural Character of the Flinders Ranges". Trans. Roy. Soc. S.A. 66 (2).

1944 MAWSON, D. "The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia". Trans. Roy. Soc. S.A. 68 (2).

1945 MAWSON, D. and DALLWITZ, W.B. "The Soda Rich Leucogranite Cupolas of Umberatana". Trans. Roy. Soc. S.A. 69 (1).

1944 MAZE, W.H. "The Geomorphology of the Central Eastern Area of New South Wales". J. & P. Roy. Soc. N.S.W. 78 (102).

1945 SPRIGG, R.C. "Some Aspects of the Geomorphology of the Central Mount Lofty Ranges". Trans. Roy. Soc. S.A. 69 (2).

1945 STILLWELL, F.L. and EDWARDS, A.B. "Uranium Minerals from Mt. Painter". This Bulletin.

1916 WARD, L.K. and JACK, R.L. "The Yudnamutana Mining Field". Geol. Surv. S.A. Rept. No.3.

1938 MAWSON, D. "Pre-Cambrian and Sub-Cambrian formations at Parachilna Gorge".
Trans. Roy. Soc. S.A. 62 (2).

1929 MAWSON, D. "Cambrian Sequence in the Wirrealpa basin".
Trans. Roy. Soc. S.A. 63 (2).

1939 MAWSON, D. "The late Proterozoic Sediments of South Australia".
A.N.A.A.S. Canberra 24.

1932 DAVID, T.E.W. "Explanatory Notes to accompany a new geological map of the Commonwealth of Australia".

1941 MAWSON, D. "Middle Proterozoic sediments in the neighbourhood of Copley".
Trans. Roy. Soc. S.A. 65 (2).

PREFACE.

This section deals with and describes the specific occurrences of uraniferous minerals, their occurrence and any development which has been carried out on them. Some of the old workings are very small in extent, and after a preliminary examination no further work was done on them. The principal work was done on No.6 Workings, Mt. Painter, and at Greenwood's Camp Area, East Painter. Except for the sinking of some test pits and shallow shafts on the "Smiler" Greenwood and "Bentley" Greenwood, no testing was carried out on any of the other deposits. However, all the other old workings were examined, and new areas for prospecting were indicated.

The present discussion represents final conclusions, of which the various stages of deduction have been given in fortnightly progress reports Nos. 1 - 31 inclusive (July 24th 1944 to Sept. 15th 1945).

Recently a paper, "The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia", by Sir Douglas Mawson, has been published (Trans. Royl. Soc. Sth. Aust., Vol. 68 (2), 1944). In this paper Mawson summarises his knowledge of the Mt. Painter deposits, and it provided much valuable information for the present investigation.

Geiger Muller surveys of the No.6 Workings, the Greenwood's Camp Area, the "Smiler" Greenwood and the "Bentley" Greenwood prospects were made and although the results are given in detail in Part IV of this Bulletin, it is appropriate to refer briefly to their value to the geologist.

In various places, such as the No.6 Workings and East Painter, the hematite breccia indicated higher radioactivity than the surrounding rocks, but it often showed no sign of torbernite or other uraniferous minerals. This radioactivity may be due to the presence of monazite, or of radium compounds which, being relatively insoluble, have been left behind when the uraniferous minerals were leached by surface waters. In either case the radioactivity of the rock indicates a higher uranium content than chemical methods disclose. This discrepancy explains the difference between the uranium content of various deposits as given in Mawson's paper and as found during the present investigation. The uranium content as given in Mawson's paper were determined electroscopically and so would be higher than if they had been determined by chemical analysis. In fact, it is quite possible for a specimen to contain no uranium at all and yet to indicate quite an appreciable radioactivity.

The geophysicists in this investigation calibrated radioactivity against known uranium content and were then able to determine the uranium content of mine samples.

An investigation of the mineralogy and ore occurrences was undertaken by Drs. Stillwell and Edwards and their report is presented as the first report of Part III. It contains a complete description of the mineralogy, and discusses the origin of the deposits. The latter subject, however, is controversial and several possible explanations have been advanced. On this account the views of the field geologists on genesis are also presented in separate reports at the conclusion of Part III.

It is hoped in this way to portray the possible modes of origin of the deposits and the future prospects associated with each of them. Although the supergene origin is generally accepted as being the best one, present knowledge is insufficient to entirely discount the other hypotheses and thus the limitations of the field cannot yet be strictly defined on scientific premises.

111. REPORTS ON INDIVIDUAL URANIUM OCCURRENCES.

1. Mount Painter Group.

A. Nos. 1, 2 and 3 Workings, Radium Ridge.

by

C. J. SULLIVAN

Geologist, Mineral Resources Survey.

(Plan 3197, Sheet 37, illustrates this report).

Radium Ridge is a very prominent topographic feature extending generally east and west for a distance of approximately 2½ miles. At its closest point it is approximately 1 mile north of the Main Camp. The slopes are steep on the southern side of the ridge and flatten out on the northern side. Some of the steepest sections of the ridge are due to the resistance to weathering of a silicified zone, somewhat similar in appearance to the "Jaspers" of Western Australia. Nos. 1, 2 and 3 workings are located on this ridge.

(i) No. 1 Working.

It consists of a costean 15 feet long and 3 feet deep. It is situated on an outcrop of hematite 45 ft. long and 15 ft. wide. The hematite is of a very dense nature and is intergrown with quartz. It contains small quantities of Fergusonite and traces of yellow uranium minerals. The hematite may have had a different origin from that associated with torbernite in the crush zones. The deposit is thought to be of little economic importance.

(ii) No. 2 Working.

This deposit is very similar to No. 1 and there is a costean on a hematite outcrop. The hematite is very hard and dense and is intergrown with quartz. Fergusonite and monazite are present in places in considerable quantities (Stillwell and Edwards 1945).

(iii) No. 3 Working.

This deposit exposed by this working resembles the usual type of torbernite occurrence associated with hematite and limonite in a crush zone. The hematite and limonite extend along the ridge for a distance of 200 feet and have an average width on the surface of 30 feet. They appear to have resulted from the replacement of a massive pink felspar rock - possibly a pegmatite or porphyry. The latter is intrusive into a paler, finer grained granitic rock. The ironstone is in the form of a capping.

At one place, torbernite occurs over a length of 5 to 10 feet. A small open cut has been made into this occurrence and a shaft has been sunk from the end of the open cut. At the time of the present examination, the shaft was not accessible. Mawson (1944) states that a crosscut off the shaft passed beneath the ironstone body and revealed "weathered granite spangled with torbernite."

No further testing of this deposit was carried out, as it was considered unlikely to produce more than very small quantities of uranium ore with a grade not less than 0.25% U_3O_8. There is a close similarity between this deposit and No.4. Both are ferruginous cappings carrying traces of torbernite and overlying weathered granite rocks containing small amounts of that mineral.

B. No.4 Workings.

by

C.J. Sullivan, Geologist, Mineral Resources Survey.
(Plan 3208, Sheet 38 illustrates this report).

(i) Situation.

This deposit is situated approximately ½ mile north of the Main Camp on Mt. Gee Creek.

A steep ridge rises 150 feet above creek level and is capped by two occurrences of hematite. The larger body has a length on the surface of 260 feet and a maximum width of 210 feet. The second body is 100 feet long by 80 feet wide on the surface. It seems likely that the hematite has a maximum thickness of 25 feet.

The deposit consists mainly of hematite with some limonite. In places it contains small amounts of pyrite. Very small quantities of torbernite have been noted in the "ironstone". In places, considerable quantities of a bright yellow mineral, at first thought to be a uranium mineral, were found in cavities in the "ironstone". It is now known that this is an iron mineral. The "ironstone" seems to have formed by the replacement of a pink felspar granite or pegmatite. An adit 15 feet in length has been driven below the "ironstone". It intersected soft gneissic rock containing traces of torbernite coating planes of schistosity and fracturing. No ore with a grade not less than 0.25% U_3O_8 was exposed.

Two steeply dipping shear zones containing hematite, manganese dioxide, quartz and biotite were also mapped. A steep rock face, exposed in the creek, reveals the iron concentration decreasing rapidly from the surface downwards. No uranium minerals were found in these deposits.

Along the creek in the western part of the area mapped there are three outcrops of unusual interest. These consist for the most part of masses of pink gypsum needles and quartz enclosing crystals of gypsum. Calcite is found deposited on the quartz. The mineralogy of these occurrences is described in detail by Stillwell and Edwards (1945).

C. No.5 Workings.

by

E. Broadhurst, Geological Survey of South Australia.

This deposit is on the west side of Mt. Gee, where a shaft about six feet deep has been sunk. The deposit is associated with the epithermal quartz and hematite which is characteristic of Mt. Gee. Torbernite occurs as thin veins in brecciated zones. The brecciated zones are most irregular and no definite prospecting programme could be laid out. In no place in this vicinity did payable ore appear to have been found.

D. No.6 Workings.

by

E. Broadhurst, Geological Survey of South Australia.
(Plans 3184, 2951, 3176, 3179, 3175, 3183, 3177, 3180, 3178, 3182, 3185, 3193, 3187, 3196, 3190, 3181, 3188, 3189, 3186, 3201, Sheets 1 - 11, 11A, 12 - 18 inclusive, 34, illustrate this report).

(i) Location.

These workings are the most important at Mt. Painter, and are situated on the east bank of Radium Creek about a quarter of a mile south of the main camp.

(ii) Geology.

The predominant rock types in the immediate vicinity of the ore deposit are banded gneisses. These were originally sedimentary rocks which have undergone various stages of granitisation, but from which the traces of bedding planes have not been eradicated. Centres of granitisation, or areas where this process has proceeded further than usual, are indicated by the presence of pegmatitic granite.

One of the outstanding features near the ore deposit is the silicification of the gneisses, a process which was described by Stillwell and Edwards (1945). The greater portion of the silification occurs in the southern part of the mine area. Going north along the No.6 ridge there is a fairly sharp contact between the gneiss and silicified gneiss on the western side of the ridge, but just to the north of the surface shaft the silification dies out. On the eastern side of the ridge the contact is rather indefinite and cannot be accurately fixed. The definite western contact came on an E-W course to the south of the mine, where its course changed to N-S along the western side of the No.6 ridge. In this portion, with which the ore deposit is chiefly concerned, the silicification dies out rapidly in depth.

The silicification of the gneisses has resulted in a number of rock types, ranging from a rock resembling a quartzite to one resembling an igneous rock. For mapping purposes the silicified gneiss was distinguished from the banded gneiss by the absence of bedding planes, traces of which had been destroyed in the process of silicification.

In the vicinity of the ore deposit, extensive brecciation has occurred. Brecciation is always confined to the silicified gneiss, probably owing to the brittle character of this rock compared to the banded gneiss. Accompanying, or closely following the brecciation was the introduction of ferruginous material discussed in the report by Stillwell and Edwards (1945). Where this ferruginous material encountered the breccia, it reacted with the matrix to give a hematite breccia, a rock consisting of angular quartzite-like fragments in a matrix consisting largely of hematite and felspar with the felspar often almost completely kaolinised. In non-brecciated rocks, the ferruginous material is confined to veins, a few inches thick, in silicified gneiss. These veins consist of hematite with kaolin, and were evidently formed by the action of the ferruginous material on the gneiss along joint planes. In a few places the ferruginous material seems to have soaked into the enclosing rock, giving an 'argillaceous ironstone' traversed by veins of hematite rock.

The matrix of the breccia varies in composition. Specular hematite is present in nearly all of it, but the nature of the material accompanying the hematite varies. Sometimes it is clayey, making the 'swelling ground' which made mining and diamond drilling operations so difficult in some places. In other places the matrix is hard, giving a black matrix studded with the shiny flecks of hematite. On the surface the hardening effect on the clay and probably some 'case-hardening' effect have converted the clayey to a hard matrix, so that the former is rarely seen on the surface.

The predominant strike of the hematite breccia is N.E.-S.W., which is shown by 'fingers' having this strike at each end of the main mass of hematite breccia. This strike is also clearly shown on the north side of the lode channel on the 100 ft. level. However, at the northern end of the massive outcrop on top of the ridge the contact with the silicified gneiss seems to have a N.W.-S.E. strike. Actually, of course, there must be more than one strike to allow the bulging of the hematite breccia that is seen in the big outcrop.

Besides the brecciation, the rocks show numerous signs of deformation in the presence of fault planes. The most persistent of these is the 'lode channel', an E-W shear dipping 70° S, which persists from the surface to the 150 ft. sub-level in the No. 2 winze. The shearing movement along this plane originated as the movement along a bedding plane in the banded gneiss, and which continued through the silicified gneiss and the hematite breccia. On moving away from the contact with the banded gneiss into the silicified gneiss and hematite breccia, the shear shows a tendency to split into branches, due probably to the removal of the concentration of shearing stress imposed by the bedding planes.

It is not quite clear from the field evidence whether the shearing movement of the lode channel occurred before or after the formation of the hematite breccia. It is more than possible, of course, that the stresses which caused the shearing along the lode channel would also cause the brecciation of the silicified gneiss, but against this conception is the fact that there is no spreading of specular hematite rock along the lode channel from the hematite breccia, a phenomenon which would be expected if the hematite were formed after the lode channel. The difference of configuration of the hematite breccia projected on to the foot-wall and hanging wall of the lode channel suggests that the shearing movement was pre-brecciation. However, the shape of the breccia is very lenticular and it is quite likely that shearing would occur at any sudden change of thickness, and the configurations of the projections of the breccia on the lode are confused by the complexity of faulting in the vicinity. Bearing these facts in mind, it seems more likely that the shearing occurred after the brecciation.

(iii) Ore Deposit.

The ore deposit is formed in the lode channel. From the surface to a depth of 25 ft. it was reported that there was a vein of torbernite one inch wide, but at that depth there was a sudden widening of the lode to several feet. Between this depth to the 50 ft. level, and along a length of about 25 ft., the main stope of the mine occurred. There was no ore worked beneath the 50 ft. level. Thus the lateral extent of the ore shoot was only 25 ft., with a vertical extent of only 25 ft., making an ore shoot of very small dimensions.

According to Mr. G.A. Greenwood, the ore consisted of a breccia in which the matrix was autunite and manganiferous ironstone. The fragments of the breccia consisted of silicified gneiss. The manganiferous ironstone is a prominent feature of the ore deposition. It can be seen around the sides of the old stopes occurring along the course of the various shear walls, and usually it forms ore containing about 1% U_3O_8. In the richest of the ore mined during earlier operations apparently the autunite had replaced the ironstone, giving the ore described above in which part of the matrix of the breccia consists of autunite. Away from the main ore shoot the manganiferous ironstone rapidly thins to a vein about an inch wide and finally disappears. However, even when there is no continuous vein of manganiferous ironstone on the shear wall there are often fragments of this material in the crushed 'lode channel', and these fragments usually contain flecks of torbernite.

The recent development on this deposit showed that small amounts of torbernite tend to occur near the outer margins of the hematite breccia. Sometimes the torbernite occurs in the silicified gneiss near its contact with the hematite breccia. When the torbernite occurs in the hematite breccia, it is often as flakes in the joint planes of the fragments of silicified gneiss.

The rocks surrounding the main ore shoot are very faulted. To the north, on the footwall of the main shear the rocks show practically no sign of faulting. To the south on the hanging wall of the main shear, where the ore body occurred, a number of faults occur whose courses are shown by manganiferous ironstone. Another significant feature is the presence in the rocks on the hanging wall of the shear plane of a "V" shaped mass of silicified gneiss between two bodies of hematite breccia. This "V" shaped mass is 20 ft. wide at the surface and tapers to nothing at the 50 ft. level, so that below this level the two bodies of hematite breccia become one. The shape of the main ore-body seems to have been largely controlled by the shape of this wedge. In the open cut, the top of the stope seems to follow not so much the main shear as the silicified gneiss-breccia contact. At this point an echelon structure of the main shear occurs, and the shear wall on the 50 ft. level is to the south of that on the surface. The ore body seems to occur in the echelon gap between the two shears.

In considering the deposition of the ore-body there seems to have been two main factors involved. The first of these is obviously the main shear. The other, not so obvious, is the edge of the hematite breccia. The main ore-shoot was situated at the intersection of these two features. What influence the "V" shape of the silicified gneiss and its much-faulted structure had upon the deposition of the ore has already been discussed.

(iv) Development.

The development of the mine was influenced throughout by the stage of knowledge of the lode structure which had been reached. At the beginning of the present operations nearly all the workings consisted of drives on the lode channel or cross-cuts in the banded gneiss or breccia, and there was little indication of the structure of the hematite breccia. The initial examination revealed first, of course, the association of the ore deposit with the main E-W shear (lode channel), and second, the proximity of the deposit to the contact of the banded gneiss and the silicified gneiss and hematite breccia, which at that stage were grouped together under the name of 'unbanded' rocks. A development program was then laid out, first to continue the prospecting of the lode channel with winzes from the 50 ft. level and 100 ft. level, and second to search for possible parallel lode channels by driving north from the 100 ft. level and continuing the 100 ft. level south Portal. Also diamond drilling was to be conducted north from the 100 ft. level. The northern drive from the 100 ft. level was to follow the contact of the banded gneiss and the 'unbanded rocks' which it did for the first part of its course. However, torbernite had been encountered in one of the diamond drill holes, so the drive was turned to this point. Near the point where torbernite had been encountered the drive branched, one branch going to the north in accordance with the original programme, and the other to the east to reach the spot where torbernite had been encountered. This point was on the eastern margin of the breccia, and it became clear that the flakes of torbernite were connected with the edge of the breccia. Thus the northern branch was abandoned and the eastern branch more or less following the eastern contact of the breccia was continued. Flakes of torbernite were encountered all the way but the assays of the samples averaged only 0.07% U_3O_8 with a maximum of 0.16% U_3O_8. The No. 1 winze from the 50 ft. level encountered a fair showing of torbernite at its collar, but it passed almost immediately out of this and continued on flakes of torbernite to a depth of 76 ft., passing to the south of the 100 ft. level. On the suspicion that the ore showing at the collar may have had a flat pitch, an intermediate level (80') was driven from this winze to the east, but no values were encountered although there were flakes of torbernite. No. 2 winze was sunk from the 100 ft. level on the footwall of the lode channel, but for the last ten feet it was sunk too steep and the lode wall was to the south of the winze. A crosscut was driven from the bottom of the winze through the lode channel, which consisted largely of a seam of manganiferous material about 1 ft. thick. Towards the end of the operations a drive was started north from the 50 ft. level to cut any possible parallel lode channels in the hematite breccia near the surface, but all the drive showed was the occurrence of torbernite flakes near the contact of the breccia.

A surface shaft was sunk on an outcrop of hematite breccia where a slickensided shear and a Geiger-Muller 'high' coincided. In the first six feet this shaft passed out of ironstone although the wall continued. The shaft was continued on the shear to a depth of 18 ft. and abandoned.

Altogether seven diamond drill holes, totalling 1249 feet were drilled. The first hole from the 100 ft. level was almost due north (grid north) and horizontal. The second was on the same line but depressed at an angle of 22½°. These holes passed into banded gneiss at about 50 ft. and so two holes farther to the east were fixed. No.3 hole, horizontal, was in hematite breccia for most of its course and several flakes of torbernite were seen in the core. No.5 hole was drilled to the east from the same set-up to test any recurrence of hematite breccia. No.6 hole was designed to cut the hematite breccia near the water-level. With a drill of limited capacity and the behaviour of the breccia at depth uncertain, the hole was awkward to design. The hole passed through kaolin for one foot from 199 to 200 feet which evidently represented the downward continuation of the main shear. No rock similar to the hematite breccia was encountered in the hole, but from 238 ft. to 256 ft. some brecciated silicified gneiss with hematite veinlets and kaolinised felspar was met. This rock may represent the downward continuation of the hematite breccia.

No.7 hole was drilled in place of continuing the 50 ft. level north drive.

(v) Future Development.

The origin of the deposit has been discussed fully elsewhere in this bulletin (Broadhurst 1945). The probability of a payable ore-body occurring depends on the origin of the deposits. It will be seen there are several possible explanations of the origin of the deposits. All of them are connected with the hematite breccia, and an exploration in depth of this feature seems to be more important than even the exploration of the main shear. However, none of the prospecting programmes are very attractive, since it is only one possibility in many that the hematite breccia is the oxidised portion of a payable uranium lode which would be encountered below water level, with perhaps some enrichment at the water level. Owing to the irregular slope of the breccia, the holes would have to be closely spaced, and it is estimated that 8-10 holes totalling 3,000-4,000 feet would be necessary to trace the breccia to water-level, if it persists as far as that. The failure of the No.6 diamond drill hole to strike a hematite breccia definitely similar to that near the surface was discouraging. Altogether, future development on this deposit is not attractive, and certainly nothing should be attempted until drilling has been carried out on some East Painter deposits.

(vi) Results of Sampling.

The sampling on these workings revealed that there was some possible ore from the following sources:-

Dumps. In the dump between the 25 ft. and 50 ft. levels, there is about 100 tons of material containing 0.36% U_3O_8.

Stope Filling. In the stope between the open cut and the 50 ft. level, there would be about 50 tons of about the same grade as the dumps.

Ore. Some ore has been left on the side of the stopes, and would be difficult to extract. There would not be more than 50 tons containing about 0.4% U_3O_8.

Mount Painter No.6 Workings.

Tables of Sampling Results.

SURFACE

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of Samples	Remarks
Dump A (Fine)	Surface dumps	-	-	0.36	Bulk	Fine dump between 25 and 50 feet levels.
Dump A (Coarse)	"	-	-	0.15	"	Coarse dump between 25 and 50ft. levels.
72	"			0,04 0.02	"	Dump between 25ft. level and brace.
71	"			0.05	"	Dump at south of brace.
70	"			0.05 0.05	"	Dump east side of ridge below costean.
69	"			0.04	"	Dump below Sample No. 70
68	Surface	5.0		0.27 0.13	Channel	Cut above open cut.
67	"	4,5		0.04	"	"
66	"	4.5		0.02 0.10	"	"
65	"	5.0		0.07 0.17	"	"
64	"	5.0		0.12 0.07	"	"
63	"	5.0		0.02 0.05	" "	"
62	"	5.0		0.03 0.04	"	"
61	"	5,0		0.05 0.02	"	"

25 FT. LEVEL AND OPEN CUT

Sample No.	Position	Length Feet	Geiger Muller	Assay U3O8%	Type of samples	Remarks
33	Open Cut	1.1		1.44 0.56	Channel	
34	"	5.0		1.10 1.11	"	
35	"	5.0		0.15 0.17	"	
36	"	5.0		0.07	"	
37	"	5.0		0.15	"	
39	25 ft. Level	5.0		0.16 0.21	"	
40	"	5.0		0.11 0.08	"	
41	"	5.0		0.02 0.06	"	
42	"	5.0		nil 0.03	"	
43	"	3.0		0.04 nil	"	
44	West drive	4.0		0.18 0.03	"	

EAST ADIT

Sample No.	Position	Length Feet	Geiger Muller	Assay %U3O8	Type of sample	Remarks
1	Wall of Adit	10.0		0.02	Channel	
2		10.0		0.02 0.04	"	
3	"	10.0		0.03 0.06	"	
4	"	10.0		0.04 0.04	"	
5	"	10.0		0.01 0.01 0.03	"	
6	"	10.0		0.02 0.05	"	
7	"	10.0		0.03	"	
8	"	4.4	0.03	0.04 0.02	"	

32 FT. SUB-LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of sample	Remarks
73		3.0		0.38 0.36	Channel	On lode
73X		3.5		0.07 0.10	"	On mullock on footwall
74		2.0		0.31 0.32	"	On lode
74X		3.0		0.04 0.03	"	Mullock on footwall
75		2.0		0.07 0.02	"	On lode
75X		3.0		0.04 0.02	"	Mullock on footwall
76		2.0		0.04 0.07	"	On lode
76X		3.0		tr. 0.05	"	Mullock on footwall
77		3.0		0.03 tr.	"	On lode

RISE ABOVE 50 FT. LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of sample	Remarks
1		3.0		0.02	Chip	
2		3.5		0.01		
3		4.0		0.02		

50 FT. LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of sample	Remarks
9	Main drive	3.5		0.01	Channel	
10	"	3.6		0.20	"	
11	"	4.1		0.02 nil	"	
21	"	5.4		0.06 0.01	"	
22	"	5.6		0.02 0.02	"	
80	"	4.0		0.25 0.13	"	Floor sample.
81	"	4.0		0.03 0.13	"	" "
82	"	4.0		0.15 0.13	"	" "
28	"	3.9		0.49 0.49		
30	"	4.7		0.40 0.32		
31	"	4.4		0.04 0.06		
12	S.W.Cross-cut	5.0		0.09 0.15		
13	"	5.0		0.01 0.08		
14	"	5.0		0.04 0.05		
15	"	4.4		0.08 0.03		
16	"	5.0		tr. 0.01		
17	"	5.0		0.01 0.01		
18	"	5.0		0.06 0.01		
19	S.W.Cross-cut	5.0		0.09 0.02	Channel	
20	"	6.5		0.08 0.02	"	
23	South Crosscut	5.0		0.16 0.09	"	

50 FT. LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of sample	Remarks
24	South Crosscut	5.0		0.02 0.01	Channel	
25	"	5.0		0.07 0.03	"	
26	"	5.0		0.02 0.01	"	
27	"	4.0		0.07 0.02	"	
29	South cuddy	6.5		0.32	"	
1	"	2.0		0.30	chip	Check sample
2	"	2.0		0.09	"	"
3	"	2.0		0.07	"	"
10	North drive	3.0			bulk	
9	"	3.0			"	
8	"	3.0			"	
7	"	3.0			"	
6	"	3.0			"	
5	"	3.0			"	
4	"	3.0			"	
3	"	3.0			"	
2	"	3.0			"	
1	"	3.0			"	
1A	"	3.0	tr.		"	
2A	"	3	nil		"	
3A	"	3	tr.		"	
4A	"	3	<0.1%		"	
5A	"	3	tr.		"	
6A	"	2	nil		"	
7A	"	5		0.05	"	
8A	"	5		0.01	"	
9A	"	4		0.03	"	
10A	"	3		tr.	"	

50 FT. LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of Sample	Remarks
11A	North Drive	3		0.03	Bulk	
12A	"	2		0.05	"	
13A	"	2		0.27	"	
14A	"	3		0.08	"	
15A	"	2.5		0.12	"	
16A	"	2.5		0.10	"	
17A	"	3		0.10	"	
18A	"	2		0.10	"	
19A	"	3		0.17	"	
20A	"	3		0.04	"	
21A	"	3		0.03	"	
22A	"	3		0.03	"	
23A	"	4		0.06	"	
24A	"	3		0.08	"	
25A	"	2		0.04	"	
26A	"	3		0.10	"	
27A	"	3		0.04	"	
28A	"	4		nil	"	
29A	"	3.5		tr.	"	
30A	"	3.0		tr.	"	

NO. 1 WINZE

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of Sample	Remarks
1		3.3		0.03	Chip	
2		1.6		0.03	"	
3		3.0		0.03	"	
4		3.7		0.04	"	
5		4.0		tr.	"	
6		4.0		tr.	"	
7		6.0		0.06	"	
8		6.7		0.03	"	
9		4.4		tr.	"	

NO. 1 WINZE

Sample No.	Position	Length Feet	Geiger Muller	Assay %U308	Type of Sample	Remarks
10		1.0		tr.	Chip	
11		2.7		nil	"	
12		2.7		tr.	"	
13		5.0		tr.	"	
14		4.0		nil.	"	
15		2.8		nil.	"	
16		4.2		nil.	"	
17		3.6		tr.	"	
18		4.8		nil.	"	
19		4.7		tr.	"	
20		10.0		0.02	"	
21		4.0		0.02	"	
22		4.0		0.02	"	
23		4.0		0.02	"	
24		4.5		tr.	"	
25		4.0		0.02	"	
26		3.5		0.02	"	
27		3.5		tr.	"	
28		3.5		0.01	"	
29		3.5		nil	"	
			80 FT. SUB-LEVEL			
1		4.7	< 0.1%		Chip	
2		4.7	"		"	
3		5.0	"		"	
4		4.2	"		"	
5		4.0	"		"	
6		3.9	"		"	
7		4.3	"		"	
8		3.9	"		"	
9		3.9	about 0.1%	0.02	"	
10		4.0	< 0.1%		"	

80 FT. SUB-LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay $\%U_3O_8$	Type of Sample	Remarks
11		3.6	< 0.1%	0.02	Chip	
12		3.4	"		"	
13		4.2	about 0.1%	0.02	"	
14		3.9	"	0.03	"	
15		3.9	"	0.02	"	
16		3.3	> 0.1%	0.07	"	
17		3.3	about 0.1%	0.02	"	
18		2.2	"	0.06	"	
			100 FT. LEVEL - SOUTH PORTAL			
54	Main drive	3.2		0.03 0.03	Channel	
55	"	4.0		0.01 0.04	"	
56	"	3.2		0.04 tr.	"	
57	"	3.6		0.03 0.01	"	
58	"	3.2		0.01 0.01	"	
59	"	3.4		tr. 0.02	"	
60	"	3.8		0.01 0.03		
1	"	3.3		tr.	Chip	
2	"	3.3		tr.	"	
3	"	3.5		tr.	"	
4	"	4.5		tr.	"	
5	"	5.0		tr.	"	
6	"	4.7		tr.	"	
7	"	4.0		tr.	"	
8	"	5.0		tr.	"	
9	"	3.7		0.02	"	
10	"	3.8		tr.	"	
11	"	3.3		0.03	"	
12	N.W. drive	5.0		0.03	"	Wall sample

100 FT. LEVEL - SOUTH PORTAL

Sample No.	Position	Length Ft.	Geiger Muller	Assay %U_3O_8	Type Sample	Remarks
13	N.W. Drive	5.0		0.06	Chip	Wall sample
1	"	3.1	about 0.1%	0.02	"	
2	"	3.6	"	0.07	"	
3	"	3.7	<0.1%		"	
4	"	3.9	about 0.1%	0.04	"	
5	"	3.8	<0.1%		"	
6	"	4.2	"		"	
7	"	3.8	"		"	
8	"	3.9	"		Chip	
9	"	4.2	"		"	
10	"	3.7	"		"	
11	"	3.8	nil		"	
12	"	3.2	"		"	
13	"	3.7	"		"	
14	"	3.8	"		"	
15	"	3.7	"		"	
16	"	4.0	"		"	
17	"	4.2	"		"	
				100 FT. LEVEL		
48	Main Drive	4.8		tr. 0.03	Channel	
49	"	4.9		0.06	"	
49X	"	3.4		0.04 0.02	"	
50	"	4.6		0.05 0.08		
51	"	3.3		tr. tr.	"	
52	"	3.0		tr. 0.01	"	
53	"	3.3		0.04 0.04	"	
47	South cross-cut	5.0		0.03 0.03	"	

100 FT. LEVEL

Sample No.	Position	Length Ft.	Geiger Muller	Assay $\%U_3O_8$	Type of Sample	Remarks
46	South Cross-cut	5.0		nil nil	Channel	
45	"	5.0		nil nil	"	
1	N.E. Cuddy	7.0		nil 0.03	"	
2	"	4.0		0.12 0.06	"	
1(a)	North Drive	5.0		tr.	Bulk	
2(a)	"	5.0		Nil	"	
3(a)	"	5.0		nil	"	
4(a)	"	5.0		nil	"	
1	"	4.2	nil		Chip	
2	"	4.4	nil		"	
3	"	3.9	nil		"	
4	"	4.0	nil		"	
5	"	4.6	nil		"	
6	"	4.4	nil		"	
7	"	4.2	nil		"	
8	"	4.3	nil		"	
9	"	3.7	nil		"	
10	"	3.8	nil		"	
11	"	4.2	< 0.1%		"	
12	"	4.3	nil		"	
13	"	4.4	nil		"	
14	"	4.2	nil		"	
15	"	4.4	tr.		"	
16	"	3.3	nil		"	
17	"	3.9	nil		"	
18	"	4.7	tr.		"	
19	"	4.0	< 0.1%		"	
20	"	4.8	"		"	
21	"	5.7	"		"	
22	"	5.0	"		"	

100 FT. LEVEL

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of Sample	Remarks
1X	North Drive N.Branch	3.0	tr.		Bulk	
2X	"	3.0	$< 0.1\%$		"	
3X	"	3.0	"		"	
4X	"	3.0	tr.		"	
5X	"	3.0	nil		"	
6X	"	3.0	tr.		"	
7X	"	3.0	nil		"	
1A	"	3.0	nil		"	
2A	"	3.0	nil		"	
3A	"	3.0	nil		"	
4A	"	3.0		nil	"	
5A	"	3.0		nil	"	
6A	"	3.0		nil	"	
1A	North Drive E.Branch	4.0	$< 0.1\%$		Chip	
2A	"	4.0	$< 0.1\%$		"	
3A	"	4.0	$< 0.1\%$		"	
4A	"	4.0		0.05	"	
5A	"	4.0		0.01	"	
6A	"	4.0		0.02	"	
7A	"	4.0		0.06	"	
8A	"	4.0		0.09	"	
9A	"	4.0		0.12	"	
10A	"	3.5		0.16	"	
11A	"	3.5		0.05	"	

NO. 2 WINZE

Sample No.	Position	Length Feet	Geiger Muller	Assay %U_3O_8	Type of Sample	Remarks
1		4.7		0.02	Chip	
1(a)			nil		"	
2		4.2	nil		"	
3		4.0	tr.		"	
4		3.8	nil		"	
5		4.1	nil		"	
6		3.7	nil		"	
7		4.0	nil		"	
8		3.9	nil		"	
9		4.2	nil		"	
10		3.8	nil		"	
11		3.5	nil		"	
12		3.7	nil		"	
150 FT. SUB-LEVEL						
1	W.Drive	3.0	nil		Bulk	
2	"	3.0	nil		"	
1A	S.Cross-cut	3.0		0.03	"	
2A	"	3.0		0.04	"	
3A	"	3.0		0.03	"	
4A	"	3.0		0.02	"	
SURFACE SHAFT						
	Depth Ft.					
1	6	5.0		tr.	Chip	
2	9	5.0		nil	"	
3	12	4.0		tr.	"	

(vii) Diamond Drill Logs.

Diamond Drill Hole No. 1

0' - 21'	Fine Hematite breccia, felspathic fragments.
21' - 35'	Specular hematite and kaolin. At 35' possible wall.
35' - 52'	Siliceous gneiss. Broken surface reveals only quartz but smooth surface shows felspar remnants. All sheared with small amount ironstone veins. At 43ft. some quartz vughs and kaolin seams with some slickensides. At 52ft. many slickensided faces and rock is broken; probably fault plane.
52' -101'4"	Rocks richer in mica, occurring in parallel bands probably banded gneiss. At 73ft. small quartz vughs with flecks of torhernite (small core recovery). At 88ft. - 1ft. of silicified gneiss. At 92ft. brecciated rock with seams of quartz crystals. (Banded gneiss shows signs of injection of igneous material, from thin veins to those 1ft. wide.)
101'4"- 154'	Banded gneiss with injected igneous material and much biotite. At 110ft. 1in. seam of clay. At 125ft. about 6in. of clay with slickensided surfaces: probably a shear. At 128ft. 1in. of clay.
154' - 159'	Siliceous ferruginous breccia injected with much igneous material.
159' - 187'	Kaolinised pegmatitic granite.
187' - 201'	Banded gneiss.

Hole completed at 201'.

Diamond Drill Hole No. 2

0' - 10'	Siliceous breccia.
10' - 11'6"	Argillaceous ironstone.
11'6"- 13'	Kaolin and specular hematite.
13' - 14'	Siliceous breccia.
14' - 17'	Argillaceous ironstone.
17' - 20'	Siliceous breccia.
20' - 23'	Hematite breccia.
23' - 25'6"	Siliceous breccia.
25'6"- 27'	Argillaceous ironstone.
27' - 28'	Kaolin and specular hematite.
28' -- 32'	Siliceous breccia.
32' - 35'	Hematite breccia.
35' - 43'	Crush zone of kaolin and fragmented rocks.
	At 43' probable wall.
43' - 46'	Silicified gneiss.
46' - 60'	Banded gneiss injected with much igneous material.
60' - 63'	Mainly granitic injection gneiss.

63' - 112'9"	Banded gneiss with varying amounts of injection averaging about 50% igneous material.
	At 87ft. possible shear. At 106ft. quartz 6in. wide.

Hole completed at 112'9"

Diamond Drill Hole No.3

0' - 4'	Siliceous breccia
4' - 11'	Siliceous breccia.
11' - 31'	Hematite
	At 24' and 26' flecks of torbernite.
31' - 35'	Siliceous breccia.
35' - 45'	Hematite breccia.
	At 44' torbernite seam abour 1/16" thick.
45' - 52'	Brecciated quartzite.
	At 45'-48' torbernite flecks through core. 51'-52' slickensided faces with torbernite. Probable shear at 52'.
52' - 110'	Hematite breccia. Breccia much firmer and much more complete replacement by iron than before, in places appearing entirely hematite.
	52'-53' slickensided faces with torbernite.
	At 80' hole along joint with torbernite. At 90'2" torbernite flecks and vesicular texture. Possible shear. At 97' 1ft. of sheared quartzite with torbernite along joint plane. At 107' slickenside with flecks of torbernite.
110' - 120'4"	Hematite breccia, development of hematite as intense as above.
120'4"-161'5"	Hematite breccia showing about 20% kaolin.
	At 133' sulphide stain. 147'-149' some slickensides with small quartz vughs.
161'5"-163'9"	Silicified gneiss, rather slickensided. Possible shear.
163'9"- 200'	Breccia with specular iron and fine grey ground mass.
	At 198' and 200' about 4" of silicified gneiss.
200' - 221'11"	Mainly hematite breccia, with bands, average width 6" of silicified and biotite-rich gneiss.

Hole completed at 221'11"

Diamond Drill Hole No. 4

0' - 3'	Hematite breccia
3' - 8'6"	Siliceous breccia.
8'6" - 13'6"	Specular kaolinitic hematite breccia.
13'6" - 15'6"	Silicified gneiss with vughs filled with mammillary quartz.
15'6" - 24'	Alternating silicified gneiss and specular kaolinitic hematite breccia. At 23'6" torbernite along joint.
24' - 28'	Silicified gneiss.
28' - 32'	Mixture of hematite and silicified gneiss.
	At 28ft. small amount of torbernite along joint.

32'	58'6"	Slightly brecciated silicified gneiss.
58'6"	59'6"	Hematite breccia.
59'6"	64'	Silicified gneiss.
64'	69'	Hematite breccia.
69'	70'6"	Siliceous breccia.
70'6"	72'	Hematite breccia.
72'	79'	Siliceous breccia with biotite flakes
79'	80'	Hematite breccia
80'	81'6"	Silicified gneiss.
81'6"	105'	Hematite breccia. At 88'6" torbernite along joints.
105'	105'6"	Silicified gneiss.
105'6"	107'4"	Hematite breccia.
107'4"	115'1"	Hematite breccia with both clayey and dense hematite matrix. At 115' ferruginous quartzite.

Hole abandoned at 115'1" on account of caving ground.

Diamond Drill Hole No. 5

0'	5'	Silicified gneiss.
5'	6'	Soft hematite breccia.
6'	9'6"	Silicified gneiss.
9'6"	12'6"	Hematite breccia.
12'6"	14'	Silicified gneiss.
14'	15'	Hematite breccia.
15'	27'	Silicified gneiss.
27'	29'	Hematite breccia.

At 27ft. angle of contact about 30°.
Considerable clay near contact.

29'	31'6"	Silicified gneiss.
31'6"	33'	Siliceous breccia.
33'	34'	Silicified gneiss.
34'	35'	Decomposed pegmatitic granite (?) Clay and decomposed iron ore.
35'	37'	Biotite rock. Biotite at 30° to hole.
37'	38'6"	Hematite breccia.
38'6"	43'	Silicified gneiss.
43'	44'	Soft hematite braccia.
44'	45'	Silicified gneiss.
45'	48'	Much sheared silicified gneiss.
48'	60'	Silicified gneiss with biotite bands and sheared in places.
60'	62'6"	Reddish brecciated rock with blebs of iron ore up to ¼" diameter.
62'6"	83'	Siliceous breccia with some biotite.
83'	100'	Silicified gneiss with occasional quartz vughs and biotite rich bands.

Hole completed at 100'

Diamond Drill Hole No. 6

0'	100'	Banded gneiss.
100'	103'	Clay mineral with manganiferous ironstone
103'	194'6"	Banded gneiss.
194'6"	199'	Ferruginous and biotite-rich sheared gneiss.
199'	200'	Kaolin, probably fault material.
200'	238'	Gneiss, showing various stages of granitisation.
238'	246'	Silicified gneiss, veined with hematite and with kaolin in vughs.
246'	256'	Brecciated gneiss with kaolinised felspar and secondary hematite.
256'	279'	Silicified gneiss with considerable biotite and veins of kaolin.
279'	294'	Silicified gneiss, only occasional veins of kaolin and small proportion of biotite.

294'	-	330'8"	Gneiss, rather silicified with local developments of biotite. Some hematite veinlets.

Hole completed at 330'8".

Diamond Drill Hole No. 7

0'	-	14'	Hematite breccia.
14'	-	16'	Brecciated silicified gneiss, ferruginous matrix.
16'	-	28'	Silicified gneiss.
28'	-	46'	Brecciated silicified gneiss, ferruginous matrix.
46'	-	55'	Mixture of lamprophyric rock and silicified gneiss.
55'	-	58'	Brecciated silicified gneiss, ferruginous matrix.
58'	-	136'	Hematite breccia.
136'	-	142'	Chiefly silicified gneiss with some hematite breccia.
142'	-	147'	Hematite breccia.
147'	-	167'11"	Brecciated pegmatitic granite.

Hole completed at 167'11".

E. No. 7 Workings

by

E. Broadhurst, Geological Survey of South Australia.

(Plan 2997, Sheet 19 illustrates this report).

(i) Location

The No. 7 Workings are situated on the spur to the west of the No. 6 Workings, on the opposite side of Radium Creek being about 500 feet from the creek.

(ii) Geology

The rocks consist of sediments which have become altered to gneisses but which still show bedding planes. Some of the bedding is regular, but in the vicinity of the workings it is contorted and shows evidence of much movement.

(iii) Workings

The main workings are on top of the spur, consisting of a costean across the top of the spur and a shaft sunk 40 feet from the surface, or 29 feet from the bottom of the costean. The shaft was sunk on a north-south shear, on which the costean had been cut, dipping to the west at about 80°. At a depth of 30 feet the shear becomes indistinct. At a depth of 20 feet from the surface a drive was extended 11 feet along the shear, and at the bottom of the shaft a cross-cut was driven 10 feet to the west.

The bottom of the shaft was cleaned out and examined during the recent operations, but the whole of the bottom level consists of normal gneiss.

At the north end of the costean a shaft 11 feet deep has been sunk, and a small heap of ore containing torbernite has been obtained from this shaft.

Farther down the spur to the east of the main shaft there are two old open cuts. These were formed from the working of seams of manganiferous ironstone up to one foot in width, containing torbernite. The torbernite apparently was confined to the manganiferous ironstone, which occurred in zones of contorted gneiss.

800 feet to the south of the main No. 7 shaft, there is an occurrence of uranium ore which was worked in open cuts on the face of a bold outcrop. These workings were known as the Minerva Heights. Between the main No. 7 shaft and the Minerva Heights workings, the gneisses have a fairly regular strike and dip, but about 30 feet to the north of the latter workings the gneisses become contorted. The torbernite was associated with a strong shear plane striking at 100° and dipping 80° N, along the course of which manganiferous ironstone occurred. As with the No. 7 workings, the torbernite was associated with the manganiferous ironstone. Only odd specimens can now be seen.

F. No.8 Workings

by

C.J. Sullivan. Geologist, Mineral Resources
Survey

(Plan 3215, Sheet 39 illustrates this report).

(i) Situation.

This prospect is situated approx. 20 chn. north of the Main Camp.

(ii) Geology and Workings

Torbernite was discovered in a sheared zone in granitic rocks very little hematite being present. The zone varies from 5 to 10 feet in width and has an east-northeasterly strike. The torbernite is not associated with hematite nor with a particular crushed zone. The zone has been tested by means of open cuts 3 to 8 feet in depth over a length of approximately 150ft. At the eastern end of the zone, a shaft has been sunk to a depth of 40 feet and a drive from it extended for a distance of 20 feet to the west along the shear zone.

The workings were carefully inspected but traces only of torbernite could be found. Four samples cut from the most promising looking places assayed a trace of uranium only.

There are intrusions of pegmatite in close proximity to the occurrence and from observations made elsewhere, it is considered possible that the uranium was derived from these intrusive rocks. It may have been carried into the sheared zone by ground waters.

G. Locality "G"

by

R.C. Sprigg, Geological Survey of South
Australia.

Extensive outcrops of "ironstone" are located on the western extremities of Radium Ridge at a locality marked "G" on the geological plans. They occur within a zone of crushed granitic rock. The main "ironstone" formation is more or less semi-circular in plan and dips at 30-40° to the east towards a centre, in a manner suggestive of a cone dyke. Tabular outcropping masses of the "ironstone" are preserved on steep slopes with which they conform.

Geiger Muller "highs" have been recorded in several instances on the northern and western extremities of the formation. They are all located at points of maximum elevation within the ironstone mass. The hematite is the main constituent of the "ironstone" body and has a more massive form at the G.M. "high" points. It has a manganiferous coating at the surface.

No uranium minerals were noted in the "ironstone" and it is not known whether monazite is present.

H. Locality "J"

by

R.C. Sprigg, Geological Survey of South Australia.

Mt. Painter is a centre of extensive crushing and on its south-western shoulder there is a large mass of vughy quartz and siliceous ironstone which is known as locality "J" and marked as such on the geological map. There is also brecciated material on the north-western face of Mount Painter and this area appears to be favourable for the occurrence of uranium ores and warrants detailed examination. Although no uranium mineralization was discovered in a short visit to the locality, Mr. W. B. Greenwood is reported to have found torbernite in this area.

2. EAST PAINTER GROUP

A. General Statement

by

E. Broadhurst, Geological Survey of South Australia.
(Plan 2949, sheet 20, illustrates this report)

(i) Introduction

The East Painter field is known as a locality distinct from Mt. Painter, due principally to the geographical isolation of the two camping centres from each other. Although the Mt. Painter camping area and Greenwood's Camp at East Painter are less than three miles apart, there is a high ridge (the continuation of Mr. Painter) between them and their distance apart by road is 50 miles.

Previous to the present investigations, there were some small workings known on the east field as 1E, 2E, etc. These workings were all small, smaller than the Mt. Painter workings. Previously the water supply had been a major difficulty, but with the construction of the road and the striking of a water supply in the bore at the mouth of the East Painter Creek gorge this difficulty was overcome. Most of the work done in the present investigation was on fresh deposits found in the Greenwood's Camp area.

(ii) Geology (East Painter or Greenwood's Camp area)

The rocks in this area consist of sedimentary gneisses and their brecciated and granitised remnants. The readings of the strikes and dips of the bedding planes of the gneisses have fairly regular results. The strikes and dips indicate that selective brecciation seems to have occurred, by which is meant that certain beds, or groups of beds, have yielded more easily than others to brecciation, so that the contact of the brecciated and unbrecciated rocks follows the strike of the bedding on the surface. This is particularly clear in the gully containing the Nos. 1 and 2 Workings, where a bed 60 feet wide is brecciated and replaced by hematite, between two fairly massive beds of gneiss which show only minor brecciation.

The breccia shows all stages of replacement by hematite, from practically hematite-free breccia to rock consisting almost entirely of hematite. The occurrence of hematite is very irregular, and the breccia rich in hematite seems to occur in scattered disconnected area.

The area, mainly in the brecciated zones, is traversed by a system of shears which show up very prominently as ravines passing across the cliffs of breccia, or on the sides of the outcrops forming cliff faces. Usually a vein of manganiferous ironstone, about 1ft. wide, occurs along the course of the shears. The shears are all practically vertical, and their strikes seem to fall into parallel groups - N.W. - S.E., E.-W., and N.E.-S.W. Probably these groups represent different periods of stress to which the area was subjected.

While the workings will be considered individually later, it is convenient to consider their relation to the general geology at this stage. All the occurrences of torbernite are associated with the breccia. Curiously, the best showings of torbernite do not seem to be in the main outcrop of the breccia. Geiger Muller surveys over the main outcrop have shown radioactive 'highs' which generally coincide with the more intense developments of hematity. Napping over the outcrops generally reveals occasional flakes of torbernite over these 'highs'. The principal discovery of torbernite is in the No. 5 Workings, which ovvur on a shear striking at 345°, and is just at the foot of the clipp forming the main outcrop. In these workings the torbernite is associated with the shear planes. The Nos. 1 and 2 workings are of a different nature, and occur in a bed of hematite breccia between granitised sediments. Here there are no shears and the flakes of torbernite are present in 'leached cavities' and along irregular joint planes in the hematite breccia formation. At the surface these occurrences are very irregular, although the hematite breccia is uniform.

(iii)<u>Geiger Muller Survey</u>.

A Geiger Muller survey was undertaken in the Greenwood's Camp Area. One portion of this survey was in the gully containing the Nos. 1 and 2 Workings, and showed that there were two areas of higher radiation corresponding to these two deposits. Another portion was carried out over the cliffs of breccia to the N.E. of Greenwood's Camp, and the twelve points showing the highest radiations were marked. A detailed description of the survey is given in the section of this bulletin dealing with the geophysical investigations, but it is convenient to give here a table showing the geological descriptions of these twelve localities with their 'radiation equivalents' and actual contents of U_3O_8.

Locality	Geological Description	% U3O8	
		Geiger Muller Estimate	Chemical Assay
G.M. 1	Near shear, strike 30° and dip 75°E. Dense hematite with boxwork cavities and very occasional flakes of torbernite.	< 0.1	Nil
G.M. 2	On cliff face in cave. Brecciated granitised sediments, with small proportion of hematite in matrix. Some veins of solid hematite with manganese minerals pseudomorphs.	0.11	0.16
G.M. 3	On same shear as G.M. 1. Similar rock to G. M. 1 Note-Between G.M.1 and G.M.3 brecciated granitised sediments occur.	0.031	0.01
G.M. 4	Small irregular veins, up to 1ft. wide, of dense hematite with numerous boxwork cavities. In brecciated granitised sediments with manganiferous facings.	0.023	Nil

Locality	Geological Description	% U_3O_8	
		Geiger Muller Estimate	Chemical Assay
G.M. 5	An outcrop of granitised sediments with manganiferous coating. Hematite breccia within 3ft., with manganese replacing some mineral, possibly magnetite.	$<$0.1	0.02
G.M. 6	Irregular replacement of brecciated granitised sediments with hematite, which contains numerous boxwork cavities and some torbernite flakes.	$<$0.1	0.01
G.M. 7)	All in brecciated granitised sediments being replaced by hematite. The high points seem to depend on the amount of replacement by hematite.	$<$0.1	0.04
G.M. 8)		$<$0.1	0.02
G.M. 9)		$<$0.036	0.03
G.M. 10)		0.14	0.20
G.M. 11	More intense replacement of brecciated granitised sediments by hematite than G.M. 7 - G.M. 10. Some torbernite flakes and boxwork cavities.	0.036	0.03
G.M. 12	Granitised sediments with little brecciation and only small amount of alteration by hematite. No distinctive geological features from surrounding rocks.	0.018	Nil

NOTE : " 0.1 signifies less than 0.1 per cent but not accurately determined

B. Nos. 1 and 2 Workings, East Painter (Greenwood's) Camp

by

E. Broadburst, Geological Survey of South Australia.

(1) Description

Up the length of the gully formed by the brecciated bed between two prominent outcropping series of beds, there are a number of torbernite occurrences which are grouped as Nos. 1 and 2 Workings. One group of occurrences is near the bottom of the gully and is known as the No. 1 Workings, and another group, near the top of the gully, is known as the No. 2 Workings. The surface workings are all small, the deepest shaft being 6 feet. The majority of the workings are merely pot-holes formed by a few shots being fired to disclose the nature of the rock on the surface. Other pot-holes were formed by removing the surface soil to expose the underlying rock. An adit, 50 feet in length from the portal, was driven underneath the No. 1 Workings.

The torbernite occurs in two forms:

(1) As 'leached cavities', which are boxwork cavities in dense hematite ironstone breccia. A series of specimens of these leached cavities were examined by Stillwell and Edwards (1945) and their conclusions are given in full in his report which appears earlier in this Bulletin. Briefly, his conclusions are that the cavities originally consisted of torbernite existing as plates extending across the cavity, thus making a boxwork structure. The torbernite boxwork near the surface becomes replaced to some extent by limonitic or phosphatic material, tending to diminish the uranium content of the rock near the surface.

To what extent this process has been carried out cannot be determined. Cavities with only an encrustation of lepidocrocite (limonite) are indicative merely of the former presence of pyrite.

(ii) As 'torbernite paint', which is sometimes thick enough to be described as thin seams. This torbernite occurs along joint planes principally in the ironstone.

There is probably no material in these occurrences which can be regarded as ore. However, if all the leached cavities had originally been occupied by torbernite, the unleached material would probably be payable ore. For this reason the deposits are worth further investigation.

A disturbing feature of these deposits is the size of the outcrops. The outcrops of torbernite-bearing ironstone were small and irregular and it is clear that any torbernite occurrence would be difficult to follow in a shaft from the surface. For this reason adits along the course of the hematite breccia were recommended to disclose the nature of the breccia at a reasonable depth beneath the surface.

The Geiger Muller survey in this area has shown that two definite areas of radioactivity exist, corresponding to the Nos. 1 and 2 Workings, and between them the radioactivity falls to normal. Two adits were recommended, one for each of these areas. The lower of these adits, at the No. 1 Workings, was driven 50 feet from the portal and reached a point 50 feet below the best concentration of torbernite exposed on the surface. The drive was in hematite breccia throughout its course. The samples from the adit are given in the table below. No walls or geological features likely to cause a concentration of values were present in the adit, so the assay values given are likely to be representative of the complete bed of breccia. All the samples were tested by the Geiger Muller before being sent for chemical assay, and those samples indicating a U_3O_8 content of less than 0.1% were not forwarded for assay. For this reason samples 4, 5, 11 and 12 were not assayed.

Since there was no distinct torbernite in the breccia the assay results from the original bulk samples, although not high, were higher than expected. For this reason check channel samples were taken along the back of the adit. The assay results from these samples were much lower than the original bulk samples. The two sets of assay results are shown in the following tables:

(ii) Sampling Results

The results of Sampling are set out in the following tables:

ADIT BULK SAMPLES

Sample No.	Distance from Portal		Length	Geiger Muller % U_3O_8	U_3O_8 %	Remarks
	From	To				
1	12	15'6"	3'6"	About 0.1	0.11	
2	15'6"	18'6"	3'	" "	0.08	
3	18'6"	20'6"	2'	" "	0.12	
4	20'6"	23'	2'6"	<0.1	-	Not assayed
5	23'	25'	2'	<0.1	-	" "
6	25'	28'6"	3'6"	<0.1	0.17	
7	28'6"	32'9"	4'3"	"<0.1	0.17	
8	32'9"	36'	3'3"	"0.1	0.22	
9	36'	39'	3'	About 0.1	0.13	
10	39'	42'	3'	" "	0.15	
11	42'	45'	3'	<0.1	-	Not assayed
12	45'	50'	5'	" 0.1	-	" "

CHECK CHANNEL SAMPLES

Sample No.	Distance from Portal		Length	U_3O_8 %	Remarks
	From	To			
1	0	10'	10'	0.02	
2	10'	20'	10'	0.10	
3	20'	30'	10'	0.10	
4	30'	40'	10'	0.04	
5	40'	50'	10'	0.02	
			4'6"	0.01	Sample at face of adit.

(iii) Conclusions

None of the results of the sampling indicates the presence of ore containing not less than 0.25% U_3O_8. However, the nature of the breccia with its leached cavities indicates that there may exist a richer deposit at a depth where the leached cavities become occupied with torbernite instead of the ferric compounds which now largely occupy them. The large extent of the breccia makes this the most suitable deposits for examining the behaviour of the hematite breccia in depth by means of a diamond drilling campaign.

It is recommended that if diamond drilling is to be carried out with this object in view that operations should commence on this deposit.

C. No.3 Workings, East Painter (Greenwood) Camp.

by

E. Broadhurst, Geological Survey of South Australia.

The locality of this deposit is shown on the plan of the Greenwood's Camp Area. It is one of the old workings and consists of a shaft 8 feet deep. During the present operations the shaft was cleaned out. The sides and bottom of the shaft was found to be in hematite with knife-like seams of torbernite. In no place could uranium ore be said to occur.

D. No.5 Workings, East Painter (Greenwood) Camp

by

E. Broadhurst, Geological Survey of South Australia.

(Plans 3191 and 3195, Sheets 21 & 22 illustrate this report)

(i) Development

The development on this lode consists of an open cut (which is now filled with broken ore), a shaft 40 feet deep, and a drive at the 40 feet level in the shaft.

(ii) Geology

The outcrop of the lode is about 20 ft. from the main mass of breccia, on a shear striking at 325°. The shear showed at the surface as a wall of hematitic rock with the shear wall showing on the eastern side. The lode outcrop consisted of veins of torbernite up to 3/4 of an inch thick traversing the hematite adjacent to the shear. An open cut of an average depth of 4½ feet was taken adjacent to the wall, the mineral from the cut consisting largely of surface loam, and bulk sample of the material from the open cut assayed 0.33% U_3O_8.

At the top of the open cut, opposite the best showing of torbernite in the 'hematite wall', a shaft was sunk to a depth of 40 ft. in decomposed iron-stained granitic material showing occasional quartz veins. The material in the shaft was rather disappointing.

In the drive, however, more torbernite was encountered. The drive was in the same material as the shaft, but the torbernite was associated with shears along which clayey ironstone occurred. These shear planes were all sub-parallel with an average strike of 325° and a practically vertical dip. On the surface the shears were seen to echelon to the right, and there is a suggestion that the same is occurring on the 40 ft. level.

(iii) Ore Occurrence

The difficulties associated with sampling this deposit have been discussed in a separate report, and only the results will be given here.

The open cut, from which about 25 tons were extracted, gave an assay of 0.33% for a bulk sample.

The first 20 feet of the shaft, or about 15 feet below the open cut, gave an assay of 0.11% U_3O_8 from a bulk sample.

The next 14 feet to the top of the drive gave an average assay value of 0.17% U_3O_8, which shows a close agreement with the assay of 0.15% U_3O_8 from the bulk sample.

A series of check samples of the shaft gave an average assay value of 0.26% U_3O_8.

The probable true value of the material from the shaft is the weighted average of the above results, i.e.

15ft. assay value 0.11%
14 ft. " " 0.15% and
29 ft. " " 0.26%

giving a weighted average of 0.19%. The amount of material from the shaft would be about 25 tons.

In the drive the quantity of ore proved amounts to 8 tons per vertical foot.

The weighted average of the assay results of the chip samples is 0.38% U_3O_8.

The weighted average of the assay results of the bulk samples is 0.77% U_3O_8.

Probably the former are the more correct.

(iv) Conclusions.

On this deposit operations were stopped at a stage where its possibilities were merely indicated. The 40ft. level had not passed out of ore in the ends of the drives, and the behaviour of the ore body beneath this level is quite unknown. At the present stage all that can be said of the deposit is that at a depth of 40 feet there is at least 8 tons per vertical foot of ore with an assay value probably of 0.38% U_3O_8. At this depth values seem to be increasing with depth, but whether they will continue to do so cannot be stated.

SURFACE SAMPLES.

Sample No.	Position	Length	Assay U_3O_8	Remarks
4	Open Cut		0.33	Bulk sample of open cut.
6a	Outcrop	3'	0.18	4ft.-7ft. W. of shaft.
7	"	3'	0.31	1ft.-4ft. W. of shaft.

SHAFT CHIP SAMPLES

Sample No.	Depth Ft.	Length Ft.	Assay % U_3O_8	Remarks
1	6	3	0.17	North end of shaft.
2	8	3	0.33	North end of shaft.
3	10.5	3	0.17	North end of shaft.
5	13.5	3	0.25	North end of shaft.
9	16	3	0.16	North end of shaft.
8	16	3	0.14	South end of shaft.
6	17	3	0.11	North end of shaft.
11	19	3	0.14	North end of shaft.
10	19	3	0.07	South end of shaft.
12	22	3	0.36	North end of shaft.
13	22	3	0.05	South end of shaft.
14	25	3	0.15	North end of shaft.

SHAFT CHIP SAMPLES

Sample No.	Depth Ft.	Length Ft	Assay % U_3O_8	Remarks
15	25	3	0.06	South end of shaft.
16	28	3	0.29	North end of shaft.
15(a)	28	3	0.14	South end of shaft.
17	31	3	0.18	North end of shaft.
18	31	3	0.14	South end of shaft.
20	34	3	0.12	North end of shaft.
19	34	3	0.20	South end of shaft.
24	37	3	0.10	North end of shaft.
25	37	3	0.31	South end of shaft.
31	40	3	0.65	Bottom of shaft.

SHAFT BULK SAMPLES

Sample No.	Depth		Length Ft.	Assay % U_3O_8	Remarks
	From	To			
12A	6	20	14	0.11	
26	20	34	14	0.15	
22	34	35.5	1.5	0.20	
23	35.5	37.5	2	0.16	
27	37.5	40	2.5	0.24	

SHAFT CHIP CHECK SAMPLES

Sample No.	Depth Ft.	Length Ft.	Assay % U_3O_8	Remarks
1	9	3	0.26	N. end of shaft.
2	9	3	0.22	S. end of shaft.
3	14	3	0.47	N. end of shaft.
4	14	3	0.30	S. end of shaft.
5	19	3	0.25	N. end of shaft.
6	19	3	0.27	S. end of shaft.
7	24	3	0.12	N. end of shaft.
8	24	3	0.12	S. end of shaft.
9	29	3	0.26	N. end of shaft.
10	29	3	0.11	S. end of shaft.
11	34	3	0.32	N. end of shaft.
12	34	3	0.39	S. end of shaft.

DRIVE CHIP SAMPLES

Sample No.	Distance from Shaft.	Length	Assay % U_3O_8	Remarks
34	0ft. S.	2.5	0.24	W. side
35	"	2.5	0.06	E. side
38	2ft. S.	3.0	0.24	
43	4ft. "	1.5	0.77	E. side
43A	4ft. "	1.5	1.06	W. side
56	6ft. "	1.5	0.86	E. side
56A	6ft. "	1.5	0.35	W. side
61	8ft. "	1.5	0.53	E. side
61A	8ft. "	1.5	0.17	W. side
32	0ft. N	2.5	0.10	W. side
33	0ft. "	2.5	0.23	E. side
39	2ft. "	3.0	0.13	
41	5ft. "	1.5	0.20	E. side
41A	5ft. "	1.5	0.46 0.19	W. side
42	7ft. "	1.5	0.19	E. side
42A	7ft. "	1.5	0.37	W. side
46	9ft. "	1.5	0.13	E. side
46A	9ft. "	1.5	0.15	W. side
49	11ft."	1.5	0.40	E. side
49A	11ft."	1.5	0.42	W. side
51	12ft."	1.5	0.37	E. side
51A	12. ft	1.5	0.32	W. side
53	14ft."	1.5	1.65	E. side
53A	14ft.	1.5	0.32	W. side
54	16ft."	1.5	0.68	E. side
54A	16ft."	1.5	0.59	W. side
58	17½ft.	1.5	0.88	E. Side
58A	17½ft.	1.5	1.00	W. side
63	19½ft.	1.5	0.26	E. side
63A	19½ft.	1.5	nil	W. side

DRIVE BULK SAMPLES

Sample No	Distance from shaft		Length Ft.	Assay % U_3O_8.	Remarks
	From	To			
28	0'	3'E	3	0.21	
29	3'E	6'E	3	0.22	
30	0'	2.5'W	2.5	0.05	
37	0'	2' S	2	0.30	
44	2'S	4'S	2	1.41	
57	4'S	6'S	2	2.13	
62	6'S	8'S	2	1.45	
36	0'	2'N	2	0.29	
40	2'N	5'N	3	0.69	
45	5'N	7'N	2	1.91	
47	7'N	9'N	2	0.62	
48	9'N	11'N	2	1.91	
50	11'N	12'N	1	0.68	
52	12'N	14'N	2	0.74	
55	14'N	16'N	2	0.49	
60	16'N	17.5'N	1.5	0.66	
64	17'.5N	19.5'N	2	0.50	

DRIVE CHECK CHANNEL SAMPLES

Sample No.	Distance from shaft	Length	Assay %U_3O_8	Remarks
1	3'N	3ft.	0.17	Back of drive.
2	3'N	3ft.	0.19	Floor of drive.
3	5'N	3ft.	0.19	Back of drive.
4	5'N	3ft.	0.11	Floor of drive.
5	7'N	3	0.17	Back of drive.
6	7'N	3	0.25	Floor of drive.
7	9'N	3	0.17	Back of drive.
8	9'N	3	0.14	Floor of drive.

E. No. 1E Workings, East Painter (Greenwood) Camp

by

C. J. Sullivan, Geologist, Mineral Resources Survey.

This prospect is situated on a steep hill-slope, approximately 360 feet north-east of the old well at East Painter Camp. Moderately ferruginous crushed granitic rock has been open cut over a length of 20 feet and a width of 7 feet. No uranium minerals were noted and it is believed that the original prospectors may have been misled by the presence of the pale blue copper-bearing mineral, turquois.

F. No. 2E Workings

by

C.J. Sullivan, Geologist, Mineral Resources Survey

These workings are situated on the right bank of Heighty Creek, approximately 500 feet on a bearing of 160° meg. from the dry well.

An adit has been driven on a band of moderately sheared ferruginous granitic material, containing traces of torbernite. The adit is on a bearing of 200° meg. The distance from the beginning of the approach cutting to the face of the adit is 30 feet and the distance from the portal to the face is 15 feet; the width of the adit is 3 to 5 feet. From near the face, a crosscut has been extended for 6 feet to the eastward. A chip sample was taken around the walls of the workings, the assay result being 0.03 per cent U_3O_8.

The prospect is not considered likely to produce significant quantities of ore of higher grade.

G. No. 4E Workings

by

C. J. Sullivan, Geologist, Mineral Resources Survey.

This prospect is situated approximately 1700 feet west-northwest from the East Painter Camp. A shallow pit has exposed small amounts of torbernite in a small body of hematite surrounded by crushed granite. A representative sample of the material from the pit assayed 0.06 per cent U_3O_8.

H. Bentley Greenwood Deposit

by

C. J. Sullivan, Geologist, Mineral Resources Survey.
(Plan 3204, Sheet 36, illustrates this report).

(i) Situation

The deposit is situated approximately 150 yards north of Heighty Creek which flows past East Painter (Greenwood) Camp and is about three quarters of a mile up the creek from the camp. The East Paint road follows Heighty Creek to a point approximately 10 chains further up the creek from this deposit.

(ii) Geology

The rocks surrounding the deposit are nearly all granitic in typs but they contain masses which are relatively coarse in grain and are often highly felspathic.

These masses may be dykes but their boundaries are rather indefinite and they may merely represent segregations in the granite. Traces of sedimentary structures were found in two places, but they are not very definite. Sedimentary rocks occur in the east bank of a creek approximately 200 yards south south-east of the deposit.

The rocks have a general east-west grain and are to some extent arranged in bands parallel to this direction. Lines of shearing also has this trend and dips to the south at angles of 60 to 70 degrees.

(iii) Uranium Minerals

Torbernite has been found in a band of hematite situated on a spur approximately 200 feet above creek level. The hematite trends east and west for a distance of 140 feet and has a width ranging from 15 to 35 feet. It occupies a slightly crushed zone. On its northern edge, the hematite is bounded by a shear, along which torbernite has been localised, apparently by supergene processes. Five pits have been sunk along this shear over a distance of 65 feet to depths of 2 to 5 feet. They reveal manganese dioxide and hematite coated material showing torbernite along the planes and cracks. A composite sample taken from the pits assayed 0.36 per cent U_3O_8. The width of ore bearing material exposed in the pits ranges from 6 inches to 1 foot. An adit has been driven northerly to the wall of the shear at a point approximately 10 feet below the surface. From this point the sheared zone was explored by means of a drive extending 15 feet to the west. It was found that the torbernite at this level amounted to traces only, and that the hematite content had fallen proportionately. Hence ir will be seen that the amount of ore available in this section is not likely to be more than 10 to 20 tons of ore with a grade not exceeding 0.5% U_3O_8.

At the western end of the hematite, an open cut 13 to 15 feet wide has been extended into the hill for a distance of approximately 10 feet. It attains a maximum depth of 8 feet. In this cut, the hematite is seen to form a capping 1½ to 3 feet in thickness. It rests on relatively uncrushed highly felspathic rock. The hematite body contains some torbernite in its lower portions, but close to the surface, much of the torbernite has been leached out. The cavities from which it has been leached, contain a boxwork of gypsum with a very distinctive pattern. Several stages in the replacement of the torbernite by the gypsum were noted. Specimens were submitted to Dr. Stillwell who confirmed the fact of the leaching of the torbernite. The latter is the metatorbernite variety.

In the felspathic rock below the hematite, torbernite occurs in cavities completely enclosed in solid felspar. The grade of this material would be low. It is possible that this torbernite has resulted from the oxidation of a primary uranium bearing mineral, which formerly occupied the cavity. However, the torbernite may replace a non-uranium bearing mineral.

(iv) Testing.

Three test pits were sunk at the points shown on the plan. These were located at points to the south of the ironstone outcrop showing relatively high radioactivity readings. The test pits revealed only weathered granite below the soil cover. The high readings were evidently due to uranium minerals which had rilled down hill from the "ironstone" outcrop.

No other testing was undertaken and the deposit is not considered likely to produce important quantities of ore with a grade 0.25 to 0.5% U_3O_8.

I.Similar Greenwood Deposit

by

C.J. Sullivan, Geologist, Mineral Resources Survey.

(Plan 3205, Sheet 35 illustrates this report)

(i) Situation

This deposit is situated approximately 1½ miles north of Mt. Painter. Access is gained from the East Painter road by way of a camel pad.

(ii) History

This locality is marked No. 15 on Mawson's map and is described in his report (Mawson 1945). There were no workings on the deposit at the time of the first inspection in 1944 and it was considered advisable to made geological and geophysical surveys in order to help in the formulation of a prospecting campaign.

(iii) Geology

The accompanying plan and section illustrate the geology of the deposit. It should be noted that most of the eastern section of the area mapped is covered with detritus and hence information shown from this portion is scanty. The section illustrates inferences drawn from the limited information available.

Briefly, a mass of porphyritic granite trending roughly north and south and apparently dipping at 20 to 35 deg. to the west, overlies a series of sedimentary and metamorphic rocks consisting of sandstones, quartzites and muscovite schists. This apparently sedimentary series could not be mapped in detail for the reasons stated above.

Close to the contact between the two groups of rocks, the granite has been crushed and brecciated and this zone of brecciation forms one of the main loci of ore deposition. The crest line of contours of radioactivity as determined by Geiger-Muller readings, follows this crushed zone fairly closely. The contact is a folded one and the maximum crushing takes place at the crests of small anti-clines. In these places there is a capping of hematite with some manganese dioxide. Traces of torbernite have been found associated with the hematite and manganese.

Other important features are the shear zones shown on the accompanying plan. The two main ones have lengths of 180 and 100 feet respectively and would each average approximately 10ft. in width. They consist of relatively soft material derived from the alteration of the granite. Not all of the minerals present have been determined, but they include manganiferous hematite in considerable quantities, and also some fluorite and torbernite. The torbernite is usually found in traces only, but occasional pieces of ore weighing in the vicinity of one pound can be dug out of the soft material with a geological pick. Some of these pieces would probably assay in the vicinity of 1 to 2% UO_3. Also present are other radioactive minerals which have not yet been definitely identified. It is thought that they may be uranium ochres. These minerals were found in one place only. the total specimens obtained weighing a few pounds. The crest lines of contours of radioactivity follow the shear zones fairly closely and there appears to be little doubt that the latter are important localising features.

In one case, torbernite and uranophane were found in the granite where there was a very low iron content and no manganese was present. The amount of shearing was also very small. In this case, some of the uranium minerals were in cavities in the rock as distinct from its usual occurrence on cracks and planes of parting. The Geiger-Muller readings showed a regional high in this area.

An intensely crushed zone forms a steep hill at the extreme southern end of the area mapped. The brecciation here is more intense than in the case of the zone which has localised the ore, but there is very little hematite associated with it and no uranium minerals were noted. The fragments in in the zone range from fine rock flour to pieces 12" in diameter. A large proportion of the fragments are 6" in diameter. The brecciation seems to have been formed by a fault which bounds the occurrence on its eastern side. Three small deposits of hematite are associated with this fault. They contain fluorite and also traces of torbernite.

Apart from the hematite and manganese dioxide, fluorite and a little quartz were the chief minerals noted as being associated with the uranium minerals.

(iv) Testing.

As a result of the geological and geophysical Surveys, 12 places were selected for possible testing. At points 2 and 3 pits 3 to 4 feet in depth penetrated the ferruginous material and revealed weathered granite. The quantity of torbernite exposed was small. At point 4, a shaft 13½ feet in depth exposed unpromising material and a sample taken at a depth of 12 feet assayed nil.

(v) Conclusion

The deposit appears to be unimportant as a possible source of uranium ore with a grade not less than 0.25% U_3O_8.

Locality "B"

by

C.J. Sullican, Geologist, Mineral Resources Survey.

A major fault has been partly filled at this point by masses of quartz. A fragment of quartz containing hematite was found to be coated with small quantities of torbernite. The fault in this vicinity should be worth further investigation

3. PARALANA HOT SPRINGS GROUP

A. Localities D, E and F

by

E. Broadhurst, Geological Survey of South Australia.

Some showings of torbernite were found in the locality of the Paralana Hot Springs. The showings are peculiar in the absence of any ironstone accompanying them. They are mainly slight showings of torbernite paint along joint planes in areas of crushed granite, although a few crystals in quartz vughs were seen. Altogether the occurrences are unpromising for the formation of ore bodies, since there seem to be no concentrating agents such as ironstone bodies on definite sheared lodes. However, the area is worth prospecting for the chance of an unexpected ore-body occurring.

4. REFERENCES

1944 Mawson Sir D. "The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia".
Trans. Roy. Soc. S.A. 68(2)

1945 Stillwell F.L. and Edwards A.B.
"Uranium Minerals from Mr. Painter".
This bulletin.

1945 Broadhurst E. "Origin of the Mount Painter Uranium Deposits".
This bulletin.

RB-40

111. REPORTS ON INDIVIDUAL URANIUM OCCURRENCES.

1. Mount Painter Group.

A. Nos. 1, 2 and 3 Workings, Radium Ridge.

by

C. J. SULLIVAN

Geologist, Mineral Resources Survey.

(Plan 3197, Sheet 37, illustrates this report).

Radium Ridge is a very prominent topographic feature extending generally east and west for a distance of approximately 2½ miles. At its closest point it is approximately 1 mile north of the Main Camp. The slopes are steep on the southern side of the ridge and flatten out on the northern side. Some of the steepest sections of the ridge are due to the resistance to weathering of a silicified zone, somewhat similar in appearance to the "Jaspers" of Western Australia. Nos. 1, 2 and 3 workings are located on this ridge.

(i) No. 1 Working.

It consists of a costean 15 feet long and 3 feet deep. It is situated on an outcrop of hematite 45 ft. long and 15 ft. wide. The hematite is of a very dense nature and is intergrown with quartz. It contains small quantities of Fergusonite and traces of yellow uranium minerals. The hematite may have had a different origin from that associated with torbernite in the crush zones. The deposit is thought to be of little economic importance.

(ii) No. 2 Working.

This deposit is very similar to No. 1 and there is a costean on a hematite outcrop. The hematite is very hard and dense and is intergrown with quartz. Fergusonite and monazite are present in places in considerable quantities (Stillwell and Edwards 1945).

(iii) No. 3 Working.

This deposit exposed by this working resembles the usual type of torbernite occurrence associated with hematite and limonite in a crush zone. The hematite and limonite extend along the ridge for a distance of 200 feet and have an average width on the surface of 30 feet. They appear to have resulted from the replacement of a massive pink felspar rock - possibly a pegmatite or porphyry. The latter is intrusive into a paler, finer grained granitic rock. The ironstone is in the form of a capping.

At one place, torbernite occurs over a length of 5 to 10 feet. A small open cut has been made into this occurrence and a shaft has been sunk from the end of the open cut. At the time of the present examination, the

APPENDIX I.

Correlation of Nomenclature of Uranium Occurrences.

Locality (Mawson #)	Locality (this Bulletin)	Remarks
No. 1 Workings	No. 1 Workings	
" 2 "	" 2 "	
" 3 "	" 3 "	
" 4 "	" 4 "	
" 5 "	" 5 "	
" 6 "	" 6 "	
" 7 "	" 7 "	
" 8 "	" 8 "	
" 9 "	Locality G.	Shown on regional map.
" 10		Not found during this investigation.
" 11 "	1E and 2E	
" 12 "	No. 3 East Painter	
" 13 "	4E	
" 14 "	-	This deposit cannot be definitely identified, although it was probably found during prospecting operations.
" 15 "	"Smiler Greenwood"	
" 16 "	-	This deposit cannot be accurately identified.
-	"Bentley Greenwood"	
-	Localities D, E & F.	Paralana.
-	Locality B.	
-	Locality J.	
-	No. 1 East Painter.	
-	No. 2 East Painter.	
-	No. 5 East Painter.	

"The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia", by Sir Douglas Mawson, Trans. Royal. Soc. S.A. 68 (2) 1944.

APPENDIX II.

SAMPLING.

The sampling of the deposits presented the usual difficulties connected with small and irregular shoots. In the original sampling of the No.6 Workings samples were taken at every 10 feet in the drives and at 5 feet lengths in the cross-cuts (or in 10 feet lengths in special circumstances such as the East Adit). The samples were channel samples taken with a hammer and moil. Samples accompanying the development were taken at smaller intervals, usually 2 or 3 feet, and were generally smaller samples taken with a pick.

After some experience of the nature of occurrence of the torbernite it seemed that the face or back samples would not give accurate results, owing principally to the friable nature of the mineral. The torbernite generally occurs in thin seams along the joint planes, and owing to its friable nature it is easily removed from these planes. Thus much of the valuable mineral is removed during the ordinary mining operations at the face so that, especially for a chip sample, the face is not representative of the ore. The chip sample was collected on a sampling cloth spread in front of the face, and during the cutting of the sample the valuable mineral from the thin veins was liable to slide down the face behind the cloth and so be lost. These two factors would tend to make the proportion of valuable mineral in the sample too small. For this reason sampling from the broken dirt at the face or from the trucks was instituted but the objection to this method is that by taking handfuls of broken dirt, the sampler was inclined to take too large a proportion of the fines and so of the valuable mineral, thus making the uranium contents too high.

The two methods of sampling, bulk and chip, came under comparison in the No.5 Workings at East Painter, which was sampled by both methods. In the shaft the bulk sample agreed with the chip samples taken on the sides, but in the drive there were pronounced differences. The bulk samples gave a weighted average assay value of 0.77% U_3O_8 for the level, while the chip samples gave a weighted average of 0.38% U_3O_8, being half of the former. Although some difference had been expected, a 2:1 ratio was much larger than expected. At the northern end of the level the bulk samples agreed with the chip samples, but near the shaft and in the south drive large discrepancies occurred, the bulk sample always having a larger uranium content than the chip. As a check, channel samples from the back and floor of the drive were taken along the first nine feet of the northern part of the drive. These gave even lower results than the chip samples, although the difference between the two was small. The chip-sampling results were generally accepted and used for assessment purposes, but the ultimate test can only be applied with mining operations. All the assays of the various samples, chip, bulk and check, are given in the lists of assays in this report.

APPENDIX III.

LIST OF SPECIMENS FORWARDED TO ENGLAND

illustrating

GEOLOGICAL REPORTS.

Identification by Mr. E. Broadhurst.

Specimen No.	Locality and Identification.
1	Mt. Painter No.6 Workings manganiferous ironstone from dump at 50' level.
2	Mt. Painter No.6 Workings Hematite breccia (Kaolinitic) from east branch of north crosscut 100' level.
3	Mt. Painter No.6 Workings Hematite breccia (Kaolinitic) from East branch of north crosscut 100' level.
4	Mt. Painter No.6 Workings silicified gneiss from fragments of breccia in South portal.
5	Mt. Painter No.6 Workings silicified gneiss (with torbernite seam) from drive South of open cut.
6	Mt. Painter No.6 Workings Hematite breccia from end of North drive 50' level.
7	Mt. Painter No.6 Workings banded gneiss from 185' level.

Identification by Mr. R.C. Sprigg.

8	Aplitic Granite Mt. Painter road near sunshine Pound.
9	Porphyritic Granite on Mt. Painter road near Sunshine Pound.
10	Dolerite Volcanic Neck 1 mile East of Sprigg Nob.
11	Torbernite No.5 Workings East Painter.
12	Metamorphosed Amygdaloidal? Basalt near Arkaroola Well.
13	Acid Biotite gneiss near Mt. Painter main camp.
14	Sericitic Quartzite Mt. Pitt.
15	Sphene from Amphibolite sphene lode Mt. Painter road.

Specimen No.	Locality and Identification
<u>Identification by Dr. F.L. Stillwell.</u>	
16	No.6 Workings torbernite vein in mica schist.
17	Mt. Painter No.6 Workings yellow autunite and green torbernite.
18	Torbernite No.5 lobe East Painter.
19	"Nail hole" quartz from Radium Ridge between No.2 Workings and Mt. Gee.
20	Smiler Greenwood's prospect East Painter torbernite and fluorite disseminated through ironstone.
21	No.5 Workings East Painter with limonite from open cut. Torbernite plates tend to form a coarse boxwork.
22	Ore relatively "rich" in Fergusonite (marked with red pencil) and consisting essentially of hematite and monazite No.2 Workings Radium Ridge nearest equivalent to primary ore.
23	No.3 Workings. Radium Ridge Metatorbernite.
24	Smiler Greenwood's prospect East Painter with pseudomorphs after pyrite.
25	Uranophane (yellow needles in vughs) No.2 Workings Radium Ridge.
26	Small open cut at No.6 Workings Mt. Painter camp. Seam of weathered torbernite with uranophane (as described under weathering of torbernite).
27	No.2 Workings Radium Ridge hematite and epithermal quartz.
28	B. Greenwood's prospect East Painter white boxwork developing as a replacement of torbernite boxwork.
29	B. Greenwood's prospect East Painter boxwork of torbernite plates in small pits.
30	B. Greenwood's prospect East Painter white boxwork residual after original torbernite boxwork.
31	Torbernite goethite and psilomelane in fractures of a breccia No.7 Workings Mount Painter camp.
32	No.7 Workings Mt. Painter camp torbernite and metatorbernite.

Specimen No.	Locality and Identification
33	"Nail Hole" epithermal quartz (cows teats) from Radium Ridge between No. 2 Workings and Mt. Gee.
34	"Nail Hole" epithermal quartz from Radium Ridge between No. 2 Workings and Mt. Gee.
35	Torbernite on joint plans South portal at 100' level No. 6 Workings.
36	No. 5 Prospect Mt. Painter Camp. Pits with lepidocrioite lining them former from leaching of pyrite, still present in places.
37	"Nail Hole" quartz with gypsum filling in the nail holes from rear No. 5 Workings Mt. Painter Camp area.
38	Mt. Painter torbernite from No. 7 Workings. This torbernite fluoresces presumably indicating a film of copper-free uranium salts on the surface of the torbernite.
39	No. 6 Workings torbernite and autunite (in part cupriferous) in vughs in psilomelane - original shoot.
40	No. 5 prospect East Painter.
41	"Gummite" (yellow) in vugh, No. 2 Workings Radium Ridge. Similar to analysed material massive hematite ore, characteristic of this deposit with intergrown monazite (brown).
42	Mt. Painter district Radium Ridge No. 2 Workings contains fergusonite and monazite.

IV. - A POSSIBLE EXPLANATION OF THE GENESIS OF THE MOUNT PAINTER URANIUM DEPOSITS LARGELY BASED ON FIELD OBSERVATIONS

by

C. J. SULLIVAN
Geologist, Mineral Resources Survey.

---

1. INTRODUCTION.

This report describes the field evidence in support of a supergene origin for the torbernite deposits in the Mount Painter district. This origin is also emphasised by Stillwell and Edwards(1). The report was originally written in response to a preliminary report by Stillwell(2) in which a hydrothermal genesis was favoured.

2. BRIEF DESCRIPTION OF THE DEPOSITS.

Except for the autunite in the shoot mined in the No.6 workings, the most important uranium mineral so far found at Mount Painter is torbernite - hydrated copper uranium phosphate. The torbernite occurs as stains or films coating cracks and partings in the rock. The majority of the deposits so far discovered are below 0.1 per cent U_3O_8 in grade. At the locality F (see geological map, Plans 3288-3291 inclusive), southwest of Paralana Hot Springs, these stains give a strong impression of having been formed by weathering of the granite boulder on which they occur. At the No.6 deposit, Mount Painter, however, a rich shoot of autunite and torbernite, which is thought to have contained about three (3) tons of uranium trioxide, was mined. At the No.5 deposit, East Painter, development has indicated about 320 tons of ore, possibly containing 0.38 per cent U_3O_8. Apart from these occurrences, no ore of mineable grade has been found.

3. IRONSTONE(3).

Most of the torbernite deposits are associated with ironstone and sometimes with minor amounts of fluorite, barytes and quartz. Most of the ironstone concentrations occur in brecciated zones. At the No.3 and No.4 deposits at Mount Painter, the "Bentley Greenwood" and probably the "Smiler Greenwood" deposits at East Painter, the ironstone forms a surface capping from 3 to 10 feet thick. Underneath this is weathered granitic rock, usually containing spangles of torbernite sparsely disseminated in cracks and partings. A

---

(1) F.L. Stillwell and A.B. Edwards: Uranium Minerals from Mount Painter.

(2) F.L. Stillwell: Uranium Minerals from Mount Painter. Minerographic investigations of the Council for Scientific and Industrial Research. Report No.315.

(3) The term "ironstone" is used in the sense of an aggregate of ferruginous minerals consisting chiefly of hematite with in some cases, limonite, psilomelane, magnetite and subordinate amounts of other minerals.

crosscut driven under No.3 deposit, Mount Painter, revealed weathered red granite with traces of torbernite, whilst under the ironstone capping of the No.4 deposit, Mount Painter, a weathered gneiss with traces of torbernite was encountered. In the case of the "Bentley Greenwood" a solid red felspathic rock underlies the mantle of ironstone. The occurrence of the ironstone at the surface appears to be too marked to be an accident of erosion. In each of the cases cited, the ironstone extends over hundreds of square feet at the surface. If these occurrences represented lodes formed by hydrothermal processes, one would not expect them to be so often localised along flat planes, which would enable them, on being exposed by erosion, to parallel the present surface.

In many cases the major structural planes present do not appear to have controlled the deposition of ironstone in the way one would normally expect, if it was of deep-seated origin. Thus, at the "Smiler" deposit, the major shears strike at about 340 degrees and dip at 60 to 70 degrees in an easterly direction. Test pits on these shears have, however, revealed that the manganiferous ironstone is largely confined to the first 3 to 5 feet from the surface, and that below this weathered granite predominates. There are no obvious structural planes which could have been expected to localise mineralization parallel to the present surface. In many parts of the Mount Painter area stains and thin coatings of ironstone may be seen on the surface of granitic rocks.

In all cases where the ironstone forms a mantle only, the degree of brecciation has been relatively slight. At the No.6 deposit at Mount Painter, hematite extends to depths exceeding 100 feet but there is a concentration of hematite at the surface and the proportion of hematite decreases with increasing depth. The main crush zone at East Painter has been tested to a depth of only 50 feet, and hematite persists to that depth. In both these localities the degree of brecciation has been relatively great, thus allowing the deep penetration of iron-bearing solutions from above. However, some of the hematite of No.6 deposit appears to be of hydrothermal origin.

It is thus concluded from the field evidence, that a large proportion of the ironstone was deposited near the land surface from meteoric waters(1). The striking scarcity of hydrothermal gangue minerals such as quartz or carbonates, also supports this mode of origin.

---

(1) There is evidence that there are at least two types of ironstone at Mount Painter.
The above discussion deals with the ironstone chiefly associated with the occurrences of secondary uranium minerals. In most cases this is manganiferous, contains limonite and is rarely bladed. It shows a close association with the present land surface. The second type mainly consists of bladed hematite and is closely associated with the Mount Gee quartz mineralisation. It occurs on the slopes of Mount Gee, on Radium Ridge, on spurs leading from Mount Painter and in many other places. This type of hematite shows little tendency to be concentrated at the surface and is usually associated with large quantities of quartz. The No.2 deposit on Radium Ridge may represent a third type. Here a mass of hematite (after magnetite) is associated with monazite and fergusonite. It is reported by Kleeman that the fergusonite has a high lead content. This appears to connect the deposit with the granites in which it may be in the nature of a segregation.

4. ASSOCIATION OF THE SECONDARY URANIUM MINERALS WITH IRONSTONE.

The ironstone and the secondary uranium minerals (torbernite, etc.) appear to have a close genetic relationship, as they are usually found closely associated in the field. As in the case of ironstone the greater concentrations of torbernite found so far appear to be confined to shallow depths below the surface. For example the shoot in the No.6 deposit commenced approximately 25 feet from the surface and petered out at a depth of 50 feet. Extensive exploration to a depth of 100 feet revealed material carrying only traces of torbernite. In the No.7 and No.8 workings at Mount Painter, impoverishment begins at about 30 feet below the surface. Radiation surveys by Messrs. Thyer and Dooley(1) indicated that most of the ironstone masses have a higher degree of radio-activity than the surrounding rocks. Many specimens of ironstone exhibit radio-activity, but chemical assay proves that they contain no uranium.

5. LEACHING OF URANIUM.

The observed radio-activity could be due to the disintegration products of uranium left behind in the ironstone after the uranium had been leached out(2). In this connection the relative solubilities of radium and uranium should be noted. Nearly all common salts of radium are practically insoluble, e.g. $RaSO_4$ has a solubility of .000002 grams per 100 c.c. of water at 25°C, while common uranium salts are relatively soluble in water, e.g. UCl_3 is very soluble, $UO_2SO_4.3H_2O$ has a solubility of 20.5 grams per 100 c.c. at 15°C and $UO_2SO_4.7H_2O$ is very soluble(3). Thus, when a uranium mineral containing radium is exposed to dissolving waters, radium tends to remain in situ, while uranium salts go into solution.

6. AGE OF THE TORBERNITE.

On the basis of mineralogical studies, Stillwell(4) suggested the possibility that the formation of the torbernite was contemporaneous with the Mount Gee quartz mineralisation. Stillwell and Edwards(5) give it as an alternative explanation to the supergene origin. The radium/uranium ratio of torbernite and autunite from the No.6 deposit, Mount Painter has been estimated by R.G. Thomas(6) from the results of a number of analyses and tests at the experimental treatment plant and refinery of Australian Radium Corporation N.L. which worked the deposits at Mount Painter between 1926 and 1934, at 2.7×10^7 against an equilibrium ratio of 3.4×10^7. Uranium-radium equilibrium is reached in approximately 10^6 years(7), so that the above results

---

(1) R.F. Thyer and J.C. Dooley, elsewhere in this bulletin.

(2) R.F. Thyer and J.C. Dooley, elsewhere in this bulletin.

(3) Handbook of Chemistry and Physics 28th Ed.,Chem. Rubber Publishing Co., Cleveland, Ohio. pp.442-3, 482-3.

(4) F.L. Stillwell. Report No.315. C.S.I.R. etc.

(5) F.L. Stillwell and A.B. Edwards. Uranium Minerals from Mount Painter. Elsewhere in this bulletin.

(6) R.G. Thomas, "The Processing of Radium Ores in South Australia". Aust. Chem. Inst. Jour. and Proc. Vol.IX, 6th June, 1942.

(7) Danas System of Mineralogy, Centennial Anniversary Issue, 1884 - 1944, Vol.I, John Wiley and Sons, New York. 1944. p.617.

strongly suggest that the torbernite is less than 10^6 years in age. This is supported by the low lead content of the torbernite(1). Later analyses by Mr. Dalwood Analyst and chemist of the South Australian Mines Department have proved that the lead content of a sample of autunite and torbernite from No.6 Deposit, was less than 0.0001 per cent. but have revealed a higher lead content (0.045 per cent.) corresponding to an age of more than 7 million years, for a sample of torbernite from No.5 Deposit, East Painter. This result is very interesting and will be referred to later.

The youngest magmatic activity known in the district is not later than Palaeozoic. It has been suggested that the Paralana Hot Spring indicates magmatic activity but this seems most unlikely as the spring is situated on a fault near the margin of the Great Artesian Basin, and the temperature and chemical composition of the water is comparable to that issuing from adjacent artesian bores. Investigations by the South Australian Mines Department of the Mesozoic beds to the eastward and westward of the Mount Painter field reveal that they have not been intruded by igneous rocks or been subjected to mineralisation(2). There is no quartz mineralisation along the Tertiary fault bounding the eastern side of the range. The quartz of the Mount Painter field is confined to the late Proterozoic and early Palaeozoic rocks and appears most likely to be genetically connected with the old granitic intrusives of that field. The above evidence indicates that the torbernite is of Cainezoic age. It is therefore doubtful if it could have been deposited with the Mount Gee quartz, which is presumably pre-Mesozoic in age.

7. TEMPERATURE OF FORMATION OF TORBERNITE.

As pointed out by Stillwell(3), the torbernite must have been deposited below 60°C, as above this temperature it loses water and forms meta-torbernite. This very low temperature of deposition suggests a supergene origin though it does not preclude an epithermal origin as stated by Stillwell.

8. ASSOCIATION OF THE URANIUM DEPOSITS WITH GRANITE.

The uranium deposits are confined to the area shown as granite on the geological map (see Plans 3288-3291 inclusive). None has been found in the adjoining areas of folded and faulted sedimentary rocks which have been intruded by the granite. There is evidence to suggest that in places, the granite contains primary radio-active minerals. Thus Mawson(4) states that - "the red aplitic granite from certain localities, for instance from No.16 contains particles of hematite and other black specks, some of which were found by autoradiographic test to be radioactive." Both radioactive ilmenite(5) and a mineral similar to fergusonite(6) have been reported from the field. At a point

---

(1) F.L. Stillwell and A.B. Edwards, in this bulletin.

(2) R.C. Sprigg, Verbal communication.

(3) F.L. Stillwell. Report No.315. C.S.I.R.

(4) Douglas Mawson. The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia. Trans. Roy. Soc. Sth. Aust. Vol.68, Pt.2 1944, p.355.

(5) Douglas Mawson, Op.Cit. p.356.

(6) F.L. Stillwell and A.B. Edwards, in this bulletin.

55 chains northeast of the East Painter camp, a specimen of granite was found to be radio-active though it contained no torbernite or cavities after torbernite. At locality D west of Paralana Hot Spring, torbernite was found in small cavities in uncrushed coarse granite and it is believed that the torbernite was most likely derived from the oxidation of a primary mineral. At locality E it was found in a crushed granite, apparently unaccompanied by hematite or by any hydrothermal minerals. Much useful work could be done by collecting radioactive granites and identifying the minerals showing this property.

9. TORBERNITE DEPOSITS IN OTHER PARTS OF THE WORLD.

Torbernite-autunite occurrences in other parts of the world are attributed to the action of weathering agents on primary minerals such as uraninite, (including pitchblende), euxenite, samarskite and betafite. In Madagascar, it was found that waters draining an area intruded by pegmatites containing betafite, samarskite, euxenite and other uraniferous titano-columbo-tantalates, deposited their uranium content as autunite in peaty clay beds resting on the ancient granite(1). In Portugal, primary uranium minerals in quartz and pegmatite veins, were found to weather into torbernite(2).

At the important deposit of Shinkolobwe, Belgian Congo, more than half of the uranium in the oxidised zone was in the form of torbernite. In parts of the deposit, the torbernite occurred at considerable distances from the primary lode containing pitchblende together with iron, copper and nickel sulphides(3).

10. SUGGESTED HISTORY OF THE FORMATION OF THE DEPOSITS.

The above evidence suggests that the history of the formation of the uranium deposits at Mount Painter may have been somewhat as follows:-

The sedimentary rocks of the Pre-Cambrian series were intruded by granite in late Pre-Cambrian or early Palaeozoic time. This granite contained disseminated uranium-bearing minerals such as uraninite and minerals of the pyrochlore-fergusonite-samarskite type, which were most concentrated in the highly felspathic and pegmatitic fractions. Fracturing and brecciation of the rocks of the area were followed by the deposition of quartz (and possibly some primary hematite) with some magnetite, pyrite, fluorspar, barytes and perhaps uraninite from hydrothermal solutions. The centre of the quartz mineralisation was Mount Gee, but it also impregnated though to a lesser degree, the crush zones which now contain the uranium deposits. This primary mineralisation was probably associated with the granite referred to above.

---

(1) Ore Deposits of the Western United States. Lindren Volume. New York 1923, p.476.

(2) Alteration des Minerais Noirs D'Uranium Portugais. Chimie et Industrie. 30:805 June, 1933.

(3) Minerals Year Book, 1934, United States, Department of the Interior, Bureau of Mines. p.498-502.

The history of the area in the Palaeozoic era is not well known. It is possible that Mesozoic sediments were deposited over the area and have since been removed by erosion. In any case, the area, in common with the remainder of Australia, was probably reduced to a peneplain by early or middle Tertiary time. Weathering of the crush zones began as soon as any overlying Mesozoic sediments had been removed, and continued during the remainder of the peneplanation. In Kosciusko(1) (Late Pliocene) time, considerable earth movements began and resulted in the uplift of the Mount Painter block; erosion was intensified and the dissection of the area began. The crush zones formed suitable channels for the circulation of ground waters and it is suggested that these waters have thoroughly leached and kaolinised the granite in these zones; they have taken iron, manganese and uranium into solution and concentrated them in the form of the present manganiferous ironstone-uranium deposits, mainly as cappings over the crush zones.

The leaching of the uranium content of the ironstone cappings is proceeding at the present time. The observed radioactivity of much of the ironstone which does not now contain uranium and the dissemination of torbernite in sheared granite immediately below some of the ironstone masses point to this. Leached cavities after torbernite, may be seen at the surface in several deposits. The torbernite at No.6 Deposit has apparently been deposited during late Cainozoic times.

The origin and history of the individual deposits may be interpreted as follows:-

(I) The Torbernite at a point 2½ miles southwest of Paralana Hot Spring is definitely associated with pegmatitic granite. There is no sign of hydrothermal activity. There appears to be little doubt that the primary uranium-bearing minerals from which the torbernite was derived by weathering, occurs sparsely disseminated in the pegmatitic granite.

(II) At a point 110 chains southwest of Paralana Hot Spring the pegmatitic granite is somewhat crushed and sheared, but is not greatly altered. A few flakes of torbernite and thin coatings of ironstone occur along shearing planes. This may represent the early stages of the development of a torbernite deposit by ground water leaching of granite. The No.8 and parts of the No.7 deposits at Mount Painter, where little ironstone is present, may also belong to this stage.

(III) Deposits such as the "Smiler" and "Bentley" at East Painter and the No.3 and No.4 deposits at Mount Painter, in which surface cappings of ironstone are underlain by more or less sheared granite containing a few flakes of torbernite, may represent the next stage in the development of torbernite deposits by the leaching of granite. If so, no primary ore of minable grade could normally be expected. However, hydrothermal minerals such as barytes, fluorite, some pyrite and a little quartz have been noted in this type of deposit and it is not impossible that a primary uranium mineral was introduced with these minerals. The amount of secondary uranium mineralisation so far found in these deposits is, however, small and could have been derived from primary minerals in the granite and pegmatite.

(IV) The nature of the primary mineralisation from wheih were derived the torbernite and autunite deposit at No.6 working,

---

(1) W.R. Browne. An Attempted Post-Tertiary Chronology for Australia. Proc. Linn. Soc. N.S.W., Vol.LXX, Parts 1-2, 1945.

Mount Painter and the torbernite deposits in the East Painter crush zone, is not clear. As in the above cases, there is a concentration of ironstone at the surface and there has been widespread leaching of the granitic rocks in the crush zones. The degree of circulation of ground water in the brecciated granitic rocks appears to have been sufficient to have leached out their uranium content and re-deposited it in the form of the present torbernite and autunite shoots. Drilling to a depth of approximately 245 feet in the No.6 deposit revealed kaolinised granite with occasional veinlets of hematite.

However, there is considerable evidence of hydrothermal activity in the East Painter crush zones. Thus, in the vicinity of Nos.1 and 2 deposits, East Painter numerous cavities and pseudomonphs after pyrite sometimes filled with torbernite have been noted(1). A prominent ironstone outcrop, containing traces of torbernite, which occurs to the north of No.5 prospect, East Painter, consists mainly of hematite, but contains numerous boxworks of limonite which may represent former sulphide crystals. It is evident that the crush zones have been deeply and thoroughly leached and many primary minerals which might have been present would have been removed in solution or very much altered. The No.5 deposit at East Painter, possesses particularly interesting features. A rich torbernite seam is largely confined to a narrow channel which on the footwall side, is seamed with quartz. As stated previously, torbernite from this deposit shows a considerably higher lead content than that from No.6. Assuming no admixture of lead from other sources the lead-uranium ratio of the ore indicates an age of approximately 7 million years. This torbernite could have been produced by the oxidation of a uranium-bearing vein, (the primary mineral possibly being uraninite) more or less in situ. This possible origin is in contrast with the leaching of uranium from large bodies of granite and the formation of torbernite even up to the present time. It seems likely at least, that the torbernite has not been in solution in the past 7 million years.

11. <u>CONCLUSIONS</u>.

The above evidence strongly suggests that the torbernite deposits are of supergene origin. It is believed that in many instances, their uranium content was derived from primary minerals sparsely disseminated in the granite and pegmatite and that in these cases, there is little hope of finding workable deposits of primary ore(2).

There is evidence of hydrothermal mineralisation in the East Painter crush zone and one small torbernite shoot (No.5) and two rather promising torbernite-bearing outcrops (Nos.1 and 2) have been found in this section. Further testing in the East Painter area, particularly below the zone of weathering will indicate more precisely the origin of these deposits.

<u>Canberra</u>. 30/10/45.

---

(1) F.L. Stillwell and A.B. Edwards:- Uranium Minerals from Mount Painter ...... This bulletin.

(2) The writer is referring to ore averaging 0.25 to 0.5 per cent. U_3O_8. It is possible that future scientific developments will render much lower grade ore attractive.

IV - ORIGIN OF THE MOUNT PAINTER URANIUM DEPOSITS

by

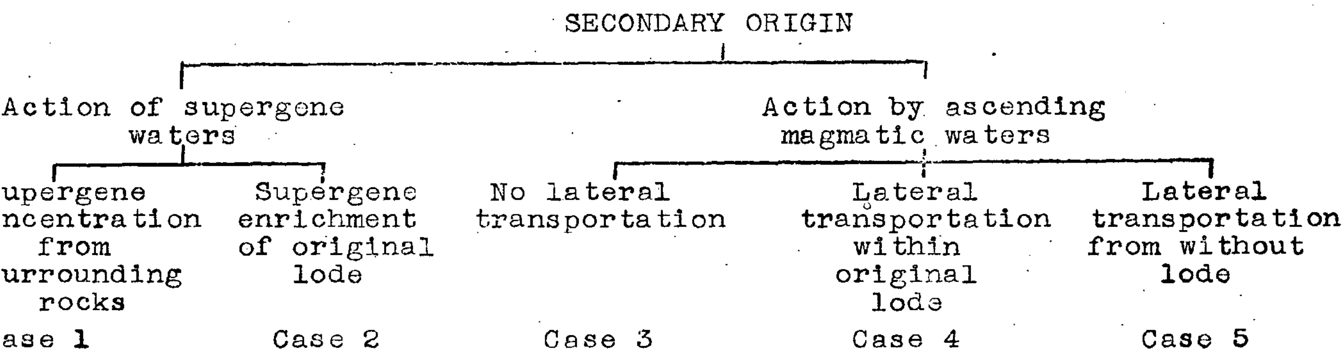
E. BROADHURST

Geological Survey of South Australia

1. Introduction

The present discussion is meant not so much to present an original view on the origin of the deposits, as to collect the possibilities suggested by Stillwell and Edwards and by Sullivan. There is a general agreement that the uranium minerals other than fergusonite are of secondary origin. This means that the uranium has been derived from primary minerals, but there are various possibilities for the mode of origin of the secondary deposits. These possibilities are summarised in the following table.

POSSIBLE ORIGINS OF MT. PAINTER URANIUM DEPOSITS



The two main divisions are based on whether the formation of the secondary minerals from the primary uranium minerals was carried out by supergene waters or by the magmatic epithermal waters connected with the formation of the Mt. Gee quartz and hematite. The further sub-divisions depend on the amount of transportation of the uranium-bearing solutions before re-deposition occurred. These various cases will now be discussed.

2. Case 1

This process is the concentration of minute quantities of uranium in the surrounding rocks into lodes by the action of supergene waters. This case has been fully presented by Sullivan, and there is no need to repeat them here.

3. Case 2

This is the ordinary process of supergene enrichment of a primary lode. It differs from Case 1 mainly in the amount of re-concentration which has occurred, and in the fact that Case 2 admits of the possibility of a primary lode below the zone of oxidation. Facts favouring this mode of origin in the No.6 Deposit are the kaolinisation of the hematite breccia, showing the passage of acidic waters through the rock, and the tendency for torbernite to occur at the edges of the breccia. The latter fact could be explained as being caused by the uranium-bearing solutions precipitating their contents when passing out from the body of the breccia to the surrounding rocks. The other arguments favouring this process have been given by Sullivan viz. the low Pb/U ratio and the non-equilibrium condition of the U/Ra. ratio of the autunite and torbernite indicating a comparatively recent age, the absence of any known signs of igneous activity later than the Palaeozoic, and

/ ...

the radioactivity of the outcrops of hematite breccia indicating that the uranium has been leached out and the radium left behind.

4. Case 3

This process is one of chemical attack by the epithermal 'Mt. Gee' solutions on the primary uranium minerals, converting them mainly to the hydrous phosphates. In this case the change has occurred in situ and the secondary minerals occur in place of the primary minerals. At East Painter, the boxwork structures which occupy the cavities left by an original mineral suggest that such a process might have occurred and that the torbernite now occupies the cavities left by a primary uranium mineral.

If the development of hematite from magnetite occurred with the introduction of the epithermal Mt. Gee quartz, then it is likely that even if the shape and uranium content of the primary lode were preserved, the primary minerals would be converted to secondary minerals by the action of these solutions. In such a case the minerals might remain in situ, or become transported and so belong to either of Cases 4 and 5.

5. Case 4

Either the fourth or fifth processes, that of the attack of the epithermal Mt. Gee solutions on the primary uranium minerals with the resulting formation of the hydrous uranium phosphates, seems from evidence cited by Stillwell and Edwards to have definitely occurred.

In Case 4 the uranium is considered to have been taken into solution and transported, but still remaining within the lode. At certain favourable structures deposition of the uranium minerals occurred. Applying this case to No. 6 Deposit, the hematite breccia would represent the original uranium lode. The only favourable position for deposition known so far is the much faulted vicinity of the "V" shaped mass of silicified gneiss near the main shear, as described in the section dealing with this deposit in detail.

6. Case 5

The process in this case is one of solution of the uranium from the primary minerals by the epithermal solutions, transportation into favourable structures such as shears and brecciated zones, and re-deposition there. This process corresponds in a general way to Case 1, except that magmatic waters act as the media of solution and transportation instead of meteoric waters. The essential difference, however, is that the process in Case 1 is confined to the zone of oxidation, probably close to the surface, while the process in Case 5 can occur at any depth in which the epithermal solutions were active.

7. Conclusions

Thus there are five different processes of concentration which could have been active to form the uranium deposits, and each has certain facts to support it. In ma ny places it is likely that a combination of the processes has occurred. The important feature is that each process throws a different aspect on the possibility of finding further ore-bodies.

In considering the future prospects of the deposits, the possibilities which each process holds out for the discovery of further ore are the essential features. The ore-bodies formed in Case 3 are identical in shape and uranium content to a primary lode, the only difference being the nature of the uranium-bearing minerals. All ore-bodies formed in Cases 3, 4 and 5 are liable to supergene alteration, i.e. Case 2.

/ ...

In Cases 1 and 2 the chances of finding another ore-body such as No.6 depend on the possibility that this deposit is a 'perched' body of surface enriched ore. This means that this ore-body occurred at a much smaller depth than the normal zone of supergene concentration. Owing to some adventitious geological occurrence such as faulting or the lack of sufficient acid solutions for re-solution of the minerals the deposit was left behind in the downward migration of the water-level accompanying erosion. Under such circumstances there might be further ore-bodies deeper in the zone of oxidation. However, such an occurrence is merely a possibility, since oxidised minerals such as autunite and torbernite could occur anywhere in the zone of oxidation and probably the No.6 ore-body, if it was formed by supergene processes, was at its normal depth from the surface. In the other deposits which have not been prospected as thoroughly as No.6, such as the Greenwood's Camp deposits (Nos. 1, 2, and 5 East Painter), there is quite a possibility that the processes in cases 1 and 2 have acted and there will be a zone of supergene enrichment. Case 2 will differ from Case 1 in that there will be a primary lode (or its equivalent through the action of Case 3) lying underneath the enriched body. Even if there is such a lode present, its uranium content may be very small. Another, and very optimistic view of the Greenwood's Camp deposits is that they are the equivalent of primary lodes, as in Case 3, which have become leached at the surface. Such an explanation of these deposits would assume ore reserves many times greater than those already produces from the field.

Cases 4 and 5, though differing largely in the mode of origin of the deposits, do not differ so much in the resulting occurrences of the ore. In Case 4, the breccias and shears represent the original lodes in which the secondary uranium minerals have become re-concentrated. In Case 5, they represent the means of access to the uranium-bearing solutions and in which the uranium minerals would be deposited where the structures were favourable. Hence in both cases the breccias and shears would contain the orebodies, but if the No.6 Deposit is any criterion they would be very small and irregular, and liable to occur at any depth.

From the above reasoning, it can be seen that there are more possibilities of further orebodies at the Greenwood's Camp deposits, and perhaps the other East Painter deposits, than at No.6 Workings. In the latter locality the possibility of further orebodies depends on adventitious geological circumstances, whereas at Greenwood's Camp there are definite possibilities that supergene enriched orebodies and even the equivalent of payable primary ores exist. Hence it is recommended that if any further development work is carried out, it should be done on the Greenwood's Camp Deposit. (Nos. 1, 2 and 5, East Painter).

PART IV - REPORT ON GEOPHYSICAL SURVEYS OF URANIUM DEPOSITS, MOUNT PAINTER DISTRICT, SOUTH AUSTRALIA.

by

R.F. Thyer and J.C. Dooley, Geophysicists,
Mineral Resources Survey.

(Plans Nos. 1169, 1170, 1180, 1220,)
1222, 1232, 1237, 1307 to 1311 inclusive.

I. INTRODUCTION.

The geophysical surveys reviewed in this report were part of an investigation carried out by the Commonwealth Government on behalf of the British Government. The responsible Commonwealth authority was the Controller of Minerals Production, Department of Supply and Shipping and operations on the field were under the supervision of S.B. Dickinson, Director of Mines, South Australia and Delegate in that State for the Controller. Deposits of uranium minerals occurring at several places in the Northern Flinders Ranges of South Australia, and centred about Mount Painter, were subjected to detailed geological examination followed by surface and underground prospecting by means of costeans, shafts, drives, etc. The geophysical work was undertaken with the object of assisting the geological examination by measuring the radioactivity of rocks in situ and of hand specimens and assisting the prospecting work by making estimates of uranium content of mine samples.

The geophysical surveys were carried out by officers of the Mineral Resources Survey Commonwealth Department of Supply and Shipping using instruments constructed by them, and consisted of two main parts namely -

(i) field surveys to locate and delineate areas of high radioactivity.
(ii) laboratory work to make approximate determinations of uranium content of mine samples by measuring the radiation from them.

In addition to the above, the instruments were used during each visit to the field to identify radioactive minerals in hand specimens. The geologists gathered specimens from outcrops during reconnaissance surveys of outlying parts of the field and brought them to the instrument at the base camps for determination of activity.

The initial work was conducted by R.F. Thyer in August and September, 1944, at the request of S.B. Dickinson. A Geiger-Muller radiation counter was built in the workshops of the Physics Department of the University of Adelaide, and the helpful assistance of Professor Kerr-Grant and members of his staff, especially John Symons, is gratefully acknowledged. This apparatus was used in making field surveys over the No. 6 deposit at Mount Painter and over the 'Smiler Greenwood' and 'Bentley Greenwood' deposits at East Painter, but did not function satisfactorily.

The field work was suspended temporarily while the apparatus was redesigned in the Mineral Resources Survey laboratory at Canberra. At the same time a second apparatus, suitable for carrying out approximate uranium determination of mine samples in a laboratory was constructed.

Field work was resumed and continued during November and December, 1944, by R.F. Thyer and J.C. Dooley and radiation surveys were made in detail over deposits near Greenwood's Camp at East Painter. The new field instrument proved to be satisfactory, although somewhat slow in operation.

Upon returning to Canberra after the abovementioned field work was completed J.C. Dooley built a unit of light weight fitted with headphones and suitable for reconnaissance work in outlying prospects difficult of access and for which qualitative, rather than quantitative, results would be of value to the geologists. At the same time an investigation was conducted at Canberra by J.M. Rayner into the possibility of using radiation measurements as a means of estimating the uranium content of samples from the Mount Painter field.

The Director of Mines, South Australia wrote in January, 1945 to the Director, Mineral Resources Survey, asking that a Geiger-Muller apparatus be made available at Mount Painter for preliminary assay work, as a large proportion of the mine samples had been found on chemical assay, to contain little if any uranium. During February and March, 1945 J.C. Dooley was engaged on this type of work at Mount Painter and while there he carried out some field work with the portable telephone equipment.

Preliminary reports were prepared on the completion of each phase of the field or assay work and the present report combines and replaces them.

It is proposed to describe briefly the fundamental nature of radioactivity with special reference to such features as have a particular bearing on the problems encountered in the field and assay work. In addition, it will be necessary to deal briefly with the geology of the various areas covered by the field work in so far as it effects the geophysical problem.

II. FUNDAMENTALS OF RADIOACTIVITY.

Radioactive substances may be defined as those which have the power of spontaneously disintegrating, and, in doing so, changing from one atomic form to another. More than forty isotopes of the heavier elements and few isotopes of some of the lighter, e.g. potassium and rubidium, are radioactive. There are, however, two main series starting from the elements uranium (atomic weight 238) and thorium (atomic weight 232) and changing from one atomic form to another to give as end products two isotopic forms of lead. A third series of lesser importance starts from a minor isotope of uranium (atomic weight 235) and is known as the actinium series. The rate at which the change from one atomic form to another takes place is characteristic of each member of a series and each member has a half period, i.e. the time required for half the mass to disintegrate, corresponding to the rate of change.

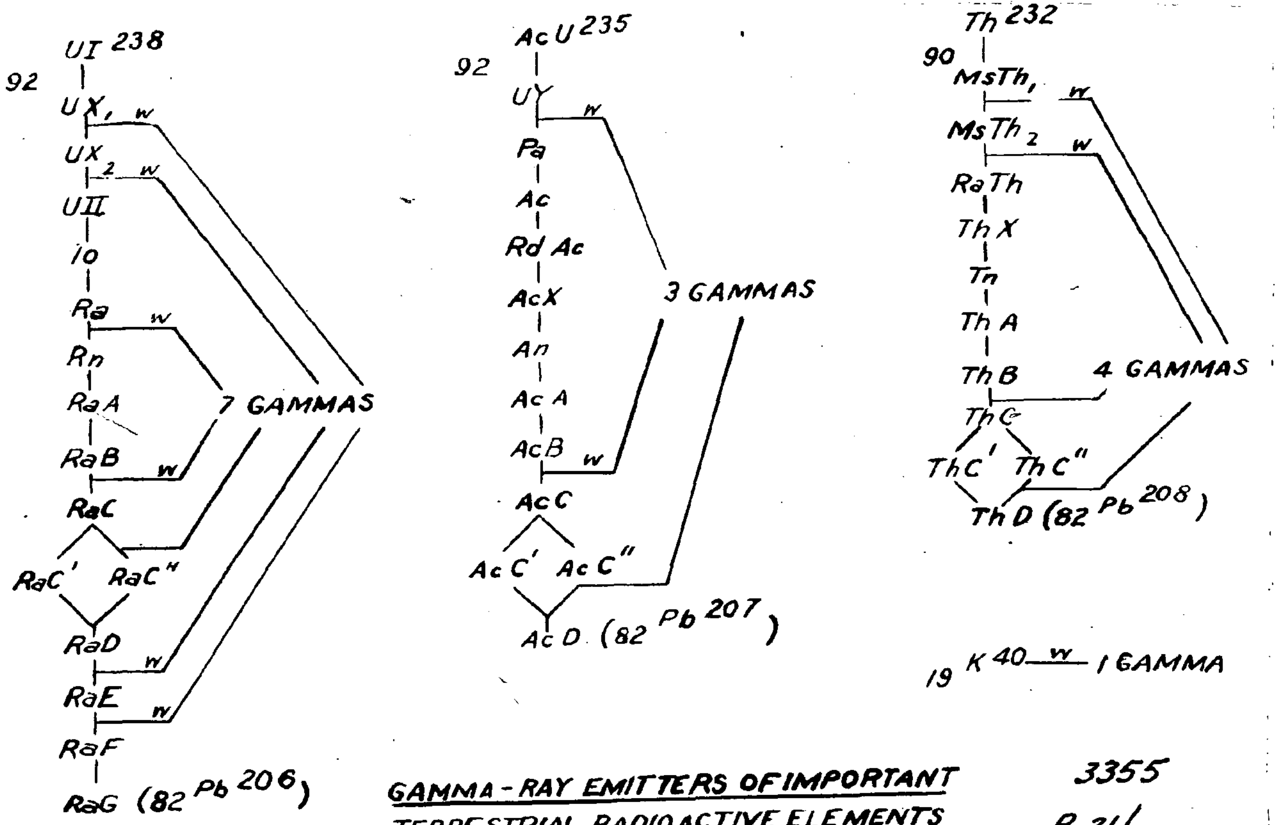
After a sufficient lapse of time, (which, in the case of the uranium series, is about a million years) the relative amounts of each member present become constant providing that the series remains undisturbed. The amounts of each member present are proportional to their respective half periods and the series is said to be in radioactive equilibrium.

The atomic transformations are accompanied by the emission of rays which are of three distinct types, namely alpha, beta and gamma rays, although all three types are not given off at each transformation. Alpha and beta rays are corpuscular in character and are absorbed by relatively small thicknesses of solid matter. Gamma rays, on the other hand, are similar in character to X-Rays, and have a much greater power of penetration. The ratio of the penetrating powers of the three types of rays are nearly 1 to 100 to 100,000 respectively. It is claimed that the gamma rays from certain members of the uranium series can pass through 15 inches of solid lead and still be detected, although they drop to half their value in passing through 1.4 cm. of lead.

In the field and assay work under review the detecting instruments were screened to such an extent that all the alpha and beta rays were excluded; and the gamma rays measured.

Gamma rays may be detected photographically, by electroscopes, ionisation chambers and by radiation ray counters such as the Geiger-Muller Counter. The last three mentioned are those methods most commonly used, and, according to R.B. Rose (1), the Geiger-Muller counter, is more sensitive than either the electroscope or ionisation chamber. In the work under review Geiger-Muller radiation counters were used as detectors.

The following table, shows the transformations which give rise to gamma rays in the uranium, actinium and thorium series and in the radioactive isotope of potassium.



GAMMA-RAY EMITTERS OF IMPORTANT TERRESTRIAL RADIOACTIVE ELEMENTS

3355
R 21/.

III. RADIOACTIVITY IN THE MOUNT PAINTER DISTRICT.

As far as the radiation measurements at Mount Painter are concerned, the gamma radiation from potassium is unimportant because there would be less than .001 per cent of the radioactive isotope (K^{40}) present in rocks such as pegmatite and granite which have high potassium contents relative to other rocks occurring in the Mount Painter area. Rubidium occurs so rarely in nature that its presence in the Mount Painter area is considered most unlikely. In any case there is no reference in standard works on radioactivity, e.g. Hevesy & Paneth (2) to any gamma radiation arising from rubidium.

On the other hand uranium occurs in some of the deposits in concentrations which exceed 1 per cent. The uranium isotope (235), the parent of the actinium series is always present with the principal uranium isotope (238) in the ratio of 1/140.

For reasons which will be given later in this report (3), it is believed that thorium occurs only as traces in the areas covered by the radiation surveys with the possible exception of Radium Ridge on which occurs the No.2 deposit where notable amounts of monazite have been reported to occur (4).

---

1. Rose, R.B. Radioactive Exploration, The Mines Magazine, Dec. 1941.
2. Hevesy G. and Paneth F.A., a Manuel of Radioactivity, 2nd Ed. p.51, Oxford Uni. Press 1938.
3. See page 6.
4. Mawson, Sir D. The Nature and Occurrence of Uraniferous Mineral Deposits in South Australia Trans. Roy. Soc. S.A. Vol.68(2) 1944.

In the gamma radiation table given above the letter W has been used to signify those radiations which are given in the Handbook of Chemistry and Physics (1) as weak. The table indicates that the gamma radiation from a uranium ore comes principally from the transformation of Radium C to Radium C^1. However, in measuring the gamma radiation from a uranium ore the relative intensities of the gamma rays from the various transformations will depend on the screening employed. Under the conditions of screening used with the Geiger-Muller apparatus described in this report, nearly all the soft gamma rays were excluded. Some absorption tests were made and will be described in detail in a later section of this report. Briefly, the results of the tests indicated that the radiation measured was composed of roughly 60 to 70 per cent hard radiation from RaC and 40 to 30 per cent hard radiation from UX_2. Generally, however, the screening used in the surveys and tests described herein was less than that used in the absorption tests and it is probable that the gamma radiation from the RaC to RaC^1 transformation accounted for more than 60 to 70 per cent of the total measured.

If the members of the uranium series are in equilibrium in a sample, the intensity of the radiation from it will be directly proportional to its uranium content (providing of course that no thorium is present). Such a relation would form a satisfactory basis for using radiation measurements for assay purposes. Unfortunately this desirable condition of equilibrium does not exist in samples from Mount Painter. R.G. Thomas (2) found that the ratio radium/uranium was $2.70 \times 10^{-7}/1$ whereas for a uranium mineral in equilibrium with its disintegration products it should be $3.4 \times 10^{-7}/1$. The ratio is only an average of those for uranium minerals of all ages in the samples. The minerals might include primary ones (which would be in equilibrium) as well as the torbernite and other secondary minerals. However, primary minerals have either not been detected or detected in only very small amounts, whereas secondary minerals are obviously present in appreciable amounts. The ratio therefore indicates that the secondary minerals must be considerably younger than the million years required for them to reach equilibrium.

However, there is another way in which the age of uranium minerals is indicated, namely by the ratio of lead in the ore to uranium content. Lead is the final disintegration member in the radioactive series and there is a definite relation between the lead/uranium ratio and the age. The ratio approaches 0.26 for the oldest known (lower Precambrian) uranium nimerals, e.g. uraninites from southeast Manitoba, Canada and approaches zero for younger minerals. Dr. Stillwell (3) states that "lead is not readily detected in individual crystals of torbernite and autunite ". In correspondence with the writers in explanation of the above, Dr. Stillwell indicated that the quantity of lead would be consistent with an age of less than a million years for these secondary minerals.

On the other hand R.G. Thomas (4) records that "many pounds" of lead chloride were separated from the Mount Painter ores during treatment. He states that no lead vessels were used in the separation and that "there are no recognisable lead minerals on the Mount Painter field" and concludes that this lead is derived from the disintegration of the uranium series. This is strongly supported by his atomic weight determination (205.8) which is very close to the value of 206 accepted for lead derived from uranium. An approximate estimate by the writers of the uranium in the ore treated by R.G. Thomas places it at less than 2,500 lb. and therefore for every 1 lb. of lead produced, the lead/uranium ratio would be 0.0004 and would represent an age of approximately 3 million years.

---

1. Handbook of Chemistry and Physics 25th Ed. Chem. Rubber Pub. Co. 1941.
2. Thomas R.G. Processing of Radium in South Australia, Aust. Chem. Inst. J. & Proc. Vol. 9 No. 6. 1942.
3. Stillwell Dr. F.L. and Edwards, Dr. A.B. Uranium Minerals from Mount Painter. This bulletin
4. Thomas R.G. op. cit.

The expression "many pounds" used by Thomas to describe the amount of lead chloride produced probably means an amount in excess of 10 lb. of lead[1] which would give an age in excess of 30 million years during which the uranium has been in the deposit, providing of course that the uranium and the lead produced from it have remained together in the deposit. Drs. Stillwell and Edwards describe (2) a mineral which was found in the workings and which they believe to be gummite (composition 92% UO_3, 2.8% PbO and 1% P_2O_5). Dana (3) describes gummite as being an oxidation product of uraninite, frequently occurring as complete pseudomorphs after uraninite. In the No. 2 Deposit gummite occurs in vughs, and in the No. 6 Deposit it is also present but is not a pseudomorph. Nevertheless it may be taken as evidence of the pre-existence in the No.6 orebody of primary uranium minerals. Furthermore it might be these primary minerals which have given rise to the greater part, if not all the lead that was recovered during treatment. The discovery of gummite containing lead in no way effects Thomas' conclusion that the lead recovered during treatment was derived from the disintegration of the uranium series.

There appears, therefore, to be a discrepancy between the average age of the uranium minerals as determined by the Ra/U ratio and that indicated by the lead/uranium ratio. It is believed that this apparent discrepancy can best be explained by the following -

i. Uranium, radium and lead derived from a primary uranium mineral of early Palaeozoic age[4] are present in the ore. The uranium and radium are present in amounts corresponding to equilibrium. The lead/uranium ratio would be high.

ii. Secondary uranium minerals (torbernite, etc.) were introduced into the ore in quantities relatively much greater than the primary uranium mineral. This introduction occurred approximately 400,000 years ago.

---

1. In correspondence with the writers, Mr. Thomas explained that no records were kept of uranium contents of the ore treated and in fact much of the uranium was discarded during treatment. He concedes that 2,500 lb. is a fair estimate. At the same time he advised that the estimate of 10 lb. lead in the ore is probably low in so far as his treatment would not have recovered all the lead.
2. Stillwell Dr. F.L. and Edwards, Dr. A.B. - in this Bulletin
3. Dana's System of Mineralogy, Seventh Ed. Vol.1 Chapman and Hall Ltd., London.
4. The youngest magmatic activity known in the Mount Painter district is not later than Palaezoic. The age of the primary mineral "Fergusonite" found in the No.2 workings and determined by its Pb/U ratio appears to be Palaeozoic. The Pb/U ratios found for two analyses by A.W. Kleeman are 0.093 and 0.058 corresponding to ages of approximately 700 and 440 million years respectively. The first of these analyses is quoted by Sir. D. Mawson (5) and appears to be that of a single sample of "Fergusonite". The other analysis however is that of samples in which every grain was subject to careful microscopic examination and according to Kleeman(6) -

   > "was carried out on five samples aggregating two grams in weight, shows 10.6% U_3O_8 and 0.72% PbO; that is a Pb/U ratio of 0.058 which corresponds to an age of 440 Million years. The difficulties inherent in the analysis and the results obtained suggest that the uncertainty is of the order of 10%. The age of the mineral may thus be stated as 450±50 million years".

   It is evident that the age determination of the latter samples is a more reliable one than the former. "Fergusonite" is a primary uranium mineral particularly resistant to weathering and it is not likely that the primary uranium mineral referred to in (i) above was "Fergusonite".
5. Mawson, Sir D. op. cit.
6. Kleeman, A.W. quoted in private communication of Sir D. Mawson to Director of Mines, Adelaide.

The results are that firstly the average ore contains only approximately three quarters of the radium compared with that which would be in equilibrium with the uranium present and secondly the amount of lead compared with the uranium gives a much lower age than that of the primary uranium mineral. That enrichment by the addition of secondary uranium minerals to the ore may have occurred is supported by the fact that radiation measurements prove that the brecciated zones which contain the isolated and small uranium deposits--more particularly those parts of the zones which have a high hematite content--are generally weakly radioactive. The radiation surveys indicated that there is widespread but very weak radioactivity throughout the crush zones, and further that those parts of the zones which contain much hematite have much greater radioactivity. It should be noted, however, that the greater part of the hematite contains no visible uranium minerals. At the No.6 deposit the outcrop of the brecciated zone is composed largely of massive hematite which was found to be increasingly radioactive as the No.6 ore shoot was approached. No uranium minerals were visible in the greatest part of the hematite outcrop although it may have contained a minute quantity. The East Adit - 30 feet level has been driven for 70 feet through massive hematite and the results of assays of samples from it average 0.03% U_3O_8. The radiation measured at the surface in the vicinity of this adit, however, was greater than would have been expected from such a low uranium content and it is suggested that the excess activity is probably due to the presence of decay products of uranium. Two mine samples from the No.6 workings were found by J.M. Rayner to have appreciable activity although they showed no uranium on assay. It is suggested that the activity displayed by these samples was due also to decay products of uranium, the original uranium having been leached from the samples.

It was thought at one time that the activity of the samples referred to above and of the massive hematite may have been due to thorium. While it cannot be claimed with certainty that thorium is not responsible (it would require chemical assay of samples of hematite to determine this) it now appears most unlikely for the following reasons. If thorium had been present in quantities sufficient to account for the excess radiation measured, then thorium lead (atomic weight 208) would have been recovered during the treatment of the ores. Thorium is a primary mineral and as such would most likely be of Palaeozoic age. Thorium of such an age would have a thorium-lead/thorium ratio of approximately 0.03 and the quantity of thorium which would have to be present to account for the excess radiation would be such that sufficient lead should have been produced to affect appreciably the atomic weight determination referred to by R.G. Thomas.[1] The atomic weight (205.8) found for the lead recovered seems to preclude the possibility of any significant amount of thorium lead being present in the ore and hence precludes significant amounts of thorium.

Dr. Stillwell[2] refers to monazite (the only known thorium-bearing mineral at Mount Painter) as being "scattered sparsely through all the prospects" with the exception of No.2 deposit on Radium Ridge where up to 30% monazite is recorded. Sir D. Mawson[3] quotes results of assays of three monazite specimens from Mount Painter as 0.3, 0.2 and 0.16% ThO_2 respectively. If the thoria contents are typical of Mount Painter monazite and if Dr. Stillwell's term "sparsely" is accepted as meaning less than 1% monazite, then the ThO_2 content of such deposits as the No.6 would be less than .003%. This figure is consistent with R.G. Thomas' statement[4] that the Mount Painter ore treated was virtually free from thorium.

---

[1] Thomas, R.G. op. cit.
[2] Stillwell, Dr. F.L. and Edwards, Dr. A.B. op. cit. (p.15)
[3] Mawson, Sir D. op. cit.
[4] Thomas, R.G. op. cit.

It is concluded, therefore, that the excess radiation from the mine samples which gave no uranium on assay, and from hematite outcrops free from visible uranium minerals is due mainly, if not entirely, to decay products products of uranium, such as ionium and radium, being present in the samples of hematite. There is thus evidence of a former widespread distribution of uranium minerals in small amounts in the crush zones and the hematite bodies and that the uranium was leached from them more than 1,000,000 years ago. The uranium from such sources would have sufficient to form the richer concentrations of secondary minerals (torbernite etc.) forming the shoot such as No.6.

IV. THE GEOPHYSICAL SURVEYS.

A. Surveys Undertaken

1. Field Work -

The field surveys carried out with a view to locating and delineating areas of high radioactivity were based on the relatively simple assumption that concentrations of uranium minerals in close proximity to the surface would form areas of high radiation. Further, it was assumed that the areas where radiation was most intense would be those most likely to contain concentrations of economic importance.

There are indirect reports that work of like nature has been done elsewhere. C.S. Lord[1] in a report on Bear Exploration and Radium leases in northwestern Canada stated that -

> "during the summer of 1939 an instrument designed to detect the presence of radioactive elements was used to explore the surface of the property in a search for pitchblende deposits. No pitchblende was found, but the work is said to indicate that it may occur in Zone 1 and 2 and in one other zone."

Ellsworth[2] discusses radiation surveys by the Russian geophysicists Bogoiavlensky, Lomakin and Cherepenikov who made a five years study of highly penetrating radiations of terrestrial origin in relation to the geology of known oil fields. It is claimed that gamma rays, harder than those from RaC, were measured, their source being diffused in the upper strata of the soil. It is also claimed that the radiation was stronger directly over oil deposits. The authors ascribe the result to oil beds being richer in radioactive elements than the clay layers which enclose them. Ellsworth concludes that -

> "If the results obtained by the authors are found to apply generally it would appear that another very valuable geophysical aid to prospecting, particularly for radium and for petroleum, has been discovered."

Recent work by R.M. Tripp[3] has confirmed that the soil overlying oil reservoirs may have a high concentration of the radioactive gases radon, actinon and thoron and their disintegration products but he ascribes this to the effect of gaseous hydrocarbons migrating upwards from the reservoir and bringing the radioactive gases with them; the sources of the radioactive gases being the sedimentary rocks overlying the reservoir. It is only the upward migration of the hydrocarbon gases that has produced the observed concentrations in the soil. The sedimentary rocks of the flanks of the oil reservoir would presumably contain a radioactive content comparable with those immediately above the reservoir but there would be no concentration by migrating hydrocarbon gas of raton, etc. in the surface soil.

---

[1] Lord, C.S. Mineral Industries of the Northwest Territories, Department of Mines, Ottawa, Canada, Geol. Sur. Memoir 230 (1941).

[2] Ellsworth, H.V. Rare-element Minerals of Canada, Econ, Geol. Series. No. 11 Dept. Mines, Ottawa, Canada (1932).

[3] Tripp, R.M. Measurement of Soil-Air Ions over the Fort Collins Anticline. Geophysics Vol.X, No.2, April 1945.

Heiland[4] refers to work done underground by Von dem Borne[5] and Behounek[6] to locate pitchblende seams, and states that

> "it is likely that measurements have been made at the surface for the same purpose in the Belgium Congo and in Canada, although definite reports have not been published."

Radioactivity surveys for purposes other than the location of uranium minerals have been numerous, and Ambronn[7] deals at length with various soil surveys in which the activity (mainly alpha ray) of the soil gas, containing radon, was measured. Soil gas activity has been used to locate faults, fissures, contacts and some types of mineral veins. The highly penetrating (gamma) radiation from these features, however, has been found by Ambronn[8] to be small relative to their alpha radiation. Heiland[9] states that -

> "anomalies caused by local concentration of radioactive products (faults and the like) may exceed those due to radioactive ores."

In view of Ambronn's findings, stated above, this statement is probably only true of alpha or total radiation, as measured by an ionization gauge. Nevertheless it is a factor which must be borne in mind when interpreting the results of a radiation survey.

Another factor of importance might be called the depth factor. As mentioned in the preceding section of this report the radiation from radioactive substances are absorbed by solid matter. There is a depth of soil or rock which will be sufficient to reduce even the most penetrating rays, i.e. the gamma rays, to negligible intensity, and any concentration of radioactive minerals, however large, below this depth cannot be detected at the surface by means of its radiation. The gamma rays from radium are reduced by one half by passing through 1.4 cm. of lead, but it is claimed by Rose[1] that they can be detected through 15 inches of lead. It is considered unlikely that any uranium minerals below a depth of three feet contribute a significant portion of the total radiation measured in the surveys to be described, and the effects are regarded as purely surface effects.

(ii) Estimation of Uranium in Samples -

Some of the factors that influence the problem of estimating uranium content of mine samples have been described above. Of these the main ones are -

(i) The ages of the uranium minerals insofar as they affect radioactive equilibrium.

(ii) The amount of thorium, if any, which might be in the samples. Gamma radiation from members of the thorium series is practically indistinguishable from that from the uranium series.

To these might be added such factors as the density of the sample tested, and other factors, (such as statistical fluctuations in the intensity of the radiation and method of screening) inherent in the method used or introduced by the technique adopted. These will be discussed in Section VI of this report which deals exclusively with the estimation of uranium content by measuring the radiation from samples.

---

4. Heiland, Dr. C.A. Geophysical Exploration. Prentice Hall Inc. N.Y.1940.
5. Borne von dem, Habilitationsschrift (Beslau) (1905).
6. Behounek, F. Phys. Zeit, 28, 333-342 (1927).
7. Ambronn, Dr. R., Elements of Geophysics, McGraw-Hill Book Co.Inc.(1928).
8. Ambronn, op. cit.
9. Heiland, op. cit.

1. Rose, R.B. op. cit.

B. The Instruments Used.

Four separate instruments which, for the purpose of description, may be called models 1, 2, 3 and 4, were used at Mount Painter.

The first of these (Model 1) was constructed in the workshops of the Physics Department, Adelaide University, by J. Symons and R.F. Thyer. The former was member of the Physics Staff whose services were made available through the kind co-operation of Professor Kerr Grant and Dr. Burdon. The instrument is a portable Geiger-Muller radiation counter incorporating a frequency meter. The Geiger-Muller tube is mounted in a stiff cardboard cylinder and is coupled to the rest of the circuit by a flexible cable about 25 feet long. The electrical impulses from the Geiger-Muller tube are fed into a tank circuit comprising a bank of condensers (20 microfarads) which discharges through a leak resistance of 5 megohms. The voltage on the condensers, and the leakage current through the resistance, reached a more or less steady value depending on the rate at which the impulses are received, i.e. on the amount of radiation passing through the tube in a given time. The voltage drop over portion of the leak resistance is used to bias the output tube whose plate current is used as an indication of the average rate of radiation impulses.

Time did not permit of a thorough testing of the instrument prior to field operations with a result that several fault developed and reduced the accuracy of the results. It was found that the characteristics of the valves used in the circuit were not constant-leading to uncontrollable changes in output current not associated with changes in impulse rate. Inaddition the output meter showed a zero drift which made correction of the readings difficult and, in some way, unsatisfactory. The instrument was not readily portable in rough terrain such as is characteristic of the Mount Painter area. It was used during August/September 1944 for radiation surveys of "Smiler Greenwood" and "Bentley Greenwood" deposits at East Painter and at the No.6 workings at Mount Painter.

The circuit of the above instrument was modified subsequently and two new instruments, one (Model 2) for use in field surveys and one (Model 3) for assay work, were built in the Mineral Resources Survey's workshop at Canberra. The circuit used in building the latter is given as Fig. (1). The circuit of the instrument already described (Model 1) and that of the field model (Model 2) differ somewhat in detail from Fig. (1), but are essentially the same in principle.

The assay instrument (Model 3) whose circuit is shown, comprises three main items - the Geiger Muller tube with supporting cradle, the amplifier chassis and the meter panel. Voltages are obtained from batteries, the high voltage required for the Geiger-Muller tube being obtained from 10 extra light duty 103½ v. batteries in series with 4 light duty 45 v. batteries, a total of 1215 v. Pulses from the Geiger-Muller tube pass through two stages of amplification. They are then rectified, and develop a voltage across a 12 microfarad condenser with a 2 megohms lead resistor. As the voltage across the condenser increases, its rate of discharge through the resistor increases and eventually a state of equilibrium is reached when the rate of discharge of the condenser is equal to the rate of charging. The voltage then remains steady and is measured by a vacuum tube voltmeter arrangement, the meter being kept at zero reading by a compensating voltage controlled by a potentiometer with a calibrated scale. The reading of the scale is taken as the measure of the activity, and experiment has shown that it is proportional to the count rate.

The field instrument (Model 2) differs from that described above in that the Geiger-Muller tube and the valve for the first stage of amplification are mounted in a steel tube and are coupled to the remainder of the circuit through a flexible cable about

35 feet long. The amplifier chassis, meter panel and amplifier batteries are mounted in a wooden case attached to a ruck-sack carrier while the high voltage supply (12 x 103½ v. extra light duty batteries) for the Geiger-Muller tube is mounted in another case also attached to a carrier.

The performance of Models 2 and 3 were considerably better than that of Model 1. Under test conditions in the workshop they appeared to be very stable as regards sensitivity and reasonably insensitive to change in voltage of the various batteries used. However, under field conditions where they were subject to large changes in temperature, and, in the case of the field instrument subject to relatively rough handling, it was found that their sensitivity varied over fairly wide limits. To overcome the difficulties introduced by this variation, a standard torbernite sample sealed in a copper tube, was used to check the sensitivity at frequent intervals, and all readings were then adjusted accordingly.

The instruments, Models 2 and 3, were calibrated for count rate as indicated by the mechanical counter. As designed, the instruments are capable of registering count-rates up to 2,000 per minute, the normal count-rate being of the order of 200 per minute.

The field equipment (Model 2) was used in surveys of areas in the vicinity of Greenwood's Camp, East Painter, during November-December, 1944. Owing to the limited size of the areas of high radiation values it was found advisable to make determinations at intervals of about 10 feet over a considerable portion of the area. Numerous factors combined to make this work slow and tedious, and it was recommended that a light weight unit, fitted with headphones, be built with the object of rapidly locating the areas of high radiation (over which detailed and quantitative work would subsequently be done), and eliminating such work over areas where the radiation was normal. When it is realised that over 70 per cent of the readings taken on the Greenwood Camp area with Model 2 were normal, and required just as much time to determine as high readings, the need for such a reconnaissance unit will be appreciated.

An instrument (Model 4) along the lines suggested was made by Mr. Dooley in January, 1945. It is a modification of a model described by R.B. Taft.[1] The high voltage (1280 v) is obtained from a Ford coil driven by two 1½ v. torch cells in series, and feeding a rectifier circuit. Only one stage of amplification is necessary to render the pulses audible in the phones. The amplifier valve is biassed beyond cut-off to eliminate variations in the voltage supply due to insufficient filtering, the pulses from the Geiger-Muller tube being large enough to be amplified when the interference is cut out. A circuit diagram is shown in Fig. (2). The whole equipment is built into a box 15" x 9" x 8" and can be operated by one man. Harness is provided for carrying the equipment on the chest of the operator.

The Geiger Muller tubes used in each of the Models described were made available to the Mineral Resources by Professor Kerr Grant whose assistance in this and other respects has been acknowledged elsewhere.

V. FIELD SURVEYS.

A. Procedure.

The procedure generally adopted in the field surveys was to place the Geiger-Muller tube (which was protected by an outer casing) on the surface of the ground at an observation point, and the activity was then read from the frequency meter. The observation points were closely spaced and were either on pegged traverses or

---

1. Taft, R.B. Two small portable Geiger-Muller Counters. Rev. Sci. Inst. Vol. 11, No. 2 (1940).

were surveyed at the time of reading by means of a plane table and alidade. The latter method proved most satisfactory and was most generally used.

The procedure of placing the cased tube directly on the surface might at first sight appear objectionable on the grounds that small flakes of uranium minerals might thus be placed in contact with it and by virtue of their close proximity give disproportionately high readings. Tests were made with the tube held at different heights of from 0 to 4 feet above the surface, and in nearly every case it was found that readings in contact with the ground bore a constant ratio to the readings at 4 feet. The only exceptions were where some of the tests for variation with height were made on mine dumps on which relatively large amounts of uranium minerals were exposed. The readings were found to be abnormally high and beyond the range of the instruments whether the tube was placed on the ground or at 4 feet.

In the survey of "Smiler Greenwood" and "Bentley Greenwood" deposits at East Painter, and the No.6 workings at Mount Painter, Model 1 instrument was used. It was realised early in the survey that the sensitivity of the instrument, as indicated by the frequency meter reading for the normal count rate, was variable. A check was kept on the behaviour of the instrument by making frequent "repeat" readings at a number of base stations.

Model 2 instrument, used for surveying the Greenwood's Camp area, was also subject to slow changes in sensitivity, due to changes in voltage (caused in part by changes in temperature) of the batteries used. This could not be eliminated, but a strict control was maintained by makinguse of a standard torbernite sample which was used to determine the instrument's sensitivity at frequent intervals. In each survey, the intensity of the radiation as indicated by the appropriate meter reading was reduced to the corresponding reading for a standard instrument sensitivity, and radiation contour plans were prepared. These will be discussed in detail in subsequent sections of this report, which deals with surveys of the individual areas.

In addition to the general procedure outlined above there are special techniques which varied with the individual areas and which will be described later in this report.

B. Results of the Field Surveys.

The results of the radiation surveys are given in the form of contour plans on plates 1,2,3,5 and 6. The contours are super-imposed on the surface geological plans for the purposes of correlation.

Two different units are used for the contour intervals due to different instruments being used. The radiation contours of the areas around "Smiler Greenwood", "Bentley Greenwood", and No.6 deposits are comparable with one another and were determined with Model 1 instrument. Likewise the contours on radiation plans for various parts of the Greenwood's Camp area are comparable with one another, but are not comparable as regards magnitude with those of the three areas referred to above.

The results of the individual surveys are discussed in detail below under the appropriate headings and the locality plan, Fig. 3, shows the position of the area relative to each other and to the Mount Painter district as a whole.

(1) The "Smiler Greenwood" Area, East Painter.

The "Smiler Greenwood" area is approximately 1½ miles north of Mount Painter, and its position relative to the other uranium occurrences is shown on the locality plan. Torbernite was first discovered by A.G. Greenwood in an extensive crushed zone, forming the summit of a ridge.

The geology of the area is described in a report by C.J. Sullivan[1] and only sich features of the geology as are essential to the proper understanding of the radiation survey results will be given here. Quoting from Sullivan's report -

> "A mass of porphyritic granite trending roughly north and south and dipping at 20 to 35 degrees to the west overlies a series of sedimentary and metamorphic rocks.--Close to the contact between the two groups of rocks the granite has been crushed and brecciated and the zone of brecciation forms one the main loci of ore deposition.-- The contact is a folded one and the maximum crushing takes place at the crests of small anticlines. In places there is a capping (?) of hematite with some manganese dioxide. Traces of torbernite have been found associated with the hematite and manganese. Other important features are the shear zones. The two main ones have lengths of 180 and 100 feet respectively and would average 10 feet in width. They consist of relatively soft material derived from the alteration of the porphyry."

a. Results.

The accompanying plan, plate 1, shows lines of equal ground radiation as measured by the Geiger-Muller instrument (Model 1) and it may be called a 'radiation contour' plan. The contours are super-imposed on the surface geology (after Sullivan) for the purpose of comparison.

The three most important features in the radiation survey results are -

(i) A general zone of high radiation values coincides with the outcrop of the crushed zone.

(ii) Small zones of higher radiation values were obtained over the prominent hematite outcrops, a feature common to all the areas surveyed.

(iii) Small zones of values higher than those in (i) and (ii) were obtained along the two shear zones at the southern end of the area covered by the survey, suggesting the concentration of radioactive minerals within the shear zones.

The dense hematite which occurs in this and similar crushed zones in the Mount Painter district appears to be relatively highly radioactive although in many places no obvious uranium minerals are visible in it. The reason for this has been discussed in detail earlier in this report[1] where it was shown that the most likely explanation is that the activity is due to disintegration products of uranium, the original uranium having been leached from the outcrops.

An interesting feature of the radiation results was the close correlation between the very high zones in the radiation contours and occurrences of uranium minerals. Some of these very high zones coincided with places where traces of torbernite or uranophane had previously been discovered. In one or two cases, however, there were no uranium minerals obvious at the immediate surface but when small holes were dug with a geological pick,

---

[1] Sullivan, C.J., East Painter - "Smiler Greenwood" working - elsewhere this Bulletin.

[1] Page 6 of this report.

torbernite or uranophane was unearthed at the points where the high radiation figure had suggested their presence. This correlation suggested that all the very high radiation zones should be thorouhgly examined by shallow pitting. This testing should reveal the extent of the uranium mineralisation which, it seems likely, is responsible for these high readings.

It has been stated earlier in this report and is again emphasised here, that the effect being measured is coming from material at a depth not greater than two or three feet. A concentration of uranium minerals at this shallow depth does not necesarily indicate concentration at a greater depth.

On the other hand, if Stillwell's theory[2] that the concentrations of secondary uranium minerals were brought about not by supergene waters but from epithermal solutions, then the occurrence of concentrations of torbernite etc. at the surface could be an accident of erosion and might in fact be the cap of a uranium deposit.

b. Testing.

Sullivan[3] recommended testing at twelve places. The selection was governed by -

(1) known occurrences of uranium minerals
(2) the geology, and
(3) the occurrence of the radiation highs.

The recommended sites are shown on the accompanying plan. (plate 1)

Some of the testing recommended by Sullivan was carried out. Shallow pits were dug at test sites 2 and 3 and a shaft was sunk on site 4. These test sites coincide with places where high radiation values were recorded.

The pits (sites 2 and 3) disclosed very irregular, low grade occurrence of torbernite. The shaft at site 4 was 13.5 feet deep, and one sample taken at a depth of 12 feet gave no uranium on assay.

(ii) The "Bentley Greenwood" Area, East Painter.

The "Bentley Greenwood" deposit is in the East Painter area approximately ¾ mile from Greenwood's Camp and about 1¼ miles north of Mount Painter. Its position relative to the other areas is shown on the locality plan. A radiation survey was made of an area approximately 500 feet by 200 feet which included the mine workings.

The geology of the area is the subject of a report by C.J. Sullivan[1] and it is proposed to outline here only such features of the geology as have a bearing on the geophysical results.

The rocks of the area, mostly granitic types, contain masses which are relatively coarse grained and often highly felspathic. These masses may be dykes, but their boundaries are indefinite; or they may be merely segregations in the granite. The rocks have a general east-west grain and are to some extent arranged in band in this direction. Some shearing has occurred, the shears following the east-west trend of the rock grain. The shears dip south at about 60° to 70°. Secondary concentration of uranium minerals has occurred to a limited extent in the most pronounced of these shears. It marks the northern limit of a mass of dense hematite of limited thickness which covers an area approximately 140 feet long b7 15 to 35 feet wide.

---

1. Sullivan, C.J. East Painter "Bentley Greenwood" working, elsewhere in this Bulletin.
2. Stillwell, Dr. F.L. and Edwards, Dr. A.B. Uranium Minerals from Mount Painter. This Bulletin p. 23.
3. Sullivan, C.J., op. cit.

Thickness of the hematite is only 2 or 3 feet where it is exposed in an open cut at its western end. In this open cut the hematite is resting on a relatively uncrushed and highly felspathic rock which contains some torbernite in cavities totally enclosed in the rock.

There is no pronounced crushed zone associated with this deposit as is the case in the "Smiler Greenwood" and No.6 deposits, but there is a small development of crushed material associated with the shears.

a. Results.

The instrument used for the survey was Model 1, and the results of the survey are given in the form of a radiation contour plan, plate 2, which also shows the surface geology (after Sullivan).

The contours show a general tendency to follow the main shear zone and ironstone outcrop. In the centre of the area, however, the contours show an enlargement of the high zone to the south of, and down hill from, the main shear. There are two possible explanations for this enlargement. The high radiation values may be due to the presence on the surface of uranium-bearing material shed from the vicinity of the ironstone outcrop or the main shear, or secondly the enlargement of the high radiation zone may truly represent conditions existing in the rocks insofar as it might be due to a segregation in the igneous rocks of a uranium-rich mass similar to the highly felspathic rock in the open cut.

The most easterly of the two highest radiation zones appears to be due to the presence of uranium rich material in the dumps and lying on the surface near the eastern tunnel, and consequently bears no relation to the geology. A small zone of radiation highs coincides with the main shear near its eastern end. A small amount of surface prospecting may be warranted in this part of the shear to reveal the extent of the uranium mineralisation which it is believed is responsible for the highs.

One other feature of interest in the contours is the relatively high zone of narrow width which occupies the northwestern corner of the area. The surface in the vicinity of this zone is covered by talus, and no geological feature was mapped which could be associated with the high values. As the zone of high values has an east-west strike, i.e. the strike of the grain in the country rocks, and also of the shears, it is probable that the radiation highs correspond to a slightly sheared or crushed zone. Radiation figures obtained over the zone were not very high and it is not considered likely that any appreciable accumulation of uranium minerals would be found along it.

b. Testing.

Three test pits were dug to test the southern enlargement of the radiation contour pattern. Their position relative to the radiation contours is shown on the accompanying plan (plate 2). They all encountered granitic rock with no sign of uranium mineralisation and it may be concluded that the first of the explanations offered above the the southerly enlargement of the contour pattern is the correct one, namely, that it is due to radioactive material shed from the vicinity of the ironstone outcrop or the main shear zone.

(iii) No.6 Deposit (Workings), Mount Painter.

The workings on the No.6 deposit are the principal ones in the Mount Painter district, and practically the entire output of uranium ore in the past came from them. Their position relative to the other uranium deposits is shown on the locality plan.

A radiation survey was made of an area of approximately 600' by 150' in the vicinity of, and the north of, the No.6 workings. It embraced the crushed and ironstone zones with which the ore shoot is associated.

The geology of this area is the subject of a report by E. Broadhurst[1], and only such features of the geology as have a direct bearing on the radiation survey results will be given here. The following notes on the geology are taken from Broadhurst's report.

The predominant rock types in the immediate vicinity of the ore deposit are banded gneisses which near the ore deposit have been silicified. A prominent feature of the area is a brecciated zone which occupied a high ridge running approximately north and south. Brecciation is confined to the silicified gneiss. Underground the breccia consists of angular quartzite-like fragments in a matrix consisting largely of hematite and felspar with the felspar often almost completely kaolinised. On the surface the breccia has been converted into dense hematite.

The ore shoot occurs at the southern end of the brecciated zone which extends from here for over 200' to the north.

At the southern end of the breccia near the lode outcrop numerous fault planes occur. These all strike east-west and dip at various angles from 30° to 76° to the south, and have horizontal slickensides.

The lode occurs on a pronounced east-west wall which can be seen in the open cut and at the 50' level. Considerable movement has taken place about this wall and the distortion and fracture which has resulted from the movement is closely associated with the localisation of the ore body. The fault planes which can be seen at the surface at the south end of the brecciated zone are believed to represent the upward continuation of the lode wall where it ceases to be a pronounced wall and has split into several planes.

About 300' to the north of the open cut there is an isolated outcrop of hematite with some coarse breccia. A fault plane, similar to those associated with the lode wall, is exposed in a trench which intersects the outcrop.

a. Results.

The results of the survey are shown in the form of radiation contour plan, plate 3, which also shows the surface geology (after Broadhurst).

The instrument used was Model 1 which has been described in an earlier section of this report.

The surface of the area in the vicinity of the open cut and mine workings is covered by dumps and loose material from the mine workings. This material contains flakes of torbernite and autunite and in consequence the radiation figures recorded were the maximum readable on the instrument. Such readings obviously bore no relationship to the normal radiation of the rocks underlying the loose material and for this reason the contours are omitted in this part of the areas.

An outstanding feature of the radiation results - and perhaps the only one of note - is the relation between the radiation highs and the occurrence of dense hematite.

---

1. Broadhurst, E. Mount Painter, No. 6 workings - elsewhere in this Bulletin.

Wherever dense hematite outcropped, radiation figures were high. This was a feature common to all the areas surveyed, and, as mentioned earlier in connection with the results of the "Smiler Greenwood" area, and in an introductory section of this report, the reason for the relatively high radioactivity possessed by the dense hematite is thought to be the presence of radium in the ironstone from which a primary uranium mineral has been leached.

The isolated high radiation zone at the northern end of the area coincides with an outcrop of dense hematite. A smaller high zone to the east of this also coincides with a small hematite outcrop. A trench through the first mentioned hematite outcrop has exposed an east-west wall dipping south. The maximum of the radiation contour plan occurs some 10 to 15 feet to the south of this wall, and it is possible that a wall similar to that described above may coincide with the radiation maximum. The association of high radiation values withthis ironstone together with the known presence of an east-west wall suggests that this is a place suitable for testing. Mr. Broadhurst has recommended such testing on geological grounds alone, but the geophysical results lend weight to the recommendation.

Two other radiation highs occur within the brecciated zone between the northern high zone previously discussed and the No.6 workings. These high zones coincide with outcrops of dense hematite but it cannot be claimed that the coincidence proves anything beyond the relative high radioactivity of the dense hematite.

In the absence of any features in the radiation results to suggest the presence of east-west cross fractures such as have localised the No.6 ore shoot, and in the absence of geological evidence of their existence, no recommendations for testing these high radiation zones are justified.

b. Testing.

A shaft was sunk at the northern end of the area at the site of the trench mentioned above. This appeared to be a favourable place for testing owing to the coincidence of a radiation high with an ironstone outcrop showing a well defined shear plane.

The shaft passed out of ironstone into granitic gneiss at a very shallow depth, but the shear was followed, on the incline, to a depth of 18½ feet, the shear being still present at this depth but indefinite. The shaft exposed no uranium minerals nor anything resembling lode matter.

As far as is known no surface prospecting was done at the place 10 to 15 feet south of the shaft where the radiation measured was a maximum for the northern part of the No.6 area.

(iv) East Painter (Greenwood) Camp Area, East Painter.

Greenwood's Camp area is situated approximately ¼ mile east-northeast of the East Painter Camp (site of Greenwood's old camp) at the junction of Heighty Creek and an unnamed creek along which runs the camel pad to the Mount Painter divide. Its position relative to other uranium deposits is shown on the locality plan. Geologists C.J. Sullivan and D.E. Gardner made a reconnaissance survey of the area during which they discovered several small torbernite occurrences. These were subsequently opened up by potholes or shallow shafts and are known as the Nos. 1 and 2 prospects. At a later stage an adit was driven for 50 feet under the No. 1 prospect.

The area was later mapped in more detail by E. Broadhurst and K. Llewellyn, and during the course of their examination Llewellyn discovered what is now known as the No.5 prospect.

As a result of his examination Broadhurst recommended that a Geiger-Muller radiation survey should be made of those parts of the area which he considered most likely to contain torbernite or other uranium minerals. The survey under review was carried out as a result of this recommendation.

a. Geology of the Area.

The following notes on the geology are based mainly on reports by E. Broadhurst [1,2].

This area is one of rugged topography with sharp ridges and spurs rising to 400 feet or more above creek level. Scree-covered slopes of 30° and over are a common feature of the area, and outcrops are mainly confined to the ridges and spurs or to slopes too steep for the accumulation of scree.

According to Broadhurst the rocks of this area consists of sedimentary gneisses and their brecciated and granitised remnants. Selective brecciation seems to have occurred insofar as certain beds, or groups of beds, have yielded more easily than others to brecciation. He states that -

> "the breccia shows all stages of replacement by hematite, from practically hematite-free breccia to rock consisting almost entirely of hematite. The occurrence of hematite is very irregular, and breccia rich in hematite seems to occur in scattered disconnected areas."

As will be shown later the areas rich in hematite were found to be generally the most radioactive--a feature common to all the areas surveyed by the radiation method.

In places the brecciated zones are intersected by shears and, owing to the softer nature of the sheared material, wide cracks or small gullies running through the ridges occur along the shears. These shears which seem to fall into groups (NW - SE and NE - SW) have been mapped only where they intersect the outcrops but from their positions on the plan it is possible that many of them intersect beneath the scree and soil surrounding the outcrops. Such intersections are regarded by Broadhurst as places most favourable for the accumulation or occurrence of uranium minerals, and the survey under review was designed in part to study the radiation from the shears and their likely intersections.

On one of these shears is a lode (the No.5) which contains appreciable amounts of torbernite. Broadhurst has described it as follows. The outcrop of this lode is about 20 feet from the main mass of brecciated rocks on a shear striking at 325°. The shear showed at the surface as a wall of hematite rock with a shear wall showing on the eastern side.

The lode has been developed by a shaft 40 feet deep and a drive at the 40 feet level. The shaft and drive were in decomposed ironstained granitic material showing occasional quartz veins. Little torbernite was seen in the shaft but torbernite was encountered in the drive mostly associated with shears along which clayey ironstone occurred.

From Broadhurst's description and from observation by one of the writers it was evident that this torbernite occurrence differs from the No.6 ore shoot in that there is very little hematite or manganese associated with it, and the quartz in the lode appears to be reef quartz different in occurrence and appearance from the epithermal quartz associated with other uranium occurrences in the Mount Painter area. Two possible explanations

---

1. See page elsewhere in this Bulletin.
2. See page elsewhere in this Bulletin.

are offered to account for these differences, namely -

(i) The torbernite has been deposited in and near the shear from circulating meteoric water, the source of the uranium being in the breccia which lies relatively close to the deposit.

(ii) The following possible explanation is based on a discussion with P.B. Nye, Assistant Director, Mineral Resources Survey, who visited the area during the course of the survey. He suggests that the quartz may have been introduced under hydrothermal conditions, and that the No.5 prospect may, in fact, be a primary uranium lode, the torbernite being derived from primary uranium minerals by the ordinary processes of oxidation.

Torbernite also occurs at scattered places in the breccia where it is generally, if not invariably, associated with hematite. In a number of such places the torbernite has been leached from the outcropping rocks and only cavities containing a white mineral, believed to be derived from the torbernite, remain. These leached outcrops are , however, radioactive and it is believed that in the leaching process insoluble radioactive salts, such as those of ionium and radium, may remain in the cavities.

About ten of these leached outcrops have been opened up by pot holes and revealed increasing quantities of torbernites at depth. Two such places are known as the Nos. 1 and 2 prospects and are thus indicated on the accompanying plan. The torbernite bodies revealed by this surface prospecting have an areal extent of only a few square yards, and they occur at intervals within the brecciated zone.

One of the objects of the Geiger-Muller survey was to determine whether or not these torbernite bodies occurred in any regular fashion and to delimit the area of breccia containing each body. Such information was of importance in laying out a programme for testing their ore potentialities at depth.

The principal object of the survey, however, was that of prospecting along, and in the vicinity of, the known shears particularly with respect to those places where shears might intersect.

b. Results.

The results of the radiation survey are shown in the form of contour plans, plate 5,6, which also shows some of the surface geology (after Broadhurst). The instrument used for the radiation measurements was the portable Geiger-Muller radiation contour, Model 2, which incorporated a frequency meter for measuring the rate at which the gamma rays cut the Geiger-Muller tube.

The normal counting rate for the Greenwood's Camp area was found to be equivalent to 2 units on the frequency meter, corresponding to a frequency of approximately 200 impulses per minute. Readings as high as 20 units were measured, this frequency being approximately 2,000 umpulses per minute. In a few isolated cases, (for instance over the outcrop of No.5 lode which contained visible quantities of torbernite), the frequency readings were considerably higher than 20 units, but the exact value could not be determined as the readings were beyond the range of the instrument.

During the survey it was found that many of the radiation high spots were extremely local in occurrence and consequently observation points in such areas were spaced at intervals of 5 feet and less. The average spacing of the observation points, however, was of the order of 10 feet.

Most of the small areas over which high radiation was measured were on rock outcrops and the effects measured were evidently due to radioactive minerals within the rock itself. However, as much of the work was done on scree-covered slopes, it is of importance at this stage to examine the probability of discovery by these means of any torbernite rich zones which might be covered by such material.

In the first place it should be clearly understood that the nature of the radiation measured, namely gamma rays, is such that about two or three feet of solid rock is sufficient to reduce their intensity to practically zero. In other words the gamma ray effect measured at the surface has its origin within two or three feet of that surface in the case of rocks and perhaps somewhat greater depth in the case of soil or scree material.

In the case of soil or scree however, there is the possibility that material shed from a uranium-rich zone may be close to the surface although the zone itself might be covered by a few feet of scree or soil. This is particularly true of steep slopes covered by soil or scree, such as were common in the Greenwood's Camp area. Support is given to some extent by the results obtained near the No.5 prospect. This prospect or lode occurs in a shear which outcrops only in one or two places. An examination of the radiation contours shows a closure of high radiation values along the strike of the lode. The contour lines for lower radiation values, viz. for 3, 4 and 5 units, show a marked tendency to depart from the strike of the lode and in fact have an attenuation downhill from the lode outcrop in a direction at right angles to the ground contour lines, a few of which are shown on plate 5 in order to illustrate this point. This fact is taken as evidence that the radiation represented by observations between the 3 and 5 contour lines is coming from radioactive material shed from the lode. It is quite probable that had the No.5 prospect been entirely covered by scree its presence could have been detected by virtue of the radio-active material in the scree above, and downhill from it.

It is possible, however, to visualise conditions whereby a radioactive lode and the material shed from it was later completely covered by three or more feet of non-active material in which case no radiation effect from it could be detected at the surface.

For the purpose of describing the results, the ground covered by the radiation survey has been divided into three areas which are separate portions of the Greenwood's Camp area. These are shown on plate 4 in their relationship to one another and to the Greenwood's Camp area as a whole.

<u>Area 1</u> is in the vicinity of, and includes, the Nos. 1 and 2 prospects. The results of the survey are shown in the form of a radiation contour plan (plate 6). Contour values range from 2 units which might be taken to indicate non-radioactive materials, to over 10 units. The latter value was obtained over an outcrop showing traces of torbernite.

It was in this area that Sullivan and Gardner discovered a number of small torbernite occurrences. The potholes and shafts shown on the plan represent places where these discoveries have been opened up. The radiatiob survey confirms what the surface prospecting has already suggested, namely, that the radioactive minerals occur as small bodies within the breccia.

Near the No.2 prospect the following character might be read into the contour pattern. In an area of brecciated rock about 250 by 40 feet the radiation values, and hence radioactive contents, are higher than normal. Within this area there are a number of isolated patches of breccia from which the radiation exceed seven units. The majority of such patches are places where torbernite, or leached cavities after torbernite, have been found and it is reasonable to assume that all such areas enclosed by the 7 unit contour line warrant prospecting for torbernite. This value of seven units has been arrived at from a purely empirical relationship between known torbernite occurrences and radiation values but it serves as a basis for assessing the possibilities of the high radiation patches found at various places on the area.

Apart from the general attenuation of the moderately high radiation zone in a northeast-southwest direction there is no regular arrangement of the isolated high spots. From their distribution and the nature of the breccia it is suggested that they are small disconnected bodies of torbernite-rich breccia within the main mass of breccia.

It is an observed fact that these small torbernite bodies are generally, if not invariably, associated with occurrences of hematite, although hematite occurs elsewhere in the area with little or no radioactivity.

Radiation survey results somewhat similar to those described for No.2 prospect were obtained over and near the No.1 prospect. Radiation values higher than normal are more restricted in occurrence than at the No.2 prospect, but the general discussion of results in the latter case apply equally well to No.1.

Area 2 is that portion of the Greenwood's Camp area which includes the No.5 prospect and surveys pegs No.s 6 and 9. Its position in relation to the Greenwood's Camp area as a whole is shown in Plate 4.

The main purpose in surveying this area was to study radiation in the vicinity of the shears, particularly near their probable inter-sections. The results of the radiation survey are shown in the form of a radiation contour plan on plate 5. With the exception of the shear on which the No.5 prospect is situated, none of the shears proved to be radioactive and, in general, they appear to have no influence at all on the distribution of the radioactive highs.

As at Area 1, radioactive high spots seem to be confined to isolated and disconnected bodies within the main mass of brecciated rocks, with the exception, of course, of the No.5 prospect.

It was an observed fact that many, if not all, of the high spots in the breccia coincided with hematite occurrences. Further, the degree of brecciation as indicated by the size of the angular fragments comprising it, seemed to influence the distribution of the radioactive highs insofar as these highs were confined to those portions of the outcrop where the rock fragments were smallest. The lack of activity in the shears is of interest. Many of these shears are stained with manganese and iron oxides which might be taken to indicate that they were channels through which hematite and uranium minerals were introduced into the breccia. Had this been so, however, it is difficult to explain the absence of appreciable radioactivity even when measurements were made in direct contact with the manganese and iron-stained portions of the shears. The occurrence of torbernite in the shear on which the No.5 lode is situated and the accompanying high radiation values is an exception to this generalisation. In this prospect, the torbernite occurs chiefly as platey encrustations in cracks and on cleavages in the rocks. There appears to be little, if any, hematite, manganese-staining or epithermal quartz in the deposit such as would be expected if the uranium minerals were deposited from epithermal solutions in this shear. Two explanations are advanced to account for the occurrence of torbernite without breccia or hematite. Firstly that the torbernite in the No.5 lode may have been deposited from circulating meteoric water, the source of the uranium beng the scattered bodies of uranium minerals in the breccia which lies relatively close to the deposit, and secondly, that the No.5 prospect may be a primary uranium lode, the torbernite being derived from uranium minerals by the ordinary processes of oxidation. If either of these explanations is correct then the occurrence of radioactive minerals in this shear would not necessarily constitute an exception to the general conclusion that the shears are not radioactive.

Prospecting the radioactive highs should be based on the premise that areas enclosed by the 7 unit contours may contain appreciable amounts of uranium minerals. To facilitate the location of such places on the ground, indication pegs have been placed in the positions shown on the radiation contour plan. In the area under discussion, namely Area 2, the indication pegs so placed are G.M. 7 to G.M.12 inclusive. They are case wood pegs approximately 3 inches wide by 15 inches long and are held in place by cairns of stones. The appropriate number has been written on each.

<u>Area 3</u> lies due south of Area 2 and includes survey pegs Nos. 10 and 11. Its position relative to Areas 1 and 2 can be seen on the locality plan, plate 4.

Broadhurst's mapping shows that ironstone outcrops over a portion of the area and that two shears cut across the area. The recognised association of hematite and radioactive mineralisation led to Broadhurst&s recommendation for a radiation survey of the area.

The results of the radiation survey are shown on plate 5, in the form of a contour plan. As at Area 2, the shears, in general, were not radioactive and appear to have little influence on the contour pattern. At the southwestern end of the main shear, however, near survey peg No.11, radiation values rise to approximately 10 units. A small amount of torbernite and leached cavities after torbernite are visible in the wall rocks at this point.

As for Area 2 the pattern of high radiation zones is irregular. The association of these high zones with hematite occurrences is more marked in this area than on Areas 1 and 2, the hematite being for the most part the massive dark variety such as is found in the outcrop of the No.6 deposit at Mount Painter. On the other hand, this massive hematite is not uniformly radioactive as an examination of the contour pattern will show. Portions of the hematite outcrop as mapped by Broadhurst lies between the 2 unit and 3 unit contour lines, or in other words, it has an activity only slightly greater than non-radioactive rocks.

<u>c. Testing.</u>

As for the other areas, testing was recommended over those portions of the surface bounded by the 7 unit contour lines. To facilitate the location of such places, indication pegs, numbered from G.M. 1 to G.M. 6, were placed on the area, their positions being shown on the contour plans, plates 5 and 6.

A limited amount of testing was done as a direct result of the radiation survey. The zones of high radiation were very small and irregular and it was obvious that, even if they were found to correspond to uranium occurrences of grade sufficiently high to constitute ore, it would be difficult and perhaps uneconomic to mine them. As a consequence of this the limited manpower available for testing the Greenwood's Camp area was concentrated on developing the No.5 lode and in driving an adit into the No.1 prospect. This testing was based mainly on geological findings but the radiation results supported and explained those findings. The only testing which was done entirely as a result of the radiation survey consisted in the first place of knapping the outcrops in the vicinity of the indication pegs to reveal any torbernite present. Later, chip samples were taken from the outcrops near the pegs.

Mr. Broadhurst examined the sites of the indication pegs and prepared geological notes which appear in his report[1]. The knapping and geological examination revealed at some places the presence of minor amounts of torbernite, mainly in dense hematite. The chip samples were examined by Professor Kerr-Grant[2] who compared their gamma ray activity with that of a standard sample, presumably of Mount Painter ore, assaying 0.09 per cent U_3O_8. Estimated U_3O_8 contents are given for seven of the twelve samples and range from 0.14 to 0.018 per cent. The activities of the remaining five is given as less than 0.1 per cent but were not accurately determined. The samples were later assayed by T.W. Dalwood with results ranging from nil to 0.20 per cent U_3O_8. There is fair agreement between the estimated and assay U_3O_8 contents. The highest assays are 0.20 and 0.14 per cent for G.M. 10 and G.M. 2 respectively and compare with the estimated 0.14 and 0.11 per cent. In five other samples where a similar comparison can be made the estimated U_3O_8 contents are higher than those found by assay which suggests that some uranium may have been leached from them, the radiation measured coming principally from unleached radium in them. A tentative programme for testing them by blasting off the top two or three feet (i.e. that part most subject to leaching) was discussed with Broadhurst but the need for concentrating manpowder on more favourable prospects prevented this programme from being put into effect.

(v) Radium Ridge Area - Mount Painter.

A limited amount of work was done over several hematite outcrops on, and near, Radium Ridge. The work was of a reconnaissance nature using the portable telephone equipment, model 4, and estimations of radiation intensities were made aurally.

The normal count-rate of the Geiger-Muller tube used was about 240 pulses per minute. When the count-rate rose to about three times this value, it was practically impossible to distinguish any further increase, as an almost continuous series of clicks could be heard from the headphones. Thus while a general area of high activity could be located, it was impossible to locate a maximum within the area, or to compare activities of different high areas. If a Geiger-Muller tube with a normal count-rate of say 15 to 20 pulses per minute were used, the instrument would be of more general use, and might eliminate the need for a survey by the more detailed methods which were used in the surveys previously described.

Areas where high radiation was detected are shaded on an accompanying loaclity plan, Fig. 3.

High activity was confined to the hematite outcrops, more particularly where the hematite was manganiferous. Traverses over some non-ironstone outcrops showed no detectable change in activity from that of the surrounding areas.

The work was done in conjunction with Mr. C. Sprigg, geologist who knapped over the hematite which gave high radiation but could find no visible torbernite or other uranium minerals to account for the radiation observed.

VI. DETERMINATION OF URANIUM CONTENT BY RADIATION MEASUREMENTS.

A. INTRODUCTION.

The work described hereunder was carried out at Mount Painter during February and March, 1945. A large proportion of the mine samples assayed prior to the commencement of this work had been found to contain little if any uranium. A request was made therefore by the Director of Mines, South Australia, that an approximate determination of uranium content, based on radiation measurements, should be made of mine samples at Mount Painter with a view to eliminating chemical assay work on very low grade samples.

---

1. Broadhurst, E., East Painter, elsewhere in this report.
2. Kerr-Grant, Prof. Communication to Director of Mines, Adelaide.

Some preliminary work was carried out at Canberra by J.M. Rayner, Chief Geophysicist, Mineral Resources Survey. He made radio-activity tests on samples of Mount Paint ore which had been assayed chemically, and established that, for the samples tested, there was an approximately linear relation between count-rate and uranium content.

It was realised at the outset that there are serious limitations to the accuracy of this method of uranium determination, as will be discussed below, but no difficulty was anticipated in eliminating those samples which contained no uranium or only traces.

This report covers work including setting up and calibrating the assay equipment, and testing 100 samples from the Mount Painter No.6 workings and 12 samples from East Painter No.1 adit.

B. THEORETICAL CONSIDERATIONS.

Several difficulties are encountered in making estimates of uranium content based on radioactivity measurements. These prevent the method from being used for accurate assay work, but provided due precautions are taken, approximate estimates can be made.

Uranium as found in natural deposits is accompanied by a series of radioactive disintegration products, and radioactivity measurements as made in the present work represent the gamma ray activity for all these products. If the uranium has been in situ and no member of the series has been removed by leaching or other causes for a period exceeding about a million years, the series will have reached radioactive equilibrium, and each disintegration product will be present in a constant proportion. In these circumstances the total radioactivity will be proportional. to the uranium content.

It has been shown, however, in an earlier section of this report, that uranium in the ore from the No.6 shoot is not in equilibrium with its disintegration products. R.G. Thomas'[1] determination of the average radium uranium ratio was 2.7 x 10^{-7} against a theoretical 3.4 x 10^{-7} for radium and uranium in equilibrium. If, however, the ratio found by Thomas applied also to the brecciated zone and is uniform throughout the zone, and if thorium is either absent or present only in minor amounts in the ore, then there would be a linear relationship between the uranium present in the samples from the brecciated zone and the gamma-radiation measured from them.

It was pointed out earlier in this report that the principle source of gamma rays from the uranium family is the transformation Ra C to Ra C'. Owing to the short half life periods of the disintegration products between radium and Ra C the latter will be in equilibrium with the radium. It follows, therefore, that the use of radiation measurements as a means of determining uranium content of ore samples will be effective only if the Ra/U ratio is constant for all samples. Conversely if the ratio between radiation measured and uranium content is constant for a number of samples it can be assumed that for them the Ra/U ratio is constant.

A set of ten samples from the No.6 deposit, Mount Painter, was tested at Canberra by J.M. Rayner. He found an approximate linear relation between count-rate and uranium content, so that for these samples, there is a reasonably constant Ra/U ratio. The test was repeated by J.C. Dooley at Mount Painter with seven samples from the No.6 deposit Mount Painter (mostly different from those used by J.M. Rayner and probably not from the No.6 shoot) with similar results. The graph Fig 4 shows that with these samples (Set No.2) there is also an approximately linear relationship between gamma radiation and uranium content and hence a relatively constant Ra/U ratio. The relationship between gamma radiation and uranium content was also determined for a set of samples (Set No.1) which had been prepared

---

[1] Thomas, R.G. op. cit.

from a quantity of clean torbernite (probably from the No.6 shoot) diluted with crushed quartzite. The graph is shown in Fig. 4. It is a straight line indicating a constant Ra/U ratio as would be expected but the interesting feature of the result is that it indicates that the radiation measured for a sample of given U_3O_8 content is on the average 30% less than for a sample of equal U_3O_8 content from set No.2. If the age of the secondary uranium minerals in the mine samples (No.2 set) was the same as in the prepared set (No.1) then it may be assumed that the mine samples contain radium in excess of that produced from the uranium they contain. In connection with this problem Mr. Dooley carried out a series of absorption tests using three samples, one from Set No.1, one from Set No.2 and one from 'ilmenitic' ore from Radium Hill. He determined absorption curves by using a series of ten lead screens of individual thickness 0.115 inches and his curves are shown in Fig. 6. His absorption curves have been compared with theoretical curves for the absorption of the two hardest components of the gamma radiation from the uranium family namely from the transformation of UX_2 and RaC with absopption coefficients of 0.72 cm. and 0.50 cm. Pb respectively. A comparison of the observed curves with the theoretical reveals that their absorption characteristics lie between those of the two gamma rays quoted above.

It is estimated that the radiations from the three samples examined could be of the form as under -

Sample	% U_3O_8	% Gamma Rays from UX_2	X Gamma Rays from RaC
Radium Hill	Unknown	34	66
Mount Painter ore (set No.2)	1.44	29	71
Mount Painter torbernite sample (set No.1)	2.00	38	62

These figures suggest that Mount Painter ore (Set No.2) has a greater Ra/U ratio than either the sample from Radium Hill or Mount Painter torbernite; they also suggest the possibility of distinguishing by such absorption tests between the radiation from uranium and that from radium in any given sample. UX_2 will be in equilibrium with uranium because it and the product (UX_1) intermediate between uranium and UX_2 have short half periods and will therefore require only a short time (of the order of months) to ready equilibrium.

As radium salts are very insoluble it is consluded that the excess of radium in Set No.2 as compared with Set No.1 has been produced from uranium that was previously in the samples and has been removed in solution. The results of the tests with No.2 Set of samples and tests with mine samples in general indicate that on the average the samples contain more radium that is accounted for by radium produced from the secondary uranium minerals present in samples.

Accepting the above explanation for excessive radium content, and hence excessive radiation in relation to uranium content, it is seen that there will be a minimum Ra/U ratio, determined by the age of the secondary uranium minerals below which the Ra/U ratio of none of the samples will fall. On the other hand this minimum Ra/U ratio may be greatly exceed as is evident in the case of the two mine samples described earlier in this report which showed appreciable activity (hence radium content) and yet assayed Nil for uranium.

As already stated, Thomas found an average Ra/U ratio of 2.7×10^{-7} for ore from the No.6 shoot instead of the ratio of 3.4×10^{-7} which corresponds to equilibrium conditions. It follows that, if the secondary uranium minerals have only 80% of the radium content that they would possess in equilibrium, the radium content of the minerals from which this uranium was leached to form the secondary uranium minerals must now have only 20% of their original radium content. One of the two samples mentioned in the preceding paragraph had an activity similar to that which would have been measured for an average sample containing 0.07% U_3O_8. No uranium was detected by chemical assay so it might be concluded that this sample originally contained about 0.25% uranium (the radium in a normal sample being responsible for about 60 - 70% of the activity measured).

In the above discussion it has been assumed that RaC whose transformation to RaC' is responsible for the greatest part of the gamma radiation measured, is in equilibrium with the radium. It can be assumed that the radium is sufficiently old for this condition to be attained but there is a possibility that, on the preparation of the mine samples for testing, one of the disintegration products intermediate between radium and RaC and which is a gas, namely radon, might be lost, thus upsetting the equilibrium conditions. This gas is present 9 only in extremely small amounts and is normally occluded in the solids. Nevertheless Beers and Goodman[1] claim that in preparing rock samples for radioactivity tests for the purposes of determining their ages they found that in some cases the pulverising of the samples resulted in a loss of radon. They consequently made a practice of keeping the pulverised samples sealed for 30 days before determining their radioactivity, and thus restored equilibrium.

The activity of one Mount Painter sample was measured 7 hours after pulverising and again 7 days after pulverising. As the half-value period of radon is 3.86 days, this time should have been ample to indicate any change; however no increase in activity was shown. Later, the activity of another Mount Painter sample, which had been kept sealed in a bottle for three months, was tested in the bottle. The sample was then removed from the bottle, thoroughly stirred and shaken, and after two hours was replaced in the bottle and tested again. If any radon had escaped while the sample was exposed to the air, a decrease in activities should have been shown, as the two hours was sufficient to allow for the decay of the short life products radium A, B,C,C' and C", However no decrease in activity was found. From these tests it was concluded that no radon was lost from the samples tested.

The total radiation emitted by the radioactive parts of a sample in the appropriate direction does not all reach the Geiger-Muller tube owing in part to absorption of the rays by the sample itself. Thus samples with different coefficients of absorption give different activity measurements for the same uranium content. This effect is more marked with the alpha and beta rays, which are far more highly absorbed than the gamma rays. As each sample to be tested was placed in iron tubes of wall thickness about 2 mm., all the alpha rays and beta rays were absorbed, together with the weaker gamma rays. Thus the measurements depended on the more penetrating gamma radiation, and discrepancies could be further lessened by ensuring that no great thickness of sample was traversed by the rays. The maximum thickness in the tests described in this report was equal to the internal tube diameter, i.e. about half an inch.

Moreover, according to Hevesy and Paneth[2] the gamma ray absorption coefficients for various substances are very nearly proportional to the density of those substances.

In the tests described below, a given weight of material was always packed into the same space; thus the effective density of samples under test was constant, and the absorption effects should be the same for all samples.

---

1. Beers, R.F. and Goodman, C. Distribution of Radioactivity in Ancient Sediments, Bull.Geol.Soc.Amer.Vol.55, No.10 (1944)

2. Hevesy, Gand Paneth, F.A. op. cit.

It should be noted that uranium itself emits practically no gamma rays; these all come from disintegration products. Hence the necessity for a constant ratio of the amounts of these products to the amount of uranium present if emphasized.

The presence of thorium minerals in a sample would give a high activity reading in relation to uranium content. However, it has been shown earlier in this report[1] that it is unlikely that thorium occurs in the samples tested in quantities greater than traces.

Thus it will be seen that accurate determinations of uranium with a Geiger-Muller counter are not possible. However, if measurements are based on comparisons with samples from the same area which have been chemically assayed, and if gamma rays are used, very approximate estimates of uranium content can be made; though checks should be made by chemical assay of any samples, the results of which are considered important.

C. EQUIPMENT AND PROCEDURE.

The Geiger-Muller counter used for this work was that described already as Model 3. The Geiger-Muller tube is supported in a wooden cradle, constructed in such a way that five iron tubes, each about one foot in length and ¾-inch diameter, can be arranged around the Geiger-Muller tube parallel to its main axis. These tubes are filled with the samples to be tested.

Samples to be tested were crushed, pulverized and thoroughly mixed, and each of the metal tubes was filled with 72.5 gm. of the pulverized sample, making a total of 362.5 gm. or about 0.8 lb. The material in each tube was packed down just sufficiently to allow insertion of the stopper. The five tubes were placed in position in the support as described above while measuring the activity of the sample. It was found that the normal reading of the instrument with the five empty tubes in position was about 13 per cent lower than when the tubes were removed. Hence all readings were taken with five tubes in position, empty ones being replaced with full ones as required.

The activity of any sample was measured by taking a series of readings with and without the sample in position. The reading for the sample was then taken as the difference between the mean normal reading and the mean reading with the sample in position.

In actual practice the meter reading did not remain steadily at zero having once been set there, but fluctuated continuously owing to random variations in the count-rate. Hence it was found necessary in taking a reading to set the meter as near to zero as possible, then to find the mean of the meter fluctuations over a period and apply a corresponding correction to the scale reading. This random variation of the reading determined the smallest quantities of uranium which could be detected, as samples whose activity was less than the mean variation in the normal reading gave no effect which could be measured reliably.

The sensitivity of the instrument varied considerably, probably due chiefly to variations in voltage caused by ageing of batteries and by temperature changes. One metal tube was filled with 72.5 gm. of the standard sample containing 2.0% UO_3; and this was used as a check on the sensitivity of the instrument. The standard reading for this sample was taken as 2.5 scale divisions, other readings being reduced in proportion to its actual reading. This standard sample was used in preference to the normal reading to determine the sensitivity, as it gave a reading over that portion of the scale which is used for measurements on samples, and it would be available if the equipment were moved to another site where the normal reading might not be the

---

[1] See Page 6.

same. It was found, however, that the ratio of standard sample reading to normal reading was reasonably constant, and measurements of the standard were made only occasionally.

There was a noticeable decrease in sensitivity during any set of readings. Hence measurements of the sample and normal readings were alternated in a symmetrical arrangement, so that when averages of the readings were taken, errors due to a steady decrease of sensitivity were eliminated. The procedure finally adopted was as follows: normal, sample, sample, normal; then repeat. It was found necessary to wait about two minutes between readings when there was likely to be any change of reading, to ensure that the condenser had charged or discharged to its new equilibrium voltage.

D. CALIBRATION.

Two sets of standard samples were used in calibrating the instrument. The first set was prepared by Mr. Dalwood, South Australian Government Analyst, from clean torbernite diluted with crushed quartzite, there being about 1 lb. of each sample. Samples were supplied in intervals of 0.1% from nil to 1%, and also for 1.5% and 2.0% UO_3. The second set consisted of seven samples from the No.6 deposit at Mount Painter and which had been assayed. There was 100 gm. of each sample, values being .05, .10, .17, .25, .36, .40 and 1.44% U_3O_8.

With the first set of samples, readings were taken using all five tubes from 0 to 0.4%; using three tubes from 0.1% to 1.0%; and using one tube only from 0.1% to 2.0%. Each set of readings shows a linear relation between scale reading and percentage, as shown in the graphs fig. 5; the ratio of the slopes of the graphs being 4.7; 3.1; 1.0.

Owing to smaller quantities of the second set of samples being available, measurements were made using one tube only. The readings corresponded approximately to a straight line (it should be remembered that when using one tube with small percentages, the experimental error is relatively great). However, the mean ratio of reading to percentage was about 1.5 times greater than with the first set of samples (see graphs, fig.4). The difference due to expressing percentages in terms of UO_3 or U_3O_8 is only 2%.

As explained in an earlier section of this report[1], the difference in the ratios (reading/percentage U_3O_8) for the two sets of samples, has been assumed to be due to the mine samples (Set No.2) containing some radium which was formed from uranium no longer present in the samples, in addition to the radium which has been formed from the secondary uranium minerals in the sample. This assumption is supported by the absorption tests which have been described in the preceeding section of this report.

Because it was believed that the mine samples (Set No.2) would more closely represent additional samples from the mine than would No.1 Set, the ratio of instrument reading to uranium content obtained for the No.2 set was used as a basis for estimating the uranium content of mine samples.

As it was anticipated that the great majority of samples to be tested would be of low grade, five tubes of these were used in order to obtain readings as high as possible. These readings were compared with the 5-tube graph of the No.1 set of samples; but estimates of percentage were based on the No.2 set, a correction factor being applied in accordance with the graphs of fig.4. This gives a ratio of reduced reading to percentage of 9.0 : 1.

---

1. See page 23 this report.

E. TESTS ON UNASSAYED MINE SAMPLES.

The activity measurements had a mean uncertainty of about 0.01%, Hence this was the lowest quantity that could be detected, though not always reliably. Samples tested were classified as follows -

Estimated U_3O_8 Content based on Activity Readings.	Class.
0 (No measurable activity)	N - Nil - Very probably less than 0.1% U_3O_8 if any
0.01%	T - Trace - of the order of 0.01%
0.02 to 0.07%	L - Less than 0.1%
0.08 to 0.12%	C - About 0.1%
0.13% and higher	M - More than 0.1%

Samples in the last two classifications were recommended for assay. A summary of the results is given.

Class	Mount Painter No.6	East Painter No.1 adit.
N	44	-
T	14	-
L	32	4
C	9	5
M	1	3
Total	100	12

All samples from the No.6 workings were channel samples, while those from East Painter were bulk samples. Samples submitted for test were as follows -

MT. PAINTER NO.6 WORKINGS.

50 ft. Cross cut - 16 samples taken along the wall over 3 ft. sections. Traces found in nearly all samples.

80 ft. Sub-level - 18 samples at 3 ft. intervals, 8 ft. to 59 ft. Definite radioactivity all along, rising towards the end with a maximum at No.16, at 53 ft.

100 ft. Drive - 25 samples at 3 ft. intervals. Very few indications of activity for the most part; definite but small activity from No.19 onwards.

100 ft. Cross-cut - 10 samples taken along the wall over 3 ft. sections. Traces in some.

Winze from 100 ft. level - 12 samples at 3 ft. intervals, depth of 18 ft. to 51 ft. Only one sample showed a trace of activity.

150 ft. Sub-level - 2 samples only, no indications of activity.

South Portal Drive - 17 samples at 3 ft. intervals; definite indications of activity from samples Nos. 1 to 10, the remaining samples were on the opposite side of a shear, and had no activity. Greatest activity was from samples near the beginning of the series.

EAST PAINTER NO.1 ADIT.

12 bulk samples, from 12 ft to 50 ft. Definite indications of radioactivity all along, highest reading from sample No.8 (32 ft. 9 in. to 36 ft.)

Details of the measurements are shown in the tables appended. Ten samples from the No.6 workings (nine class C and one class M) and eight (five class C and three class M) from East Painter No.1 adit, were recommended for assay. The results of the chemical assays of these samples are also shown in the tables.

It will be seen that in the case of the samples from East Painter there is a reasonably close agreement between the chemical assay results and estimation of U_3O_8 based on radiation measurements, or in other works the Ra/U ratio for these samples was approximately uniform and the same as for the No. 2 set of mine samples.

The chemical assay results for the ten samples from the No.6 workings however were much lower than was estimated from the radiation results and ranged from 2/3 to 1/5 of the estimated U_3O_8 content. If it were assumed that the excess radiation was due to radium which has decayed to 20% of its original mass, then it would seem that these samples may have contained originally up to 0.4% U_3O_8 and averaged about 0.24% U_3O_8. These figures would be lower if the secondary uranium minerals were younger than has been assumed, for in this case the residual radium would not have decayed to as low as 20% of its original mass.

The absorption tests described in an earlier section of this report suggest a means of distinguishing between the radiation from radium C and Uranium X_2 and hence determining separately the radium and uranium content of samples. The tests, however, were not applied to the bulk of Mount Painter samples because they were only carried out at a late stage of the investigation and the earlier results achieved the object for which they were designed, namely of eliminating samples too low in uranium to warrant chemical assay.

The results described herein illustrate the fundamental difficulty of assessing uranium content of secondary uranium ore such as that from the Mount Painter No.6 deposit.

The U_3O_8 contents of the ten samples from the No.6 deposit assayed chemically were all lower than those estimated by radiation methods and it is believed that this condition would hold generally for the remaining 90 samples from the No.6 deposit. As the estimates for all samples rejected were below 0.08% U_3O_8 it follows that no samples with an actual U_3O_8 content in excess of this were rejected. On the other hand the average U_3O_8 content of the eight samples from East Painter No.1 Adit assayed chemically was approximately 10 per cent greater than estimated and two of the samples rejected, viz. 5 and 11, may have had U_3O_8 contents as high as 0.1 per cent.

VII. GENERAL CONCLUSIONS.

It is believed that the field work described herein has shown that the Geiger-Muller radiation counter has a useful though limited application in prospecting occurrences of uranium minerals. Its chief drawback is of course its limited range insofar as the radiation measured is essentially a near-surface phenomenon.

The field surveys revealed numerous small zones of high radioactivity. A small proportion of the zones was tested by knapping and by shallow pits but without exposing any torbernite deposits of economic importance. The fact that the surveys described herein failed to disclose uranium deposits of economic value is not necessarily evidence that the methods would not have succeeded in revealing such deposits if they had been present in the areas surveyed.

Their most valuable contribution to the investigation of the Mount Painter field has been in supporting and clarifying geological premises and in giving a picture of the distribution of radioactive minerals. The radiation surveys demonstrate most clearly a coincidence of high radiation value with occurrences of hematite. The fact that a considerable proportion of the radioactive hematite has no uranium in it has been interpreted as meaning that this hematite contains radium formed from original minerals from which the uranium has been leached during the past million years.

The ratios between the quantities of radium and uranium and lead and uranium in the ore sheet in the No.6 deposit have been considered in relation to the age of the deposit. It was shown that while the average Ra/U ratio indicates an age of considerably less than a million years for the secondary uranium minerals the amount of lead recovered was far in excess of that which would have been present had the secondary minerals alone been its source.

The lead/uranium ratio of fergusonite (the only primary uranium mineral definitely known to occur in the Mount Painter district) was found by Kleeman[1] to be 0.093 corresponding to early Palaeozoic Age.

It is believed, therefore, that the above facts are best explained by -

(i) Uranium, Radium and lead derived from a primary uranium mineral of early Palaeozoic age are present in the ore. The uranium and radium are present in amounts corresponding to equilibrium. The lead/uranium ratio would be high.

(ii) Secondary uranium minerals (torbernite etc.) were introduced into the ore in quantities relatively much greater than the primary uranium mineral. This introduction occurred approximately 400,000 years ago.

The results are that -

(iii) The average ore contains only approximately three quarters of the radium compared with that which would be in equilibrium with the uranium present.

(iv) The amount of lead compared with the uranium gives a much lower age than that of the primary mineral.

The absence of visible uranium minerals in much of the radioactive hematite and the weak reaioactivity of the crush zones in general is taken as evidence that the crush zones have been leached of their uranium content at a time less than one million years ago. The uranium leached from them would have been sufficient to form the richer concentrations of secondary minerals (torbernite etc.) such as the No.6 ore sheet.

In the early stages of the investigation the Geiger-Müller equipment proved a useful tool by determining for the geologists whether the different minerals found by them were radioactive or not.

As far as the assay work was concerned the equipment's main usefulness was in indicating those mine samples whose activities were so low that it was certain that they contained only traces or very minor amounts of uranium and consequently obviating the necessity for chemical assay work on them.

Canberra.
19/9/45.

---

[1] Mawson, Sir D. op cit.

V111. RESULTS OF ACTIVITY TESTS ON MINE SAMPLES

The numbers and positions of the samples are those supplied by the Mine Manager.

N - Nil, T - Trace, L - Less than 0.1%, C - about 0.1%, M - More than 0.1%.

Sample No.	From.	Reduced Reading.	Estimated % U_3O_8	Class.	Recommended for assay.	Assay result.
MOUNT PAINTER NO.6 WORKINGS.						
100 ft. Drive.						
1		.01	0	N	No	
2		-.02	0	N	"	
3		-.03	0	N	"	
4		0	0	N	"	
5		-.04	0	N	"	
6		+.02	0	N	"	
7		-.02	0	N	"	
8		+.02	0	N	"	
9		-.04	0	N	"	
10		+.05	0	N	"	
11		.15	.02	L	"	
12		.02	0	N	"	
13		-.01	0	N	"	
14		.01	0	N	"	
15		.05	.01	T	"	
16		-.04	0	N	"	
17		0	0	N	"	
18		.10	.01	T	"	
19		.19	.02	L	"	
20		.37	.04	L	"	
21		.28	.03	L	"	
22		.24	.03	L	"	
1A		.29	.03	L	"	
2A		.30	.03	L	"	
3A		.28	.03	L	"	
100 ft. Cross-Cut						
1 x	0 to 3 ft.	.12	.01	T	No.	
2 x	3 to 6 ft.	.30	.03	L	"	
3 x	6 to 9 ft.	.16	.02	L	"	
4 x	9 to 12 ft.	.06	.01	T	"	
5 x	12 to 15 ft.	.03	.0	N	"	
6 x	15 to 18 ft.	.06	.01	T	"	
7 x	18 to 21 ft.	-.03	0	N	"	
1A	22½ to 25 ft.	.05	0	N	"	
2A	25 to 28 ft.	.04	0	N	"	
3A	28 to 31 ft.	-.03	0	N	"	

Sample No.	From.	Reduced Reading.	Estimated % U_3O_8	Class	Recommended for assay.	Assay result.
50 ft. Cross-Cut.						
1	0 to 3 ft.	.08	.01	T	No.	
2	3 to 6 ft.	.08	.01	T	"	
3	6 to 9 ft.	.09	.01	T	"	
4	9 to 12 ft.	.[illegible]	.02	L	"	
5	12 to 15 ft.	.01	0	N	"	
6	15 to 18 ft.	.06	.01	T	"	
7	18 to 21 ft.	.18	.02	L	"	
8	21 to 24 ft.	.07	.01	T	"	
9	24 to 27 ft.	.05	0	N	"	
10	27 to 30 ft.	.17	.02	L	"	
1A	36 to 39 ft.	.12	.01	T	"	
2A	39 to 42 ft.	0	0	N	"	
3A	42 to 45 ft.	.12	.01	T	"	
4A	45 to 48 ft.	.15	.02	L	"	
5A	48 to 51 ft.	.07	.01	T	"	
6A	51 to 53 ft.	-.03	0	N	"	
80 ft. Sub-Level.						
1	8 ft.	.49	.05	L	No	
2	11 ft.	.47	.05	L	"	
3	14 ft.	.41	.05	L	"	
4	17 ft.	.41	.05	L	"	
5	20 ft.	.55	.06	L	"	
6	23 ft.	.37	.04	L	"	
7	26 ft.	.54	.06	L	"	
8	29 ft.	.51	.06	L	"	
9	32 ft.	.82	.09	C	Yes	.02
10	35 ft.	.25	.03	L	No	
11	38 ft.	.65	.07	L	"	
12	41 ft.	.24	.03	L	"	
13	44 ft.	.84	.09	C	Yes	.02
14	47 ft.	.72	.08	C	"	.03
15	50 ft.	1.00	.11	C	"	.02
16	53 ft.	1.38	.15	M	"	.07
17	56 ft.	.70	.08	C	"	.02
18	59 ft.	.83	.09	C	"	.06
100 ft. Winze.						
1	18 ft.	.02	0	N	No	
2	21 ft.	.02	0	N	"	
3	24 ft.	.08	0.01	T	"	
4	27 ft.	.01	0	N	"	
5	30 ft.	-.01	0	N	"	
6	33 ft.	+.02	0	N	"	
7	36 ft.	.02	0	N	"	
8	39 ft.	0	0	N	"	
9	42 ft.	-.04	0	N	"	
10	45 ft.	.04	0	N	"	
11	48 ft.	.03	0	N	"	
12	51 ft.	.05	0	N	"	

Sample No.	From	Reduced Reading	Estimated % U_3O_8	Class	Recommended for assay	Assay result
150 ft. Sub-level.						
1		.03	0	N	No	
2		-.01	0	N	"	
South Portal Drive.						
1		.71	.08	C	Yes	.02
2		1.10	.12	C	"	.07
3		.51	.06	L	No	
4		.73	.08	C	Yes	.04
5		.31	.03	L	No	
6		.37	.04	L	"	
7		.50	.06	L	"	
8		.24	.03	L	"	
9		.21	.02	L	"	
10		.28	.03	L	"	
11		.02	0	N	"	
12		0	0	N	"	
13		.03	0	N	"	
14		-.03	0	N	"	
15		-.03	0	N	"	
16		-.02	0	N	"	
17		-.01	0	N	"	

EAST PAINTER.

No.1 Adit (Bulk Samples)

1	12 to 15½ ft.	.82	.09	C	Yes	.11
2	15½ " 18½ ft.	.81	.09	C	"	.08
3	18½ " 20½ ft.	.78	.09	C	"	.12
4	20½ " 23 ft.	.48	.05	L	No	
5	23 " 25 ft.	.59	.07	L	"	
6	25 " 28½ ft.	1.25	.14	M	Yes	.17
7	28½ " 32¾ ft.	1.35	.15	M	"	.17
8	32¾ " 36 ft.	2.28	.25	M	"	.22
9	36 " 39 ft.	0.99	.11	C	"	.13
10	39 " 42 ft.	1.10	.12	C	"	.15
11	42 " 45 ft.	0.60	.07	L	No	
12	45 " 50 ft.	.38	.04	L	"	

1X. REFERENCES.

1905	BORNE,	VON DEM	Habilitationsschrift (Beslau).
1927	BEHOUNEK,	F.	Phys. Teit, 28, 333-342
1928	AMBRONN,	Dr. R.	Elements of Geophysics McGraw-Hill Book Co. Inc.
1932	ELLSWORTH,	H. V.	"Rare-element Minerals of Canada", E Department of Mines, Ottawa, Canada, Economic Geology Series. No.11.
1938	HEVESY.	G. and	PANETH F. A. A Manual of Radioactivity 2nd Edition Oxford University Press.
1940	HEILAND,	Dr. C.A.	"Geophysical Exploration" Prentice Hall Inc. N.Y.
1940	TAFT. R.B.		Two Small Portable Geiger-Muller Counters. Rev. Sci. Inst. Vo. 11. (2)
1941	ROSE.	R.B.	Radioactive Exploration The Mines Magazine, Dec. 1941.
1941			Handbook of Chemistry and Physics 25th Ed. Chem. Rubber. Pub. Coy
1941	LORD,	C.S.	"Mineral Industries of the Northwest Territories". Department of Mines, Ottawa, Canada. Geol. Survey Memoir 230 (1941).
1942	THOMAS,	R.G.	"Processing of Radium Ores in South Australia". Aust. Chem. Inst. J. Proc. Vol. 9 (6)
1944	MAWSON,	SIR D.	"The Nature and Occurrences of Uraniferous Mineral Deposits." Royal. Soc. S.A. Vol. 68 (2) 1944.
1944	BEERS,	R.F. and	GOODMAN C. "Distribution of Radioactivity in Ancient Sedments." Bulletin Geol. Soc. Amer. Vol. 55(10).
1945	STILLWELL	DR. F.L.	and EDWARDS DR. A.B. "Uranium Minerals" from Mount Painter". This Bulletin.
1945	TRIPP,	R.M.	"Measurment of Soil-air ions over the Fort Collins Anticline". Geophysics Vol 10, (2) April 1945.
1945	SULLIVAN.	C.J.	"Smiler Greenwood" Deposit. This Bulletin.
1945	SULLIVAN	C.J.	"Bentley Greenwood" Deposit This Bulletin.
1945	BROADHURST,	E.	"Mount Painter No.6. Workings." This Bulletin.

PART V - ORE TREATMENT REPORTS.

1. URANIUM ORES FROM MOUNT PAINTER (First Report).

BY

Professor H. W. Gartvell and D. R. Blaskett.
Joint Investigation of Council for Scientific
and Industrial Research

and the

South Australian School of Mines and Industries.
Bonython Laboratory. Adelaide.

-----------oOo-----------

1. SAMPLES:-

At the beginning of this investigation there were several samples of Mt. Painter uranium ores in this laboratory which had been used for experimental work about fifteen years ago, and many of the tests described in this report were made on these old samples.

In addition to these samples this laboratory has over the past few months prepared for analysis several hundred samples from the current exploratory work and from old dumps. Some of this material has also been used for testing.

2. INFORMATION REQUIRED:-

We were requested to examine the samples with a view to devising means of concentrating the uranium minerals.

3. DESCRIPTION OF SAMPLES:-

A lengthy description of the uranium minerals and associated gangue in Mt. Painter ores has been given by Dr. Stillwell in Mineragraphic Investigations Report No.315, but in order to get a clear idea of the concentrating problem a few brief extracts should be sufficient.

"The uranium minerals occur in an area of granitic rocks, frequently in brecciated zones, which often coincide with the outcrops of small areas of ironstone. The granite approaches a pegmatite in character, and consists essentially of quartz and reddish microcline. Disseminated ferromagnesian minerals are typically absent, but bands and schlieren of biotite rock, or biotite-garnet rock, presumably related to the granite occur occasionally.

The uranium minerals so far discovered in the various prospects and lodes are as follows:- autunite, torbernite, meta-torbernite, uranophane, and an unindentified uranium silicate, dewindtite and related lead uranium phosphates and fergusonite. Associated with them are various primary minerals such as hematite, monazite, pyrite and fluorite, and various alteration products and secondary minerals such as psilomelane, goethite, collophane, iron phosphates, limonite and gypsum.

A. Autunite $CaO.\ 2UO_3.P_2O_5.10h_2O$

Autunite was a prominent constituent of the rich shoot formerly extracted from No.6 workings, but has not been found as yet in the material from the other prospects of the district except for small amounts in vughs at the No.2 workings. The autunite occurred in small seams and vughs, intimately associated with colloform manganese and limonitic minerals, as the cement of a breccia. The seams frequently consisted of massive lemon-yellow to sulphur-yellow plates up to 1 cm. across.

Autunite has refractive indices between 1.550 and 1.575. It is uniaxial or very nearly so, and optically negative. In ultra-violet light it fluoresces a vivid emerald green.

B, Torbernite ($CuO.2UO_3.P_2O_5.8H_2O$) and Meta-Torbernite ($CuO,2UO_3,P_2O_5,8H_2O$).

Torbernite or its dehydration product, meta-torbernite is the most prevalent of the uranium minerals in the Mt. Painter prospects being present in all of them. Commonly, it is the most abundant mineral showing, but in the No.2 workings it is present only as occasional flakes.

The two forms of torbernite are generally found in close association in hand specimens. Neither of them normally fluoresce, so that they can be distinguished only under the microscope. The torbernite has refractive indices between 1.58 and 1.59 and is strongly pleochroic from green to colourless. It has a good cleavage parallel to the platy faces of the crystals, and cleavage flakes appear almost isotropic, although they give a uniaxial figure of negative character. Flakes at right angles to the cleavage show blue polarization colours.

The meta-torbernite has a higher refractive index, between 1.62 and 1.63 and cleavage fragments are practically isotropic. It also has a higher specific gravity. The meta-torbernite shows a weak uniaxial figure on cleavage plates and is optically positive. Polarization colours of fragments at right angles to the cleavage are a distinctive purple, a little bluer than the colour of a gypsum plate."

4. GENERAL DISCUSSION ON METHODS OF CONCENTRATION:-

The samples which we have on hand are almost all from the No. 6 workings and the problem of concentration is essentially the separation of torbernite from a gangue consisting pricipally of hematite, quartz and microcline with smaller quantities of biotite.

At the beginning of this investigation it was stressed that water supplies are limited in the arid Mt. Painter region and that any concentration which could be done dry would be an advantage. It seems likely that if a treatment plant is to handle this ore, the final stage of uranium recovery must necessarily be a leaching process and the aim of any ore treatment should be to produce an enhanced grade of material for leaching. The amount of water required for any wet treatment process depends primarily on the percentage of water which has to be dumped out in wet tailings and this would be about the same whether a straight leaching process or a combined wet concentrating and leaching process were used. The former would however give the higher recovery of uranium in the ore and hence, considering only recovery and water consumption, would appear preferable. A dry process which would reject say 70% - 80% of the gangue and make a good recovery of the uranium mineral would be a big aid to the treatment on account of the greatly decreased amount of water required. Nevertheless wet concentration processes were by no means ruled out since it was believed that if necessary available water could be increased by further boring operations along the creek bed.

Numerous methods of concentration have been tested in this laboratory both in the current testing programme and also in the series of tests made here about fifteen years ago and brief mention will be made here of the various methods tried.

A. Wet Processes:-

(a) Jigging, tabling and classification all depend on the density and shape of the particles and such methods have not achieved much success, primarily because of the varying nature of the gangue. The specific gravities of the chief constituents of the material are as follows:-

Mineral	Specific Gravity
Torbernite	3.2 - 3.7
Hematite	4.9 - 5.3
Quartz	2.65
Microcline	2.55
Biotite	2.7 - 3.1

It will be noted that the specific gravity of torbernite, the chief uranium mineral, is between that of the silicate gangue and the hematite. Any concentration process which relied on differences in specific gravity would therefore involve two major separations, the separation of the light silica and silicate gangue minerals from torbernite and hematite, followed by the separation of hematite from torbernite.

These separations might be possible on closely sized material if the specific gravities of the various minerals conformed to the above figures, but from their behaviour, in testing it appears that much wider variations occur, probably due to porosity resulting from weathering, and this variation together with the variation in particle shape makes the separations impracticable. The fact that torbernite is commonly tabular increases the difficulty of separating it from the lighter gangue minerals.

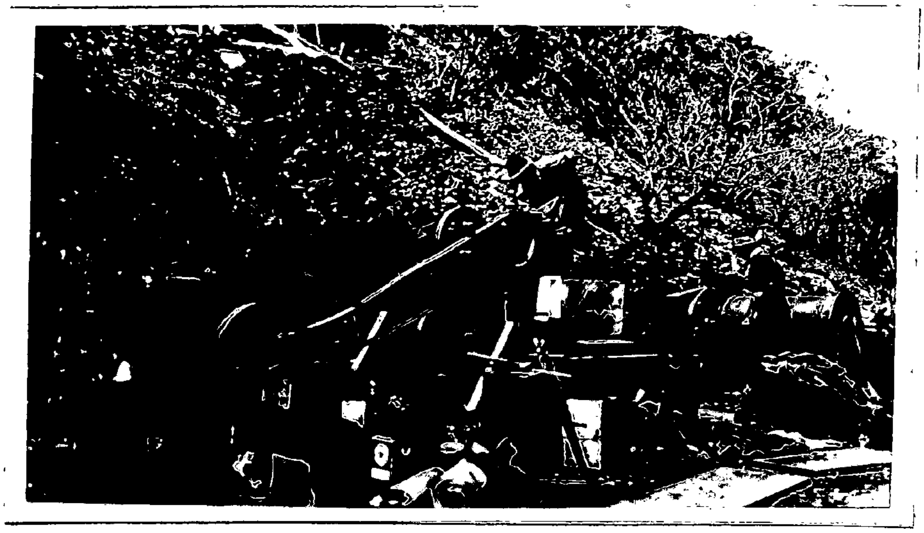
(b) Under wet processes some mention should be made of flotation as a means of separating the torbernite from the gangue minerals. In the early investigations referred to above, flotation tests gave little encouragement but since then, new reagents and increased knowledge of flotation principles have increased the range of minerals which can be floated selectively and some further flotation tests were made on floating the uranium minerals.

Preliminary tests on high grade uranium mineral showed that it could be floated with several different reagents, but later tests showed that control of the gangue would present the greater problem. Flotation from a clean siliceous gangue could probably be developed so as to give a good separation but even small quantities of hematite gave bad sliming and prevented effective selective flotation of the torbernite. The iron oxide slime evidently absorbed much of the reagent since large amounts of reagents were necessary to float anything.

B. Dry Processes:-

(a) Concentration by differential grinding. Much of the torbernite in the samples so far examined occurs as flakes which can be exposed by crushing and grinding to about 4 mesh. Rumbling at this size or a light attrition grind with small iron slugs will separate the thin torbernite flakes from the gangue and subsequent screening will concentrate the torbenite in the finer flakes. Such a process could be used as a means of preliminary concentration, but neither a high recovery nor grade of concentrate could be obtained by this process. A crude process of this nature was used in preparing ore for shipment years ago.

FIG. 1. EARLY TREATMENT PLANT. MT. PAINTER (NOV. 1926).



(b) Magnetic concentration is a means of concentrating certain types of Mt. Painter ore, but the varying nature of the gangue would not permit its use alone as a concentrator. On ores in which the gangue is predominantly hematite a large percentage of gangue could be rejected as a magnetic concentrate, but on some ores in which the gangue is principally siliceous the magnetic concentration would affect almost no enrichment.

(c) Sticky-surface concentration was suggested as a means of separation when it was noted how the flakes of torbernite would adhere to any slightly greasy smooth surface. The principle of separating minerals by their power to adhere to a greased surface has of course been known for centuries although its application in the past has been almost solely to diamond concentration. In recent years similar processes have been tested experimentally for other separations (see for example, "Sticky Surface Concentration of Gravel Sized Minerals", Mining Technology, November 1942). In these processes, the minerals to be separated suspended in a pulp, are brought into contact with a suitably prepared surface to which certain constituents of the mixture will adhere if the surface of the mineral is properly conditioned. The process therefore depends fundamentally on the same principles or the oil agglomeration process.

The process we have been testing in this laboratory differs somewhat from these earlier processes, but for want of a better name we have called sticky surface concentration. In this work the ore, after crushing to about 20 mesh, is fed dry onto a glass surface coated with a thin film of oil to which the torbernite sticks more readily than the gangue. From observations of the experiments it is believed that the chief factor operating to cause selection of the torbernite is the flaky nature of the torbernite particles which allows closer contact between the particle surface and the glass than does the more irregular surface of the gangue.

This process relying on one of the outstanding physical characteristics of the torbernite has showed considerable promise as a means of separating torbernite from the varying types of Mt. Painter ores and much time has been spent in trying to develop a laboratory machine for more extensive testing. More details of the development of our laboratory machine and results achieved by it will be given later but it has now reached the stage where we can state that the machine seems capable of making a good recovery of the torbernite while enriching the grade several times.

5. DETAILS OF TESTS:-

Because of its vivid colour, torbernite if present in appreciable quantities (say above 0.2%) is easily distinguished in samples and much of the preliminary testing of these ores relied on visual examination of the sample to indicate the value of the test.

A. Gravity concentration tests:-

Jigging and tabling tests on both sized and classified samples gave little hope of a reasonable separation. On samples containing massive hematite up to 40 per cent of the gangue might be rejected as a low grade tailing by tabling, but samples containing principally a siliceous gangue are not amenable to such concentration. The somewhat higher gravity and the flaky nature of the torbernite as compared with quartz and silicate minerals, gives almost no selection on tabling. Jigging tests showed little encouragement because of similar causes -- much of the apparent hematite even seems to have a porous structure giving particles a lower density so that part only of the hematite can be separated by gravity processes.

B. Flotation:-

Some preliminary tests were made on high grade uranium mineral which had been separated by hand picking from specimen ore. Torbernite is a hydrated copper uranyl phosphate and possible means of floating such a compound are:-

1. Flotation with a xanthate and sodium sulphide.
2. Flotation with a mercaptan type reagent.
3. Flotation with a fatty acid reagent.
4. Flotation with more recently developed cationic reagents.

The first two schemes do not appear favourable since uranium has not much affinity for sulphur, forming only a rather unstable uranyl sulphide even under properly controlled conditions. There is a possibility that the copper present in the mineral might be sufficiently reactive to form a copper sulphide coating on the surface but laboratory tests using amyl xanthate as collector after conditioning with sodium sulphide did not offer such hope of success.

Oleic acid alone did not appear to be sufficient to float the uranium mineral but the addition of a little fuel oil brought about almost complete flotation of the torbernite. In the presence of gangue, however, this collector combination had almost no selective action and the gangue was floated as readily as the uranium mineral. The presence of a small amount of slimed iron oxides completely prevented flotation, presumably because of absorption of the collector by the slime. Of the other reagent tried, flotation with trimethyl cetyl ammonium bromide and with Dupont Q seemed the most encouraging. Each of these reagents floated the torbernite over a wide pH range but selection from siliceous gangue was best in acid pulp of ph 3 - 4. Unfortunately as in flotation with oleic acid a small quantity of iron oxide slime interfered with flotation and large quantities of reagent were necessary to float anything. Desliming prior to flotation is rather ineffective as attrition in the flotation cell continually makes further slime.

Summarizing, although flotation could probably be developed to separate torbernite reasonably well from a siliceous gangue, the varying nature of the ore and the frequent occurrence of hematite gangue together with the necessity for keeping down water consumption make such a treatment scheme unattractive.

C. Differential Grinding:-

Although no tests have been made as yet, in this investigation on this process since we have been working on samples sent for assaying and have not received any sample which could reasonably be expected to approximate run-of-mine ore, it is certain from the early work to which reference has been made that some such scheme would be a valuable adjunct to a concentrating mill. To illustrate the results which might be expected from such a process the results of an early test are reproduced here:-

"Sample. 0.60% U308

The ore received in lumps up to about 5 inches was crushed in jaw crusher and rolls to about 5 per cent plus ¼ inch and then given a light grind in a rod mill, followed by screening. The distribution of total weight and of uranium in the screened fractions is shown in the following table:

Fraction	% by Weight	% U_3O_8	Distribution		
			Uranium %	Cumulative % Uranium	Cumulative % Weight
Plus 30 mesh	62.1	.08	8.1	8.1	62.1
30/80 mesh	13.5	.47	10.6	18.7	75.6
80/180 mesh	7.0	1.32	15.2	33.9	82.6
Minus 180 mesh sand	3.7	2.15	13.1	47.0	86.3
Minus 180 mesh slime	13.7	2.32	53.0	100.0	100.0

This shows that 60 per cent of the ore could be rejected at loss of 8 per cent of the total uranium."

It is believed that the above test does not show the matter in the best light since in a batch process the uranium mineral is overground, the bulk of it appearing in the finest products. For the convenience of further processing it would be desirable to keep the mineral as coarse as possible and the chances of doing so would be much better in continuous mill or rumbler where the finer material, say minus 30 mesh would be continuously discharged.

D. Magnetic Concentration:-

As mentioned earlier in this report magnetic concentration would only be of value as a concentrating process on ores in which the gangue was predominantly oxides of iron. Various types of ore have been tested on the "Rapid Magnetting Machine" in this laboratory and the weight of the magnetic portion made has ranged from 60% on a massive hematite ore down to ½% on a granite ore. If we assume that under the most favourable conditions, 60% of the weight could be rejected without loss of uranium, then the grade of enriched material would be only 2½ times that of the feed. However from examination of numerous samples received from the field and from observation of ore in situ, it is believed that the average amount of magnetic material which could be rejected would be of the order of 10-15 per cent of the weight and assuming again that no uranium were lost, the enriched material would then be not more than 1.2 times the grade of feed. It seems reasonable therefore to assume that magnetic concentration could not assume any great part in concentrating the general run of material.

E. Sticky-surface concentration:-

Before describing the machine which we are at present using for testing of this process a brief mention will be made of the development of this machine.

Our first tests were made on an inclined glass plate which was sprinkled with a few drops of reagent and excess removed by rubbing with a pad of cotton waste. The sample was then allowed to rill down the inclined plate which picked up a coating of material. This plate was used to test various substances as sticking medium reliance on visual observation being used to assess their relative merits. Amongst the substances tried were, paraffin wax, vaseline, lubricating oil, kerosene, glycerine, cresylic acid, crude creosote, castor oil, oleic acid, pine oil, linseed oil and eucalyptus. Of these lubricating oil and kerosene appeared to be as satisfactory as any and all subsequent work has been done using a mixture of kerosene and lubricating oil in about equal amounts. Any slight advantages of one substance over another would have to be worked out on a larger scale machine over long periods of testing and for preliminary tests it was felt that the results achieved with this mixture would be indicative of results which could be achieved in larger scale testing.

Following on these preliminary tests with a glass plate a few tests were made using the outside surface of a small glass cylinder rotated at about 30 r.p.m. about an inclined axis. The surface of this cylinder was periodically sprayed with an oil-kerosene mixture from a scent-spray type atomiser. The feed was dropped on the upper side of the cylinder and the adhering material was scraped off on the underneath side by a piece of rubber set in wood while the tail dropped directly over the side. With this simple apparatus it was evident that such a coated surface did have a very definite selective action in picking up the uranium mineral although the very short time of contact did not allow much chance of picking out all the mineral. Repeated treatment however appeared to give reasonably clean tails and it soon became evident that the type of machine that was necessary was one which gave a much better chance for the uranium mineral to come into contact with the surface. Of the various arrangements which were suggested the best idea seemed to be to make use of the inside surface of a rotating cylinder set on a small angle so that material fed in at the high end would gradually work down to the lower end. The machine which was then built has gone through various stages of alteration but these have been concerned mainly with distributing the sample over the surface and inducing it to rill instead of sliding, and with the means of maintaining a uniform oil coating and of removing the adhering particles. The machine as in use at present is shown diagrammatically in the attached isometric sketch.

A is an open-ended glass cylinder 11½ inches inside diameter and 18 inches long supported and rotated by rubber covered rollers B. C is the oil tank and atomisers for spraying oil onto the inside surface of the cylinder. Three air jets with orifices about 1/40 in diameter are mounted directly opposite capillary tubes of 0.02 inches bore which dip into the oil mixture in the tank. Air is fed to the jets through rubber tubing with clamps to control the supply of air to each. The air jets are spaced about 5 inches apart. Air is supplied by a small portable compressor to a tank on which a spring loaded valve is set to blow off at 15 lb. The pressure of air to the sprays can then be set to the desired figure and will remain steady throughout the experiment.

D is an agitator consisting of a wooden roller with four radial pieces of sponge rubber, the whole being rotated by a belt drive from one of the main rollers. This helps to turn over the sample and increases the chance of a flake of torbernite being picked up by the surface.

E is a sponge rubber wiper, hinged to an arm at the centre and held against the glass surface by a tension spring. F. is a trough to catch the material dropping from the rubber wiper. For the sake of simplicity this is shown as a plain trough but actually is divided longitudinally into three hoppers with chutes leading back toward the feed and so that the first portion picked up could be taken out as a concentrate and the remaining parts if so desired could be fed back as a middling product for treatment. G is a small feed bin fitted with a conical bottom and sliding feed gate with various size openings to control the rate of feed. H is a feed chute which carries the feed from the bin into the cylinder. The main driving rollers, and hence the cylinder itself are tilted at a slope of about 1 in 5. The rollers are driven through a motor and worm reduction drive so that speed of rotation of the cylinder is about 8½ r.p.m. These factors can be varied perhaps it may be found advantageously, but most of the work so far has been carried out under these conditions. To avoid confusion in the sketch no supports are shown but the oil spray tank, the rubber wiper and concentrate trough are all attached to rods held to rigid upright by screw clamps.

(i) Sizing of material for testing:-

From early tests it seemed evident that some system of sizing would be essential before any reasonable separation could be made. It is obvious that 325 mesh particle would adhere more readily than a 10 mesh flake of torbernite so that all samples tested have been sized although as yet no attempt has been made to decide the optimum closeness of sizing. Tests have been made with material as coarse as 10 mesh, but it appears that 20 mesh or finer gives better results. At the coarser sizing a heavier oil coating is required and the torbernite particles are more likely to occur in granular form.

Fine dust if present in the samples will adhere to the surface and prevent the torbernite from sticking so that dedusting either by air classification or by screening is necessary, the former being more easily applied as fine screens are rapidly blinded. So far in our experimental work no attempt has been made to treat completely a sample of ore, our tests being confined to developing the machine to handle material in the size range down to 100 mesh.

~~(2)~~ (ii) Testing Procedure and Results.

The oil consumption on the machine could only be determined accurately over long periods of operation but rough tests have indicated that it is of the order of one to two lb./ton when treating material about 40 mesh. The rate of oil addition during tests is controlled by putting a mercury manometer in the air line and setting the valve to give a definite pressure; usually three inches of mercury has been used. The rate of feed has been varied to suit different sized feed but has usually been in the range from 10 to 20 lb./hr.

A. A sample containing torbernite associated with pegmatitic gangue was crushed through rolls to minus 30 mesh and screened on 60 mesh and 100 mesh. The fractions 30/60 and 60/100 were tested on the machine, but at this time only one oil jet was fitted so that the length of concentrating surface was insufficient to obtain a high recovery in a single pass.

30/60 mesh treated by a single pass through machine.

Head Assay 0.96% U_3O_8

	Weight %	Assay % U308	% Distribution of Uranium
Concentrate	27.2	1.21	39
Tailing	72.8	0.70	61
Calculated Head ..	100.0	0.84	100

60/100 mesh treated by a single pass through the machine.

Head Assay 1.20% U_3O_8

Concentrate	12.5	1.56	19
Tailing	87.5	0.97	81
Calculated Head ..	100.0	1.05	100

A portion of the same minus 30 mesh sa ple was dedusted in the laboratory Federal Dust Classifier rejecting 15 percent of weight of sample as fine dust.

The coarse fraction was then treated under conditions similar to those of the tests above giving:-

Concentrate	10.5	1.83	23
Tailing	89.5	0.72	87
Calculated Head ...	100.0	0.83	100

Although these tests showed that the machine had a selective action for the torbernite, the recoveries were low primarily because of the short length of the concentrating surface.

The three oil jets as shown on the diagram were then made so that later tests made use of a much longer concentrating surface.

B. A sample of 20/40 mesh ore with a pegmatitic gangue was treated in a single pass through the machine with three oil jets operating.

Concentrate	51.4	2.10	90
Tailing	48.6	0.25	10
Calculated Head ..	100.0	1.20	100

The tail from this test was retreated over the machine but the final tail still assayed 0.24% U308, a difference of .01% which is not significant.

This test shows the very much improved recovery possible with a larger concentrating surface.

C. A sample of 20/40 mesh ore with hematite gangue was treated twice over the machine with three oil jets operating.

Concentrate	59.5	2.24	95
Tailing	40.5	0.17	5
Calculated Head ...	100.0	1.40	100

A sample of 40/60 mesh ore with hematite gangue was treated under conditions similar to the above giving:-

Concentrate	45.4	2.76	86
Tailing	54.6	0.38	14
Calculated Head ...	100.0	1.46	100

A sample similar to the above was treated with a lower pressure on the oil jets in an attempt to make a higher grade of concentrate in a single pass through the machine.

Concentrate	34.8	3.36	80
Tailing	65.2	0.45	20
Calculated Head ...	100.0	1.46	100

It will be noted that a higher grade of concentrate was obtained at the expense of some loss in recovery.

On the whole these tests indicate that a reasonably good recovery can be made while rejecting from 50 to 60 per cent of the gangue.

D. To confirm the above results some further tests were made on a lower grade sample in which the gangue was principally hematite.

A sample of the ore was crushed through rolls to pass 20 mesh and then screened on 40 and 60 mesh. The two fractions 20/40 and 40/60 were used for testing. In each test a rougher concentrate and a tailing were made in a single pass through the machine with an air pressure of 3 inches of mercury in the sprays. The rougher concentrates were then cleaned by two further passes through the machine, adding the first cleaner tail to the rougher tail and keeping the second cleaner tail as a middling product, which in continuous operations would be fed back with new feed. These tests were made on samples of about 3000 and 4000 grams respectively. As a check on the work done, a portion of each tailing was given a retreatment and the retreated tails sampled for assay.

The results of these two tests are tabulated below:

Sample	Product	Weight %	Assay % U308	% Distribution of Uranium	Rate of feed Gms./min.	Oil used lb./ton
	Concentrate	9.2	2.94	63		
	Middling	16.5	.35	13		
20/40 mesh	Tailing	74.3	.14	24	180	1
	Calculated head	100.0	.43	100		
	Retreated tailings		.13			
	Concentrate	4.4	2.22	43		
	Middling	4.6	.52	10		
40/60 mesh	Tailing	91.0	.12	47	130	1.4
	Calculated Head	100.0	.23	100		
	Retreated Tailing		.11			

If we assume that when retreated the uranium in the middling would be distributed between concentrate and tailing in the same ratio as that from new feed, the overall recoveries would be 72% and 48% respectively in concentrates weighing respectively 11% and 4½% of the original weight of feed.

Considering the low grade of material which was treated in these two tests, the results make the process attractive as a means of enriching low grade ore prior to a leaching treatment.

6. SUMMARY:-

1. The problem of concentrating uranium minerals from the Mt. Painter ores is primarily one of separating torbernite, the principal uranium mineral, from the associated gangue, but the properties of the torbernite and the variable nature of the gangue makes the commoner ore dressing processes of little value in effecting this separation.

2. It is believed that the ultimate stage of treatment for uranium recovery must be a leaching process and that the role of any ore dressing process should be to prepare an enriched grade of material for leaching i.e. to reject a large portion of the gangue without discarding much of the uranium content. In view of the high value of uranium any seam of specimen ore should be hand picked. It is also likely that a rumbling process as set out on page 5 would be of value.

3. Although water supply is likely to be a serious problem in the Mt. Painter region, both wet and dry processes of concentration were tried including gravity concentration, flotation and magnetic concentration but with the exception of the last these processes gave little hope of success.

4. Magnetic concentration could achieve a useful enrichment on ore in which the principal gangue is ironstone, but unfortunately it appears that much of the ore is likely to be associated with a granitic type of gangue and on this type of ore, magnetic concentration would not be of any value.

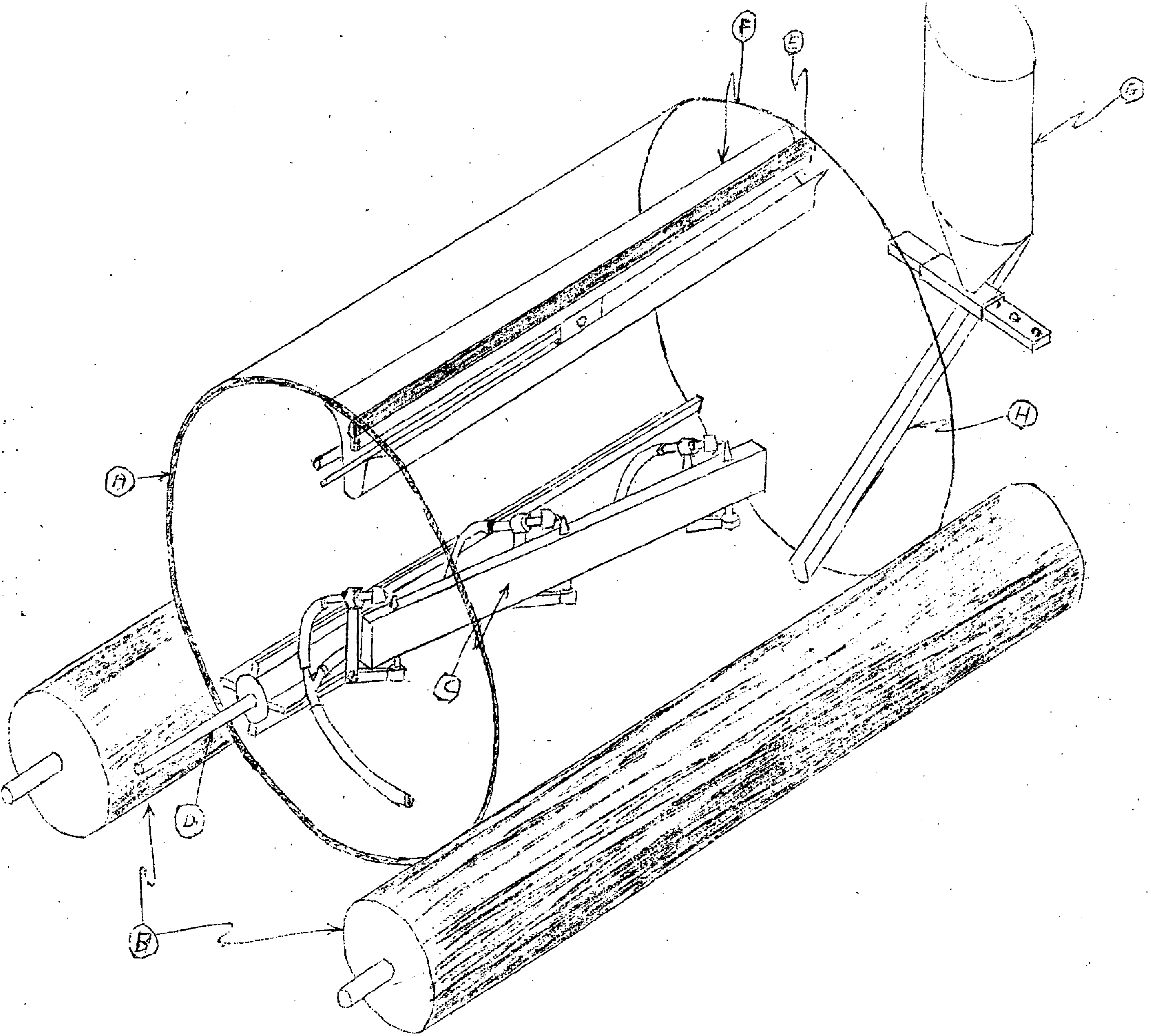
5. A variation of a sticky-surface concentration process has been tried and a laboratory machine making use of this principle has given some encouraging results. On a sample containing about 1½% uranium oxide a single treatment doubled the grade with a recovery of 80-90%, while on a low grade sample containing only 0.4% uranium oxide, three stages of treatment gave a concentrate containing 2.9% uranium oxide with an indicated overall recovery of 65-70%. The oil used in these tests would amount to between 1 and 1½ lb./ton of material treated. It is believed that such results achieved by a dry process warrant some consideration for the treatment of Mt. Painter ores.

6. The results mentioned above do not make allowance for the fines which were separated out before treatment. In crushing to say minus 30 mesh some very fine dust is made but the amount of such dust depends largely on the crushers used and the type of ore being treated. No reasonable assessment of this factor can be made at present since we have little idea of what a representative sample of run-of-mine ore will be like. It is possible that if the fine dust carries an appreciable part of the total uranium content, it may be necessary to leach it without further enrichment.

The tests with the laboratory machine are as yet far from complete and further work will be needed to check the effects of the numerous variables likely to influence the results e.g. rate of rotation, rate of feed, slope of the machine, thickness of the oil coating and sizing and type of the material treated.

As yet little consideration has been given to the problem of translating these principles to a plant-scale machine, but if further tests on the present machine confirm the results already obtained the next step will be to design and make a larger laboratory machine.

(9th December, 1944.)



Isometric Sketch showing General Arrangement of Laboratory Sticky Surface Concentrator.

11. URANIUM ORES FROM MOUNT PAINTER (Second Report)

by

Professor H.W. Gartrell and D.R. Blaskett

Joint Investigation of Council for
Scientific and Industrial Research

and the

South Australian School of Mines and Industries

Bonython Laboratory, Adelaide.

1. GENERAL

This progress report describes some recent tests on differential grinding of a sample of ore from No.5 Lode Mt. Painter East, and a complete concentration test on a sample from the 50 ft. level winze No.6 workings, Mt. Painter.

2. DIFFERENTIAL GRINDING

In our first report on Uranium ores from Mt. Painter, it was stated that this process would be a very useful part of any concentrating scheme, but at that time, no tests of this nature had been made on current samples.

An investigation of the optimum conditions for differential grinding is now in hand and the first four tests of this series are reported below.

A. Details of Tests:

The ore used for these tests was a sample from No.5 Lode Mt. Painter East, assaying 0.28% U_3O_8. This was crushed through jaw crusher, and rolls to approximately minus $\frac{3}{8}$ inch, and then screened on 20 mesh. The percentage by weight and the assay of the two fractions are set out in table 1.

Table 1

Fraction	Weight %	U_3O_8 %	Percent Distribution of U_3O_8
Plus 20 mesh	73.6	0.14	37
Minus 20 mesh	26.4	0.67	63
Composite Head	100.0	0.28	100

/The

The differential grinding tests were all carried out on the plus 20 mesh fraction and a screen test of this sample is set out in table 11.

Table 11

Screen Mesh		Screen Aperture (inches)	% Weight	Cumulative % Weight
Tyler	Equivalent British Standard			
-	-	.371	0.1	0.1
3	-	.263	3.0	3.1
4	-	.185	17.3	20.4
6	5	.131	28.0	48.4
8	7	.093	20.3	68.7
10	10	.065	14.2	82.9
14	14	.046	10.3	93.2
20	18	.033	6.8	100.0

The mill used for the grinding tests was a 10" x 8" cylindrical mill and the charge of ore in each case was approximately 2500 grams. In this series of tests two variables were investigated, the size of the grinding media and the time of grinding. The grinding media used were circular punchings from mild steel plate, having a burr on one edge. Such media were found by Shelton and Engel (U.S. Bur. of Mines R.I 3636) to be very effective for the differential grinding of carnotite from Colorado and Arizona ores. The weight of punchings used was 3000 grams and the size varied in each test, details being shown in Table 111. The grinding was carried out in stages, the times of the first two stages being the same but in the later stages, the time was double that of the previous grind. This gave roughly, the same weight of fines made in each stage of grinding. The sample was screened on 20 mesh at the end of each period of grinding and the minus 20 mesh material weighed and put aside to make up samples for analysis.

The minus 20 mesh samples from each stage of grinding were not analysed individually, but were made up into three or four composite samples, and from these

/analyses,

analyses, the results set out in table 111 were calculated. This table shows the calculated analyses of the plus 20 and minus 20 mesh fractions after different periods of grinding. The figures in the last three columns of this table refer to the total weight of minus 20 mesh material i.e. the minus 20 mesh from the primary crushing, plus the minus 20 mesh from differential grinding of the primary oversize. The figures set out in the table, are all cumulative e.g. The third line of test B, means that after 40 minutes grinding of the plus 20 mesh sample with 3000 grams of 13/32" x 1/8" punchings, 86.6 percent of the sample is plus 20 mesh, analysing .065% U_3O_8 and containing 40% of the Uranium, which was in the primary plus 20. The remaining 13.4% was ground through 20 mesh and analysed .63% U_3O_8 and contained 60% of the Uranium which was in the primary plus 20. The combined minus 20 mesh material constitutes 36.2% of the weight of original sample, analysed .66% U_3O_8 and contained 85% of the Uranium which was in the original sample.

These few tests indicate that on this particular sample, containing 0.28% U_3O_8, there is not much difficulty in grinding the sample so as to concentrate over 80% of the Uranium content in about 1/3 of the original weight. Considering grade of product and recovery, Test B using slugs 13/32" x 1/8" gave slightly better results than the other tests, but taking into account time of grinding, test C with heavier slugs and faster grinding has some advantages. Further tests are in hand to indicate the behaviours under different conditions but the ease with which preliminary concentration can be achieved by differential grinding, indicates that consideration should certainly be given to the question of leaching a smaller quantity of enriched material instead of low grade ore as mined. It has been pointed out in the recent report on Leaching of Uranium Ores from the Division of Industrial Chemistry, that the leaching of fine material presents certain difficulties, such as retention of leach liquor, but in any form of crushing a certain amount of fines will be made and the fines will normally be richer than the coarse material so that provision must be made for their treatment.

It seems that a decision has to be made between leaching of say 100 lb. of 1 inch ore containing 0.28% U_3O_8 and the leaching of 35 lb. of minus 20 mesh material containing 0.65% U_3O_8. The higher grade of the fine material would give rise to more concentrated Uranium solutions, and permit more washing without getting unduly large volumes of solution.

3. STICKY-SURFACE CONCENTRATION TEST ON ORE FROM 50 FT. LEVEL WINZE. NO.6 WORKINGS MOUNT PAINTER.

Continuing the work described in our first report, complete test has been made on the above samples. The only alteration made to the machine as described in that report has been the addition of two more oil sprays, reducing the spacing of the jets to 2½ inches. This arrangement gives a more uniform coating on the surface, and reduces the sliding of the material over the surface giving more of a rilling action.

Table 111

Test	Total Time of Grind (Mins)	20 mesh oversize			20 mesh undersize			Total 20 mesh undersize			Grinding Media
		% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	
A	5	96.3	.12	81	3.7	.73	19	29.1	.68	70	Nil.
	20	93.6	.10	69	6.4	.69	31	31.1	.67	75	
	40	92.4	.095	64	7.6	.66	36	32.0	.67	76	
B	2½	94.3	.11	71	5.7	.71	29	30.6	.67	74	300 grams punchings 13/32" x ⅛"
	10	90.2	.08	51	9.8	.70	49	33.6	.68	81	
	40	86.6	.065	40	13.4	.63	60	36.2	.66	85	
	80	85.0	.06	36	15.0	.60	64	37.5	.65	87	
C	1¼	94.8	.11	76	5.2	.64	28	30.2	.67	72	3000 grams of punchings 13/16" x ¼"
	5	88.7	.08	52	11.3	.59	48	34.7	.65	81	
	20	81.1	.065	37	18.9	.47	63	40.3	.60	86	
D	2½	95.5	.115	79	4.5	.67	21	29.7	.67	71	3000 grams of punchings 5/1[illegible] x ⅛"
	10	92.0	.095	64	8.0	.64	36	32.3	.67	77	
	40	88.8	.085	53	11.2	.59	47	34.6	.65	81	

The sample of ore as received analysed 0.91% U_3O_8. After crushing to minus 12 mesh in jaw crusher and rolls, it was screened into several fractions, and the fine dust removed by treatment in the dust classifier. The screen analysis of this material is set out in Table IV. The first four fractions were treated in the concentrator and the concentrates weighed and assayed. The fine dust was not treated, but it is considered that this material would have to be leached together with the concentrates so the uranium contained in it is regarded as recoverable. The results of these tests are set out in Table IV. The overall recovery shown in these tests is 78% in a concentrate 25% of the original weight with concentrates assaying 2.8% U_3O_8. The grade of minus 100 mesh was 2.4% U_3O_8, so that this fraction might reasonably be added to the other concentrates while that of the plus 30 mesh concentrates was only 1% U_3O_8 and represented only 5% of the total uranium. Assuming then the plus 30 mesh were discarded and that only the two intermediate fractions were treated a recovery of 79% could be obtained with a concentrate grade of 2.9% U_3O_8. Actually much of the uranium from the plus 30 mesh fraction could be put into the other fractions by a differential grind and an overall recovery of 85% seems likely.

A. SUMMARY.

1. Differential grinding tests have shown that when treating an ore containing 0.28% U_3O_8, up to two-thirds of the ore can be rejected at plus 20 mesh, leaving over 80% of the total uranium in a concentrate assaying about 0.65% U_3O_8. Such a concentration should be a great advantage prior to leaching.

2. The optimum conditions for grinding are dependent to a certain extent on the tonnage to be ground. Laboratory tests so far indicate that some grinding media is desirable but too heavy media will result in less selective grinding so that a choice will have to be made between rate of grinding and selectivity of the grind. If a large tonnage is to be handled the time required for grinding will have to be chosen so as not to require too large a mill.

3. A further test using the laboratory sticky-surface concentrator on a sample containing 0.91% U_3O_8 showed on overall recovery of almost 80% of the uranium in a concentrate assaying 2.9% U_3O_8. It is proposed to continue tests on this process after preliminary concentration by differential grinding.

Date: 23rd January, 1945.

Fraction	Screen Analysis					Concentrates Produced			
	Weight %	Cum. Weight %	% U_3O_8	% Distribution of U_3O_8	Cum. % Distribution of U_3O_8	Weight %	% U_3O_8	Recovery % in fraction	Recovery % of total
Plus 30 mesh	54.8	54.8	0.21	12.6	12.6	8.8	1.01	42	5.3
30/60	17.5	72.3	0.71	13.6	26.2	17.4	2.47	61	8.3
60/100	8.4	80.7	1.39	12.9	39.1	29.1	3.72	78	10.1
Minus 100 (sand)	16.3	186.8	2.41	16.2	55.3	33.0	4.41	60	9.7
Dust	13.2	100.0	3.08	44.7	100.0				44.7
Total	100.0		0.91	100.0		25.4	2.80		78.1
Discarding the plus 30 mesh and adding the minus 100 (sand) untreated						24.7	2.92		79.3

III. DIFFERENTIAL GRINDING OF URANIUM ORES FROM MT. PAINTER (Third Report.)

by

Professor H. W. Gartrell and D. R. Blaskett

Joint Investigation of Council for Scientific and Industrial Research

and the

South Australian School of Mines and Industries
Bonython Laboratory, Adelaide.

....

1. GENERAL.

This report describes recent tests on differential grinding of a sample of ore from No. 5 lode Mt. Painter East.

2. DIFFERENTIAL GRINDING.

In our first report on Uranium Ores from Mt. Painter it was stated that this process would be a very useful part of any concentrating scheme, but at that time no tests of this nature had been made on current samples.

This report describes the results of such tests under a variety of grinding conditions.

Because of the arid conditions in the Mt. Painter area, dry grinding was used in all tests. Wet grinding and screening would undoubtedly improve the recovery of fines buy this advantage would be outweighed by the difficulties of maintaining a difficult supply of water.

a. Details of Tests.

The ore used for these tests was a sample from No. 5 lode Mt. Painter East, assaying 0.28% U308. This was crushed through the laboratory jaw crusher and rolls to approximately minus $\frac{3}{8}$ inch.

Two series of tests were then made on this sample, series A on grinding the plus 20 mesh fraction of the roll product and series B on grinding the plus 30 mesh fraction.

(1) Series A: The percentage by weight and the assay of the plus and minus 20 mesh fractions are shown in Table I, and a screen test of the plus 20 mesh fraction is shown in Table II.

/Table 1.

Table I.

Fraction	Weight %	% U_3O_8	% Distribution of U_3O_8
Plus 20 mesh	73.6	0.14	37
Minus 20 mesh	26.4	0.67	63
Composite Head	100.0	0.28	100

Table II.

SCREEN MESH		Screen Aperture (inches)	% Weight	Cumulative % Weight
Tyler	Equivalent British Standard			
--	--	.371	0.1	0.1
3	--	.263	3.0	3.1
4	--	.185	17.3	20.4
6	5	.131	28.0	48.4
8	7	.093	20.3	68.7
10	10	.065	14.2	82.9
14	14	.046	10.3	93.2
20	18	.033	6.8	100.0

/The mill

The mill used for the grinding tests was a 10" x 8" cylindrical mill with a 2,500 gram charge of ore. The variables introduced in this series of tests were the time of grinding and the size of the grinding media.

The grinding media used were punchings from mild steel plate -- such media were found by Shelton and Engel (U.S. Bur. of Mines, R.I. 3636) to be very effective for the differential grinding of carnotite from Colorado and Arizona ores. Their report mentions that after tests had been made to select a suitable medium, "$\frac{3}{8}$" slugs were choses as the most effective. These slugs were punchings from flat plate-steel screens about 3/32" thick. They were slightly dished, the convex side being smoother than the concave side."

The weight of punchings in each of these tests was 3,000 grams, while the size of the media which varied in each test, is shown in Table III. The grinding was in each case carried out in six or seven stages, increasing the time of grind as the rate of grinding decreased. After each stage, the sample was screened on 20 mesh and the fines removed. The minus 20 mesh material formed in each stage of grinding was weighed separately but the samples were combined for analysis so that only two or three uranium determinations were made for each test.

Table III shows the rate of grinding at different stages of the tests together with the analysis of the fines produced.

The figures in the third and fourth columns of this table are taken from the beginning of the test while the figures for uranium oxide content shown in the fifth column are for the intermediate stages of the tests. Where no figure is shown in this table, the sample was combined with that from the succeding stage e.g. the fifth line of test B means that after 40 minutes, 13.4% had been ground so as to pass 20 mesh. The average rate of grinding over the 40 minutes was therefore .34% per minute. The figure in the fifth column than means that the minus 20 mesh produced in the period from 10-40 minutes assayed .43% U308.

Table III.

Test	Time (minutes)	% Weight Minus 20	Average rate of Grinding % per unit time.	% U308 in minus 20	Grinding Media used
A	5	3.7	.74	.73	
	10	5.2	.52		
	20	6.4	.32	.62	Nil
	40	7.6	.19	.54	
	80	8.7	.11		
	160	9.8	.06	.59	
B	2½	5.7	2.3	.71	3,000 grams of
	5	7.8	1.6		punchings.
	10	9.8	1.0	.70	13/32" x ⅛"
	20	11.6	.58		
	40	13.4	.34	.43	
	80	15.0	.19	.36	
	160	16.7	.10	.20	
C	1¼	5.2	4.2	.64	3,000 grams of
	2½	8.1	3.2		punchings.
	5	11.3	2.2	.55	13/16" x ¼"
	10	15.1	1.5		
	20	18.9	.95	.28	
	40	23.3	.58		
	80	28.6	.36	.20	
	160	34.1	.21	.09	
D	2½	4.5	1.8	.67	3,000 grams of
	5	6.3	1.3		punchings.
	10	8.0	.80	.59	5/16" x ⅛"
	20	9.5	.47		
	40	11.2	.28	.47	
	80	12.8	.16		
	160	14.2	.09	.35	
E	1¼	4.9	3.9		
	2½	7.3	2.9	.68	3,000 grams of punchings
	5	9.6	1.9		
	10	12.2	1.2	.64	5/8" x 1/8"
	20	14.8	.74		
	40	17.6	.44	.49	
	80	20.6	.26		
	160	23.3	.15	.20	

Table IV

Test	Time of Grind (Minutes)	20 Mesh Oversize			20 Mesh Undersize			Combined 20 Mesh Undersize			Grinding Media
		% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	
A	5	96.3	.12	81	3.7	.73	19	29.1	.68	70	Nil.
	20	93.6	.10	69	6.4	.69	31	31.1	.67	75	
	40	92.4	.095	64	7.6	.66	36	32.0	.67	76	
	160	90.2	.085	55	9.8	.64	45	33.6	.67	80	
B	$2\frac{1}{2}$	94.3	.11	71	5.7	.71	29	30.6	.67	74	3,000 gram of punchings 13/32" x $\frac{1}{8}$"
	10	90.2	.08	51	9.8	.70	49	33.6	.68	81	
	40	86.6	.065	40	13.4	.63	60	36.2	.66	85	
	80	85.0	.06	36	15.0	.60	64	37.5	.65	87	
	160	83.3	.055	33	16.7	.56	67	38.7	.64	88	
C	$1\frac{1}{4}$	94.8	.11	76	5.2	.64	28	30.2	.67	72	3,000 gram of punchings. 13/16" x $\frac{1}{4}$"
	5	88.7	.08	52	11.3	.59	48	34.7	.65	81	
	20	81.1	.065	37	18.9	.47	63	40.3	.60	86	
	80	71.4	.045	23	28.6	.39	77	47.4	.54	91	
	160	66.0	.04	19	34.1	.33	81	51.4	.51	93	
D	$2\frac{1}{2}$	95.5	.115	79	4.5	.67	21	29.7	.67	71	3,000 gram of punchings 5/16" x $\frac{1}{8}$"
	10	92.0	.095	64	8.0	.64	36	32.3	.67	77	
	40	88.8	.085	53	11.2	.59	47	34.6	.65	81	
	160	85.8	.075	45	14.2	.54	55	36.8	.63	83	
E	$2\frac{1}{2}$	92.7	.095	64	7.3	.68	36	31.8	.67	76	3,000 gram of punchings $\frac{5}{8}$" x $\frac{1}{8}$"
	10	87.8	.075	42	12.2	.66	58	35.4	.67	85	
	40	82.4	.06	32	17.6	.65	68	39.4	.66	90	
	160	76.7	.03	15	23.3	.61	85	43.5	.64	94	

Table IV shows the percentage recovery and grade of fines obtained after various stages of grinding the plus 20 mesh fraction. The overall recovery and grade from the minus 20 mesh is also shown in this table.

The figures shown are all cumulative e.g. the third line of Test B means that after 40 minutes grinding of the plus 20 mesh sample with 3,000 grams of 13/32" x $\frac{1}{8}$" punchings 86.6% of the original weight was in the plus 20 mesh fraction which assayed .065% U308 contained 40% of the total uranium. The remaining 13.4% of the sample had been ground finer than 20 mesh, assayed .63% U308 and contained 60% of the uranium which was in the original sample. The combined minus 20 mesh material constituted 36.2 of the original weight, assayed 66% U308 and contained 85% of the uranium.

<u>(II) Series B.</u> In this series, the separations were made at 30 mesh. The percentage by weight and the assay of the plus and minus 30 mesh fractions are shown in Table V.

<u>Table V.</u>

Fraction	Weight %	% U308	% Distribution of U308
Plus 30 mesh	81.4	.18	53
Minus 30 mesh	18.6	.70	47
Composite Head	100.0	.28	100

The details of the grinding media used in these tests is shown in tables VI and VII. The general procedure was as described under series A. Table VI shows the rate of grinding and the grade of the fines produced at different stages of each grinding test, while VII shows the recoveries and grades of the two products after different times of grinding. As in Table IV the columns headed, "Combined 30 mesh undersize", refer to the primary minus 30 mesh together with the minus 30 mesh obtained by grinding of the original oversize. These figures therefore represent the results of differential grinding as a means of concentrating the original ore.

In the laboratory tests the slugs were separated from the ore by screening, but magnetic separation would be preferable in a continuous treatment plant.

3. DISCUSSION OF RESULTS.

Tests A, B, D and E of Series A indicate that the recovery of uranium under these conditions is roughly a linear function of the wight ground through 20 mesh. This relation may be represented in the form:-

$$R = 1.8W + 18$$

where R is the percentage recovery of urnaium and W is the weight ground through 20 mesh. To conform to this relation the corresponding assay of the minus 20 mesh material would be :-

$$A = .50 + \frac{5}{W}$$

where A is the percentage of U_3O_8 in the minus 20 mesh. These relations represent roughly the behaviour of the sample for W in the range from 30-40%.

In the case of test C where heavier slugs were used for grinding the gauge was ground more readily and the weight of fines produced was higher than indicated by the above relation.

Of the four tests A, B, D and E, reference to Table III shows that E gave the highest rate of grinding and the conditions of this test therefore represent the best conditions of this series i.e. Grinding after punchings $\frac{5}{8}$" x $\frac{1}{8}$" which gave 90% recovery of the uranium after 40 minutes grinding with the enriched material containing .66% U_3O_8.

Table VI.

Test	Time (minutes)	% Weight minus 30 mesh	Average rate of Grinding % per unit time	% U308 in minus 30 mesh	Grinding Media used
F	2½	5.7	2.3		3,000 grams of punchings. ⅝" x ⅛"
	5	7.7	1.5		
	10	9.6	.96	.89	
	20	11.4	.57		
	40	13.1	.33		
	80	15.0	.19		
	160	16.8	.10	.67	
G	2½	6.3	2.5		3,00 grams of punch- 13/16" x ¼"
	5	9.1	1.8	.83	
	10	12.0	1.2		
	20	15.4	.77		
	40	19.3	.48	.73	
	80	24.0	.30		
	160	29.9	.19	.21	
H	1¼	5.6	4.5		3000 grams of punchings ⅝" x ⅛"
	2½	7.7	3.1		
	5	10.0	2.0	.80	
	10	12.4	1.2		
	20	15.0	.75	.68	
	40	17.8	.44		
	80	20.9	.26		
	160	24.0	.15	.29	
J	2½	5.8	2.3		1500 grams of punchings 13/16" x ¼"
	5	8.1	1.6		
	10	10.5	1.0	.82	
	20	13.1	.65		
	40	16.2	.40	.77	
	80	20.0	.25		
	160	24.9	.15	.41	
K	2½	4.9	2.0		1500 grams of punchings ⅝" x ⅛"
	5	6.9	1.4		
	10	8.8	.88	.90	
	20	10.6	.53		
	40	12.6	.31	.86	
	80	15.2	.19		
	160	18.0	.11	.66	

TABLE VII.

Test	Time of Grind (minutes)	30 mesh Oversize			30 mesh Undersize			Combined 30 mesh undersize			Grinding Media
		% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	% Weight	% U_3O_8	% Recovery of U_3O_8	
F	10 160	90.4 83.2	.11 .065	56 29	9.6 16.8	0.89 0.79	44 71	26.1 32.1	.75 .73	70 84	3,000 grams of punchings $\frac{3}{8}$" x $\frac{1}{8}$"
G	$2\frac{1}{2}$ 40 160	93.7 80.7 70.1	.14 .045 .025	72 20 9	6.3 19.3 29.9	0.83 0.76 0.57	28 80 91	23.7 34.3 42.9	.72 .73 .62	61 89 95	3,000 grams of punchings 13/16" x $\frac{1}{4}$"
H	10 40 160	87.6 82.2 76.0	.115 .085 .06	57 38 24	12.4 17.8 24.0	0.80 0.76 0.68	43 62 76	26.7 30.8 38.1	.73 .72 .64	69 79 87	3,000 grams of punchings $\frac{5}{8}$" x $\frac{1}{8}$"
J	10 40 160	89.5 83.8 75.1	.11 .065 .02	54 29 9	10.5 16.2 24.9	0.82 0.82 0.68	46 71 91	27.1 31.8 38.9	.73 .74 .68	71 84 95	1,500 grams of punchings 13/16" x $\frac{1}{4}$"
K	10 40 160	91.2 87.4 82.0	.115 .085 .045	57 39 20	8.8 12.6 18.0	0.90 0.89 0.82	43 61 80	25.7 28.8 33.3	.75 .76 .75	69 79 89	1,500 grams of punchings $\frac{5}{8}$" x $\frac{1}{8}$"

The results of series B show that in general screening the sample at 30 mesh instead of at 20 mesh as in series A, leads to higher grade of enriched product but the grinding rate is naturally slower when working at the finer size.

Table VIII has been constructed to show the results of a few of these tests in a form where the tests can be compared more readily. The figures in these tests have been interpolated graphically so as to represent in each case a 90% recovery of the uranium.

Table VIII.

Test	Screen Mesh	Time of Grinding (minutes)	Total weight of Screen undersize %	Grade of Screen Undersize % U_3O_8	Recovery of U_3O_8 %
C	20	60	46	.55	90
E	20	50	40	.63	90
G	30	60	36	.70	90
H	30	260	40	.63	90
J	30	100	36	.70	90
K	30	200	34	.74	90

The figures for test H appear to be rather erratic but the general conclusions are that screening on 30 mesh gives a higher grade of product than 20 mesh, and that slow grinding with light grinding media, is more selective than fast grinding with heavier media.

SUMMARY.

1. Tests on a sample of ore containing 0.28% U_3O_8 have shown that grinding and screening can give an enriched product, 30-40% of the original weight with a recovery of 90% or more of the uranium. This product assaying 0.65-0.70% U_3O_8, should be much better for leaching than the untreated material.

2. Slow grinding with light media is more selective than faster grinding with heavier media. The optimum conditions for grinding would have to be decided on a plant-size mill since for example, a ¾" ball or slug would cause far more impact crushing in a 4' diameter mill than it would in a 10" diameter laboratory mill.

3. Screening at 30 mesh gave higher grades of concentrate than screening at 20 mesh, but there are certain factors which may give trouble in finer screening. The larger period of grinding required when separating at finer sizes will mean that more dust is produced and this may give trouble in leaching. Finer screens would also mean more maintenance and more screen area required for efficient work. These factors could however only be assessed correctly in a continuous plant.

Date 7/3/45.

IV. CONCENTRATION TESTS ON URANIUM ORES FROM MOUNT PAINTER (Fourth Report)

by

Professor H.W. Gartrell and D.R. Blaskett
Joint Investigation of the Council for Scientific and Industrial Research

and

South Australian School of Mines and Industries
Bonython Laboratory, Adelaide.

1. GENERAL.

This report describes some further tests on the laboratory concentration described in our first report.

Test A described below was made on a hand picked parcel of ore from the 50 ft. level winze No. 6 workings, Mt. Painter while Test B was on the minus 20 mesh fraction of a sample from No. 5 Lode, Mt. Painter East.

2. DETAILS OF TESTS.

The machine used for these tests was described fully in our report No. 148A. The only alteration made prior to these tests was the addition of two more oil sprays, which reduced the spacing of the sprays to 2½". This closer spacing of the jets was found to give a more uniform coating of oil on the surface and less tnedency for the material to slide over the surface instead of rilling as desired.

a. Test A.

The parcel of hand picked ore as received assayed 0.91% U_3O_8. This grade, although higher than most of the ore samples, approximates that likely to be produced by rumbling and screening 0.3% ore. After crushing to minus 12 mesh in the jaw crusher and rolls it was screened into several fractions and the fine dust removed by treatment in the Federal dust classifier. The screen analysis of the sized material is shown in Table 1. The screens used for these tests were a set of 12" commercial screens.

Table 1.

Fraction	Weight %	Cum. Weight %	% U_3O_8	% Distribution of U_3O_8	Cum. % Distribution of U_3O_8
Plus 30 mesh	54.8	54.8	0.21	12.6	12.6
30/60 mesh	17.5	72.3	0.71	13.6	26.2
60/100 mesh	8.4	80.7	1.39	12.9	39.1
Minus 100 mesh sand	6.1	86.8	2.41	16.2	55.3
Dust	13.2	100.0	3.08	44.7	100.0
Composite	100.0		0.91	100.0	

The firsty four fractions shown in the above table were treated on the sticky-surface concentrator, allowing each sample two passes through the machine to give a tailing and rougher concentrate. The concentrate in each case was cleaned by retreatment from which the cleaner tail was again retreated to produce a small amount of concentrate and tailing. In this way a final concentrate and tailing were made from each fraction.

The fine dust cannot be treated satisfactorily on the machine but it is considered that in any treatment scheme provision would have to be made for the leaching of such dust so that its uranium content is regarded as recoverable.

The results of these concentration tests are shown in Table II. The overall recovery shown is 78% of the uranium in a concentrate assaying 2.8% U_3O_8. However, the plus 30 mesh fraction could be treated better by differential grinding by which we could reasonably expect to recover at least 80% of the uranium in a grade of about 0.7% U_3O_8 and this after separation into sized fractions should yield a recovery of about 80% of its uranium or 64% of the uranium in the original plus 30 mesh. With this modification of the treatment the indicated overall recovery is about 85% of the total uranium in a grade of about 2/9% U_3O_8. A further improvement in recovery with little loss in grade could be made if the minus 100 mesh fraction was added to the other concentrates without treatment.

Table II

Fraction	Weight	Concentrate Weight % of fraction	Analysis % U_3O_8		Recovery of U_3O_8 %	
			Feed	Concentrate	In fraction	In total
Plus 30 mesh	54.8	8.8	0.21	1.01	42	5.3
30/60	17.5	17.4	0.71	2.47	61	8.3
60/100	8.4	29.1	1.39	3.72	78	10.1
Minus 100 (sand)	6.1	33.0	2.41	4.41	60	9.7
Dust	13.2	100.0	3.08	3.08	100	44.7
Composite	100.0	25.4	0.91	2.80		78.1

b. Test B.

This test was made on the minus 20 mesh material resulting from the differential grinding tests described under series A in report No. 148B. The complete test was therefore roughly equivalent to the following:-

The original sample assaying 0.28% U_3O_8 was crushed to minus $\frac{3}{8}$" and screened on 20 mesh. The plus 20 mesh was then concentrated by differential grinding and the fines from the grinding added to the original minus 20 mesh thus producing two products, a coarse tailing and an enriched sample for further treatment. The distribution of uranium and grade of these products is shown in Table III.

Table III

Sample	Weight %	% U_3O_8	% Distribution of U_3O_8
Coarse Tailing	48	.06	11
Enriched Head	52	.48	89
Original Head	100	.28	100

The enriched head sample was then screened into fractions and the fine dust removed by treatment in the Federal dust classifier. The screen analysis of these fractions is shown in Table IV. It will be noted that the percentage of dust in this case is much less than in test A. The reason for this is that the sample for test A was crushed to minus 12 mesh before screening whereas in this test the sample was only crushed to $\frac{3}{8}$".

The first four fraction were then treated on the laboratory concentrator in a similar manner to that described under test A. The fine dust was not treated but it is included as portion of the final composite concentrate.

Table V shows a concentrate grade of 1.1% U_3O_8 and an overall recovery of 71% based on the enriched head. The complete test consisting of differential grinding and screening followed by sticky-surface concentration of the fines shows a recovery of 63% of the uranium from a sample containing 0.28% U_3O_8. The weight of concentrate obtained was 15% of the original weight while the grade of concentrate was 1.1% U_3O_8.

3 SUMMARY.

1. Some further tests on the laboratory sticky-surface concentrator have shown that recoveries of 60 - 70% of the uranium can be obtained with a concentrate 2 or 3 times the grade of the feed.

2. When treating a low grade feed a preliminary enrichment should be made by differential grinding and screening. Sticky-surface concentration of this enriched material should then give a final concentrate grade, 4 or 5 times that of the original feed.

Table IV.

Fraction	Weight %	Cum. Weight %	% U_3O_8	% Distribution U_3O_8	Cum.% Distribution of U_3O_8
Plus 30 mesh	13.7	13.7	0.27	7.6	7.6
30/60	28.0	41.7	0.45	26.1	33.7
60/100	17.3	59.0	0.64	22.9	56.6
Minus 100 (sand)	25.2	84.2	0.42	21.9	78.5
Dust	15.8	100.0	0.66	21.5	100.0

Table V.

Fraction	Weight %	Concentrate Weight % of Fraction	Analysis % U_3O_8		Recovery of U_3O_8 %	
			Head	Concentrate	In Fraction	In Total
Plus 30 mesh	13.7	14.1	.27	1.31	69	5.3
30/60	28.0	13.9	.45	1.96	60	15.6
60/100	17.3	21.1	.64	2.02	66	15.1
Minus 100 (Sand)	25.2	19.2	.42	1.37	63	13.8
Dust	15.8	100.0	.66	.66	100	21.5
Composite	100.0	30.1	.48	1.14		71.3

3. Assuming that the fine dust can be satisfactorily leached, tests have indicated that when treating a sample containing: 90% U_3O_8, a combined process of differential grinding and screening followed by sticky-surface concentration of the material in the 30 - 100 mesh size range should give a concentrate containing about 3% U_3O_8 with a recovery of 85%.

Date 7/3/45.

V. CONCENTRATION TESTS ON URANIUM ORES FROM MT. PAINTER

(Fifth Report)

by

D. R. BLASKETT

Joint Investigation of the Council for Scientific and Industrial Research

and the

South Australian School of Mines and Industries

....

1. GENERAL

This report describes some further work on the concentration of Mt. Painter Uranium ores and gives a general summary of the results of the investigations on these ores.

2. DESCRIPTION OF THE STICKY-SURFACE CONCENTRATING MACHINE.

Since the previous report on Mt. Painter ores was issued from this laboratory a larger sticky-surface concentrating machine has been built similar in principle to the earlier laboratory model described in the first report. The more recent machine has a steel cylinder 24 inches diameter and 36 inches long while the cylinder of the first machine was of glass, 12 inches diameter and 16 inches long.

The arrangement of the oil sprays and the wiper for removing the concentrate is similar to that used on the earlier machine.

3. DETAILS OF TESTS.

Tests carried out on the large laboratory machine were intended to imitate as far as possible actual operating conditions. The scheme of treatment followed was similar to that described in Report No. 148D.

a. Test A. was made on a sample of hand picked ore from the 50 foot level winze No. 6 workings. This was a sample of the same parcel of ore as was used for test A, report No. 148 D, but in this case the sample was concentrated by differential grinding before treating on the sticky-surface concentrator. The sample was crushed minus $\frac{3}{8}$ inch and screened on 30 mesh. The plus 30 mesh fraction was then treated by grinding it in 2,500 gram lots for 20 minutes with 3,000 grams of $\frac{3}{4}$" x $\frac{1}{4}$" slugs as described in the earlier reports. The ground material was screened again on 30 mesh and the plus 30 mesh fraction was set aside as the coarse tailing. The combined minus 30 mesh material was then divided into three parts by screening and treatment in the laboratory Federal Dust Classifier. The weights of these fractions, their uranium content and the distribution of uranium is shown in Table I.

Table I.

Fraction	Weight %	% U_3O_8	% Distribution of U_3O_8
Plus 30 mesh	74.8	0.13	10.7
30/60 "	9.3	1.35	13.9
Minus 60 mesh sand	8.8	2.84	27.6
Dust	7.1	6.08	47.8
Composite Sample		0.91	

The two fractions, 30/60 mesh and minus 60 mesh sand, were treated in the large scale laboratory sticky-surface concentrating machine described earlier in this report. The rates of treatment were 140 and 100 lb./hr. respectively, these rates being approximately seven times the rates used on the smaller scale machine for similar material. It appears from these few figures that with similar machines, the capacity would be proportional to a power of the diameter between 2 and 3. If this relation were maintained in larger sizes it would not be difficult to obtain sufficient capacity for a plant size unit. Because of the relatively high head values in this particular sample, each fraction was given one pass only through the machine, producing a concentrate and tailing. The results of these concentration tests are shown in detail in Tables 2 and 3.

Table II. - Concentration of 30/60 mesh fraction.

	Weight %	% U_3O_8	% Distribution of U_3O_8
Concentrate	26.4	3.68	72
Tailing	73.6	0.51	28
Head	-	1.35	-

Table III. - Concentration of minus 60 mesh sand fraction.

	Weight %	% U_3O_8	% Distribution of U_3O_8
Concentrate	40.5	4.56	65
Tailing	59.5	1.67	35
Head	-	2.84	-

Table 4 shows the uranium content and distribution of uranium in the four principal products from the complete test.

Table IV

Product	Weight%	% U_3O_8	% Distribution of U_3O_8
Coarse Tailing (from differential grinding.)	74.8	0.13	10.7
Sand Tailing (from concentrator)	12.0	1.01	13.6
Sand Concentrate (from concentrator)	6.1	4.20	27.9
Fine Dust	7.1	6.08	47.8
Original Head		0.91	

b. Test B was made on a sample of ore from No. 5 Lode, East Painter, assaying 0.17% U308. This grade is much lower than that of the sample used in Test A but the same scheme of treatment was followed, i.e. crushing to minus $\frac{3}{8}$", screening on 30 mesh, differential grinding of the oversize, separation of the minus 30 mesh into three fractions and treatment of the two coarser fractions by the sticky-surface concentrator. Table 5 shows the distribution of the uranium in the sample after differential grinding.

Table V.

Fraction	Weight %	% U_3O_8	% Distribution of U_3O_8
Plus 30 mesh	70.2	0.05	20.6
30/60 "	7.8	0.32	14.7
Minus 60 mesh sand	10.6	0.48	29.9
Dust	11.4	0.52	34.8
Composite Sample	100.0	0.17	

The results of the concentration tests are shown in Tables 6 and 7, while Table 8 shows the overall result of differential grinding and concentration.

Table VI. - Concentration of 30/60 mesh fraction.

	Weight %	% U_3O_8	% Distribution of U_3O_8
Concentrate	12.5	1.57	61
Tailing	87.5	0.14	39
Head		0.32	

Table VII. - Concentration of minus 60 mesh sand.

	Weight %	% U_3O_8	% Distribution of U_3O_8
Concentrate	14.8	2.10	65
Tailing	85.2	0.20	35
Head		0.48	

Table VIII.

Product	Weight %	% U_3O_8	% Distribution of U_3O_8
Coarse tailing (from differential grinding)	70.2	0.05	20.6
Sand tailing (from concentrator)	15.4	0.17	15.8
Sand concentrate (from concentrator)	3.0	1.50	28.8
Fine Dust	11.4	0.52	34.8
Original Head		0.17	

The results of the two tests quoted above show that the work done by the larger laboratory machine is comparable with that previously done on the small laboratory machine. Variations may be attributed to the different types of ore and to minor features of operation but broadly speaking it is believed that the results obtained on the smaller machine and detailed in earlier reports could be reproduced on a much larger scale machine.

4. GENERAL SUMMARY OF CONCENTRATION PROCESSES TESTED ON MT.PAINTER ORES.

a. INTRODUCTION.

Throughout this investigation the question of water supply for a treatment plant has been regarded as important and emphasis has been placed on the desirability of using dry concentration processes if possible.

It is believed that the ultimate stage of uranium recovery must be a leaching process and the question to be decided is whether the ore should be leached directly or concentrated prior to leaching. Wet concentration prior to leaching would involve a water consumption at least equal to a direct leaching process, but would mean a saving in the cost of acid which would have to offset the lower overall recovery. Dry concentration on the other hand would mean less leaching capacity and correspondingly lower acid and water consumption.

b. CONCENTRATION PROCESSES.

(1) Wet processes of Concentration. Jigging, tabling and classification have been tried but show no chance of successful application. Flotation has also been tried and although tests gave indication that the uranium minerals would respond to certain reagents, the variable nature of the gangue and the tendency of the minerals to slime so readily would create difficult problems in operation. It was therefore not considered worth following further since a high recovery and high grade of concentrate would be necessary for the process to possess any great advantage over direct leaching.

(2) Dry Processes of Concentration. Magnetic concentration has been tested and showed encouraging results on some types of ores while on others no appreciable amount of gangue could be taken out magnetically. These differences are due to the wide variation in the type of gangue with which the uranium minerals are associated.

Differential grinding of minus $\frac{3}{8}$ inch ore has been found to give a useful concentration of the valuable minerals in the fines and it is suggested that any treatment scheme should incorporate this process. By its use an ore containing about 25 per cent U_3O_8 can be enriched to about 0.7 per cent U_3O_8 with a recovery of from 80 - 90 per cent. In so doing about two-thirds of the ore is rejected as a coarse low-grade tailing.

A considerable amount of test work has been carried out on a process of sticky-surface concentration on the inside surface of a revolving cylinder and the results achieved in the laboratory indicate that such a process could be applied on a larger scale and made part of a general dry-treatment scheme. The process has been used on sandy material in the size range from 30 down to 200 mesh and within this range, recoveries of from 65 - 70 per cent have been obtained with an enrichment of about 5 times in the grade of product. Details of the process can be found in this and earlier reports.

c. CONCLUSION.

Since dry concentration by differential grinding and screening is so simple it is recommended that it should be part of the general treatment scheme. It has the advantage of rejecting about two-thirds of the ore as a low-grade coarse tailing with correspondingly lower consumption of acid and water and richer solutions to be handled. Since the concentrated product to be leached would be all minus 30 mesh, an agitation leach would be required in place of the percolation leach which could be used for a direct leaching process but any increase in capital cost should be largely offset by the increased extraction possible by agitation leaching of fine material.

VI. ELECTROSTATIC CONCENTRATION OF URANIUM ORES FROM MOUNT PAINTER AND EAST MOUNT PAINTER SOUTH AUSTRALIA

by

H. H. Dunkin and J. G. Hart
Melbourne University and Ore-Dressing Laborator.
under
the Joint Control of the Council for Scientific
and Industrial Research
and the
University of Melbourne (Metallurgy Department)

1. SYNOPSIS.

The work dealt with in this report was carried out at the request of Mr. S. B. Dickinson, Director of Mines, South Australia. It was restricted to testing by electrostatic methods, ore samples from Mt. Pinter and East Mt. Painter. It was found that magnetic treatment gave a useful preliminary concentration of at least one sample examined by rejecting ironstone giving enriched silicous material for the electrostatic treatment, therefore magnetic work was also carried out when it was required.

Most of the electrostatic work was carried out by courtesy of the Titanium Alloys Manufacturing Co. at their Tweed Heads, N.S.W. plant and through the kindly co-operation of Mr. W. H. Derrick, the company's Manager. Preliminary arrangements for this work were made by Mr. J. Malcolm Newman, Controller of Minerals Production, to whom thanks are due. The conditions there were not good for test work of this type, the machines were not shielded, at times therefore windy weather upset the operation. Certain other difficulties also arose which could not be rectified conveniently either by the company or the laboratory staff. Therefore, at a late stage, laboratory electrostatic equipment was constructed.

The work failed to solve the problem of concentrating the uranium minerals from the Mt. Painter ores. At its best, it gave an indication that on suitably sized material a partial, but not satisfactory concentration of the uranium minerals was obtainable.

2. MINERAGRAPHIC WORK.

Mineragraphic work on the Mount Painter ores is described in Dr. F. L. Stillwell's mineragraphic report No. 315 of 30th October, 1944. The uranium-bearing minerals mentioned are autunite, torbenite, meta-torbenite, uranophane, dewindtite, fergusonite and an unidentified silicate. The chief uranium bearing minerals present in the ore samples tested were torbenite ($Cu\ O.2\ UO3.P_2O_5.H_2O$) and meta-torbenite ($CuO.2\ UO_3.P_2O_5\text{-}8H_2O$) with some autunite ($Ca.O2\ UO_3.P_2O_5.10H_2O$). The chief gangue minerals present were quartz and limenite.

3. SIEVES.

The sieves mentioned in this report are of the British Standard Sieve Series. No responsibility is taken for the results given except in so far as they apply to the samples tested.

4. NOTES ON THE ELECTROSTATIC METHOD USED.

All of the work was carried out on a Sutton Steel and Steele type separator. In this type of separator a 6 inch brass roll, forming the earthed electrode, is rotated at relatively high speed (100-500 R.P.M.) and fed with the mineral particles on a line just behind the top center of the roll. The mineral particles are heated before being fed to the machines feed hopper and to prevent chilling they are also fed down a heated chute to the roll. A single stainless steel wire running the length of the roll is used as the main electrode. This is located some 2 to 4 inches from the roll at a position of approximately 1 o'clock at the clockwise end. A neon tube electrode, in series with the wire, is located at 3 o'clock.

While the foregoing electrode arrangement was used for most of the test work, it was found that the neon tube electrode was not necessary. An exactly similar result was obtained by using a wire in place of the neon tube. Even better results were secured by using wire electrodes at 1, 2, 3 and 4 o'clock.

High voltage smoothed direct current was supplied by a tube type rectifier with smoothing chokes. It was found that the most satisfactory separation of the torbenite was obtained by using a positive to earth feed roller. Fairly high voltage (25 to 30 KV) then held the quartz to the roller or diverted it away from the separating electrodes, while torbenite plus iron bearing quartz and residual bematite left the roll towards the separating electrodes. At low voltages (10 KV?) and the same polarity viz. positive to earth, torbenite clings to the roll and is removed by the wiper while quartz merely impinges on the roll and is thrown off to the front. The wiper is situated at a position approximating to 8 o'clock. These are only tendencies on the parts of the minerals named, there is no sharp separation between them, and to get any reasonably successful separation repeated treatment must be given to the products.

5. MT. PAINTER SAMPLE.

a. Preparation and Sizing.

The sample of ore used in the first test series came from No. 6 Workings, Mt. Painter. It contained a very high proportion of iron oxide gangue. The whole sample (2 cwt.) was crushed through a jaw crusher and rolls in closed circuit with a 10 mesh screen (aperture .0660 inch) until all the sample passed the screen. It was then sized wet and dry on a range of screens from 14 to 100 mesh. Minus 100 mesh material was separated by an hydraulic classifier into coarse and fine fractions. Table 1 shows a sizing analysis of the products obtained in the crushing and sizing. The results confirm those obtained elsewhere that the torbenite preferentially curshes to the finer sizes. The classified minus 100 mesh fines sized 99.2% minus 300 mesh, this is the highest grade fraction and was rejected as too fine for the dry processing proposed as the next treatment step. This rejection of slime is accompanied by the loss of 23.6% of the U_3O_8 content of the ore.

Table 1

Bs Screen	Aperture inches	% Weight	U_3O_8 Assay %	U_3O_8 Distribution %
+ 14	.0474	28.0	0.17	13.5
+ 18	.0336	14.6	0.19	8.1
+ 25	.0236	12.75	0.27	9.7
+ 36	.0166	7.65	0.31	6.75
+ 52	.0116	6.1	0.37	6.4
+ 72	.0083	6.85	0.48	9.25
+ 100	.0060	5.65	0.56	9.0
- 100 Coarse		9.25	0.52	13.75
- 100 Fine		9.15	0.91	23.6
Composite		100.0	0.352	100.0

The composite head value of the sample was 0.35% of U_3O_8.

b.. Magnetic treatment.

All of the sized fractions except the - 100 mesh fines were treated on the magnetic separator for removal of the ferruginous gangue; the results are shown in table 2.

In the fractions coarser than 52 mesh magnetic recoveries are lower than in the finer fractions. Rejections of magnetic material ranges around 80% at all sizes. Composite torbenite quartz particles were observed in the coarser non magnetic fractions (+14 to +36), this showed that liberation was not complete at the coarser sizes. The following tabulations show the overall results of the treatment and the recoveries of U_3O_8 obtainable by rejecting various products.

Table 2

Magnetic Separation

Product	% Wt.	U_3O_8 Assay %	U_3O_8 Dist. %	U_3O_8 Dist. % in each fraction	% Weight each fraction
+14 Mag	9.7	0.14	7.83	58.05	70.4
" Non	8.3	0.24	5.66	41.95	29.6
+18 Mag	11.7	0.14	4.65	57.4	80.2
" Non	2.89	0.42	3.45	42.6	19.8
+25 Mag	10.47	0.15	4.46	46.1	82.0
" Non	2.3	0.80	5.22	53.9	18.0
+36 Mag	6.35	0.16	2.88	42.95	83.0
" Non	1.30	1.04	3.84	57.05	17.0
+52 Mag	4.97	0.15	2.12	33.0	81.5
" Non	1.14	1.33	4.30	67.0	18.5
+72 Mag	5.3	0.13	1.96	21.15	77.5
" Non	1.55	1.66	7.31	78.85	22.5
+100 Mag	4.61	0.20	2.62	29.15	81.8
" Non	1.02	2.20	6.37	70.85	18.2
-100 Coarse Mag	7.36	0.19	397	28.9	79.6
" " Non	1.88	1.83	9.77	71.1	20.4
-100 Fines	9.13	0.91	23.58	100.	

Table 3

Overall Magnetic Results

Product	% Wt.	U_3O_8 Assay %	U_3O_8 Distn. %
All Magnetics	70.48	0.15	30.5
All Non Mag.	20.39	0.79	45.92
-100 Fines	9.13	0.91	23.58

Table 4

Product	% Wt.	U_3O_8 Assay%	U_3O_8 Distn %
Magnetics	70.48	0.15	30.50
Non Mag. + Fines	29.52	0.83	69.50
Magnetics + Fines	79.61	0.24	54.08
Non Magnetics	20.39	0.79	45.92

The minus 100 mesh fines was not amenable to magnetic treatment, it was far too fine and dusty. It is certain that such a product will not be amenable to electrostatic treatment, it must therefore be rejected. The recovery then for the magnetic treatment at the minus 100 mesh sizing is 45.9% of the total U_3O_8. If the ore had been crushed to the liberation point, or say to - 36 mesh at which sizing composites would be absent the additional slime loss would probably cancel any further recovery in the non magnetics.

c. Electrostatic treatment of non-magnetics.

Most of the electrostatic work was carried out under the following conditions:

Temperature of feed	180°F.
6" roller, R.P.M.	240
Earth polarity	positive
Separating electrode	negative
Voltage	30,000? (approx.)

The voltages quoted throughout are not really known accurately. The only exact measurement was made with an electrostatic voltmeter reading to 12 KV which was about the lowest range of the rectifier used. Voltages above this were calculated, the calculation being based on the input voltage to the HT transformer. This input voltage was controlled from a small auto transformer with numerous tappings.

The first series of electrostatic tests are shown in table 5. Plus 14 mesh material was too coarse for any satisfactory separation to be obtained, this also applied to plus 18 mesh material, however, in the latter case the testing was completed so that the results could be recorded. All testing was carried out on the non magnetic fractions shown in table 2. The samples were split by the electrostatic treatment into "concentrates", that is, the torbenite product thrown forward towards the separating electrodes, "tailings", the product pinned to the roll and carried around until it dropped off or was removed by the wiper. "Middlings" is the product dropping between the concentrates and tailings. In some cases several concentrate products were taken to see if really high grade torbenite concentrate could be selected. The "first cut" concentrate was always that portion furthest from the roll.

Table 5

Electrostatic Separation of Non Magnetic Products.

Test No.	Prod. Treated	Product	% Wt.	U_3O_8 Assay %	U_3O_8 Distn. %
1	+ 18 Non	Concentrate	31.8	0.54	33.9
		Middling	66.3	0.49	64.2
		Tailing	1.9	0.50	1.9
		Composite	100.	0.506	100.
2	+ 25 Non	Concentrate	21.1	1.77	50.05
		Middling	32.55	0.49	21.4
		Tailing	46.35	0.46	28.55
		Composite	100.	0.746	100.
3	+ 36 Non	Concentrate	64.5	1.48	83.3
		Middling	26.4	0.53	12.2
		Tailing	9.1	0.57	4.5
		Composite	100.	1.146	100.
4	+ 52 Non	Concentrate	22.0	2.48	50.7
		Middling	7.3	1.45	9.85
		Tailing	70.7	0.6	39.4
		Composite	100.	1.07	100.
5	+ 72 Non	Concentrate	10.2	3.03	21.3
		Middling	22.4	2.51	38.8
		Tailing	67.4	0.86	39.9
		Composite	100.	1.45	100.
6	+ 100 Non	Concentrate 1st cut	2.0	6.34	6.7
		Concentrate 2nd "	18.55	4.44	43.5
		Middlings	3.5	5.09	9.4
		Tailings	75.95	1.01	40.5
		Composite	100.	1.895	100.
7	- 100 Coarse Non	Concentrate	12.2	3.0	17.4
		Middling	12.1	2.96	17.0
		Tailing	75.7	1.82	65.6
		Composite	100.	2.10	100.

In general it may be said that the electrostatic work is unsatisfactory; while some concentration of uranium has certainly taken place it is accompanied by serious losses. The highest grade uranium bearing product in the above test series contained 6.3% of U_3O_8. The result are best on the plus 72 and plus 100 fractions where recoveries in concentrates and middling products range around 60% of the uranium content of electrostatic feed. At coarser sizes similar recoveries were obtained but the lower grade electrostatic feed at such sizings is reflected in lower grade concentrates. At plus 18 mesh the work has broken down altogether and there is practically no difference in grade between concentrate, middling and tailing. The work has also failed on the minus 100 mesh coarse fraction.

The plus 52 plus 72 and minus 100 mesh coarse tailings produced in the work shown in table 5 were retreated several times on the electrostatic roll to see whether any further concentration of torbenite was obtainable. The results are shown in table 6.

Table 6

Retreatment of electrostatic tailings

Test No.	Product Treated	Product	% Wt.	U_3O_8	
				Assay %	% Distn
8	+ 52 mesh	Cons and Mids	11.5	3.2	53.6
	electrostatic	Tails	88.5	0.36	46.4
	tail	Composite	100.	0.686	100.
		Special Cons	0.15	25.0	4.6
	+ 72 mesh	Mids	7.55	3.6	33.3
9	electrostatic	Tails	92.3	0.55	62.1
	tail	Composite	100.	0.817	100.
	-100 mesh	Cons	0.55	3.4	1.15
10	coarse	Mids	1.7	3.05	3.25
	electrostatic	Tails	97.75	1.55	95.6
	tail	Composite	100.	1.59	100.

In the case of the two coarser fractions the retreatment has concentrated appreciable additional uranium mineral. In the case of the minus 100 mesh product as formerly, no useful concentration of uranium has taken place. The +72 mesh treatment gave a special concentrate containing 25% of U_3O_8. This material was collected a considerable distance in front of the roller, it represents only 4.6% of the uranium content of the material fed to the roll, it indicates however, that it is possible to produce some high grade uranium concentrate by the electrostatic method. Table 7 following shows the overall recovery from the + 52 and + 72 fractions by the technique of repeated electrostatic treatment.

Table 7

Repeated electrostatic treatment

Test No.	Product	% Wt.	Assay %	Distn. %	Ratio of Concentration
	Con Test 4	22.0)	2.48)	48.05)	
4	Mids "	7.3) 37.4	1.45) 2.44	9.35) 80.2	
&	Cons & Mids Test 8	8.1)	3.2)	22.8)	2.66
8	Tails Test 8	62.6	0.36	19.8	
+52	Composite	100.	1.136	100.	
	Cons Test 5	10.2)	3.03)	21.7)	
5	Mids "	22.4) 37.8	2.51) 2.86	39.5) 75.95	
&	Spec.Cons Test 9	[illegible].1)	25.0)	1.75)	2.65
9	Mids Test 9	5.1)	3.6)	12.95)	
+72	Tails Test 9	62.2	0.55	24.05	
	Composite	100.	1.42	100.	

In the case of the plus 52 mesh electrostatic feed the recovery is 80.2% of the uranium in a 37.4% concentrate by weight containing 2.44% of U_3O_8. Minus 52 plus 72 mesh feed gave 76% recovery in a similar bulk and grade of concentrate.

(d) Overall magnetic-electrostatic recoveries.

Table 8 following shows the overall recoveries obtained by magnetic and electrostatic treatment on those products deemed suitable for treatment, that is, finer than 25 mesh. The indicators are that composites exist in coarser material and finer grinding is required to secure satisfactory liberation.

Table 8

Magnetic-electrostatic treatment

Fraction	Product	% Wt. Fract	% Wt. O/all	U_3O_8 Content	U_3O_8 % Distn	U_3O_8 content of all rejects	Ratio of conc.
+ 36	Mag Reject	83.0	6.35	0.16	40.5	feed content	9.0
	Elect Reject	5.95	0.46	0.54	9.9	0.186	
	Concentrate	11.05	0.84	1.48	49.6	0.33	
+ 52	Mag Reject	81.4	4.97	0.15	36.8		14.7
	Elect Reject	11.65	0.71	0.36	12.6	0.176	
	Concentrate	6.9	0.42	2.45	50.6	0.33	
+ 72	Mag Reject	77.4	5.3	0.13	23.6		11.3
	Elect Reject	14.0	0.96	0.55	18.0	0.194	
	Concentrate	8.6	0.59	2.87	58.4	0.43	
- 100 Coarse	Mag Reject	79.7	7.36	0.19	27.75		18.0
	Elect Reject	15.0	1.39	1.56	43.0	0.407	
	Concentrate	5.3	0.49	2.99	29.25	0.55	

Since the above fractions contain only about 45% of the uranium content of the original curshed and sized ore, the overall recovery represented by them is only around 20% of that in the ore.

If the ore had been crushed to minus 25 mesh instead of minus 10 mesh thus eliminating the untreatable plus 25 mesh fractions; and if the uranium content so liberated had distributed itself between the remaining fractions in similar proportions to those given in table 1; also, if the subsequent sizing, magnetic and electrostatic treatment of the products gave similar results to those quoted in table 8; then the overall results of treating the ore at minus 25 mesh sizing would be as follows:

U_3O_8 recovered in electrostatic cons - 28.65% of total

+ 100	Mag Reject	81.8	4.61	0.2	32.05		22.7
	Elect Reject	13.65	0.77	1.01	27.05	0.316	
	Concentrate	4.45	0.25	4.71	40.9	0.51	

(c) Electro

A few electrostatic tests were carried out on magnetic rejects with no expectation of successful results. The work was done because of the chance that it would be successful and thereby allow the magnetic step to be eliminated. The results, recorded in table 9, were quite unsuccessful.

Table 9

Electrostatic treatment of magnetic rejects.

Test No.	Product Treated	Conditions	Products	% Wt.	U_3O_8 Assay %	U_3O_8 Distn. %
11	+72 Mag	+ feed roll 30 KV	Cons	37.75	0.07	28.35
			Mids	54.75	0.10	58.75
			Tails	7.5	0.16	12.9
			Composite	100.	0.09	100.

Table 9 Contd.

Test No.	Product Treated	Conditions	Products	% Wt.	U_3O_8	
					Assay %	Distn %
12	+72 Mag	- feed roll 30KV	Cons	41.2	0.08	37.95
			Mids	51.65	0.07	55.45
			Tails	7.15	0.06	6.6
			Composite	100.	0.06	100.
13	+100 Mag	+ feed roll 30KV	Cons	40.2	0.20	46.6
			Mids	51.8	0.15	45.05
			Tails	8.0	0.18	8.35
			Composite	100.	0.17	100.

6. EAST MT. PAINTER SAMPLE.

The sample of East Mt. Painter ore was said to be rather higher in grade than others tested. It proved to contain around 0.3% of U_3O_8. It appeared to be generally similar to the sample from No. 6 workings except that the proportion of iron oxide gangue was very much lower.

a. Preparation of samples.

In view of the evidence obtained on the earlier sample that crushing to at least 25 mesh was desirable to secure liberation, the sample was crushed to this sizing. It was then split into two portions. One half of the sample was sized wet and dry and the minus 100 mesh fraction classified into coarse and fine. The other half of the sample was split by an hydraulic classifier into several spigot products and an overflow. These different methods of treatment were adopted to determine whether the electrostatic method of separation operated better on sized or classified feed. Table 10 shows the distribution of uranium throughout the sized products whilst Table 11 gives similar results for the classified products.

Table 10

Sizing analysis. Sized portion of sample

BS Screen	Aperture inches	% Wt.	Assay %	Distn. %
+36	.0166	21.45	0.165	12.5
+52	.0116	15.3	0.213	11.5
+72	.0083	12.55	0.262	11.6
+100	.0060	9.55	0.327	11.0
-100 Coarse		21.2	0.417	31.25
-100 Fine		19.95	0.32	22.1
Composite		100.	0.283	100.

Screen tests of -100 mesh fractions.

Bs Screen	Aperture inches	% Weight	
		-100 Coarse	-100 Fine
+150	.0041	38.4	2.6
+200	.0030	17.4	2.7
-200		44.2	94.7

Table 11.

Classified portion of sample - distribution of U_3O_8

Product	% Weight	U_3O_8	
		Assay %	Distn.%
Spigot 1	21.15	0.236	16.65
Spigot 2	18.95	0.223	14.1
Spigot 3	14.5	0.264	12.75
Spigot 4	33.3	0.411	45.6
Overflow	12.1	0.27	10.9
Composite	100.	0.30	100.

Screen tests of spigot products.

Bs Screen	Aperture inches	% Weight				
		Spigot 1	Spigot 2	Spigot 3	Spigot 4	Overflow
+52	.0116	75.5	67.75.	42.5	0.6	
+72	.0083	15.35	19.5	23.9	6.5	
+100	.0060	6.3	9.1	16.1	14.7	
+150	.0041	1.75	2.5	8.7	19.1	
+200	.0030	0.55	0.45	3.4	11.9	100.0
-200		0.55	0.7	5.4	47.2	

As in the case of previous samples, crushing breaks down the torbenite more than the gangue minerals with the result that the torbenite segregates in the finer fractions when the crushed ore is creened or classified.

b. Magnetic treatment.

Tables 12 and 13 give the results of magnetic separation of the coarser products produced by the sizing or classification.

Table 12

Magnetic treatment of sized products.

Product	% Wt.	U_3O_8			% Wt. each fraction
		Assay %	Distn %	Distn % in each fraction	
+36 Mag	6.23	0.08	1.75	14.05	29.05
+36 Non	15.22	0.2	10.75	85.95	70.95
+52 Mag	7.08	0.1	2.5	21.7	46.25
+52 Non	8.22	0.31	9.0	78.3	53.75
+72 Mag	5.42	0.08	1.5	13.15	43.15
+72 Non	7.13	0.4	10.1	86.85	56.85
+100 Mag	3.83	0.07	0.95	8.6	40.1
+100 Non	5.72	0.5	10.05	91.4	59.9
-100 Coarse Mag	7.21	0.14	3.55	11.4	34.0
-100 Coarse Non	13.99	0.56	27.7	88.6	66.0
-100 Fine	19.95	0.32	22.1	100.	
Bulk - all Mag	44.2	0.08	13.1		
Sizes except Non	55.8	0.42	86.9		
fines Comp	100.	0.27	100.		
All Mag + fines	49.7	0.185	32.4		
All Non Mag	50.3	0.381	67.6		
	100.	0.283	100.		

Table 13

Magnetic treatment of Classified products

Product	% Wt.	U_3O_8			
		Assay %	Distn.%	Distn % in ea. fraction	% Wt. ea. fraction
1 Spigot Mag	14.35	0.05	2.4	14.35	67.85
1 Spigot Non	6.8	0.63	14.25	85.65	32.15
2 Spigot Mag	9.3	0.07	2.2	15.35	48.95
2 Spigot Non	9.7	0.37	11.9	84.65	51.05
3 Spigot Mag	5.7	0.07	1.35	10.45	39.35
3 Spigot Non	8.8	0.39	11.4	89.55	60.65
4 Spigot Mag	3.55	0.17	2.05	4.45	10.7
4 Spigot Non	29.75	0.44	43.6	95.55	89.3
Overflow	12.1	0.27	10.85	100.	100.
All (Magnetics & overflow	45.0	0.125	18.85		
All (Non Mag	55.0	0.443	81.15		
All (Composite	100.	0.300	100.		

The following iron and insoluble assays were obtained on the various magnetic and non magnetic products.

Product	% Fe	% Insol	Product	% Fe	% Insol
Magnetic +36	37.6	34.6	Non Magnetic +36	12.5	76.9
" +52	35.7	36.6	" +52	6.2	87.3
" +72	43.0	34.2	" +72	7.3	86.1
" +100	37.5	33.0	" +100	7.4	84.8
" -100 Coarse	36.0	36.2	" -100 Coarse	11.0	78.5
-100 Fine	16.8	64.8			
Magnetic 1 Spigot	39.3	30.7	" 1 Spigot	4.9	89.7
" 2 "	34.2	37.8	" 2	5.1	89.5
" 3 "	32.2	42.5	" 3	4.5	89.9
" 4 "	32.8	40.7	" 4	12.9	74.4
Overflow	17.5	62.7			

Table 14 following, shows the iron and insoluble distribution in the composite magnetic and non magnetic products.

Table 14

Iron and Insoluble distribution in products from magnetic separation

Product		% Wt.	Assays % Fe	Assays % Insol	Distribution % Fe	Distribution % Insol
Sized	Magnetics and fines	49.7	29.2	46.55	74.8	36.15
	Non Magnetic	50.3	9.75	81.25	25.2	63.85
	Composite	100.	19.4	64.0	100.	100.
Classified	Magnetics and overflow	45.0	31.0	43.05	73.35	30.2
	Non Magnetic	55.0	9.2	81.4	26.65	69.8
		100.	19.0	64.1	100.	100.

The magnetic treatment on the sized material excluding fines has resulted in the recovery of 86.9% of the uranium in a 55.8% by weight non magnetic product. When the fines are also rejected the overall recovery in non magnetics is only 67.6%. The magnetic treatment and sizing rejected 74.8% of the iron from the products.

Similar treatment on the classified fractions resulted in the recovery of a 55% by weight non magnetics fraction containing 81.15% of the total uranium. 73.35% of the iron was removed in the magnetics and overflow.

The iron and insoluble contents of the different products show that, at the sizing used, there is no clean separation between ferruginous and silicate gangue, the magnetic work, however, has been useful in rejecting a substantial part of the iron-bearing portion.

c. Electrostatic treatment of non-magnetics. The electrostatic treatment was carried out in a manner similar to that detailed earlier with repeated passing of the tailings over the roll. Table 15 gives the results of the treatment applied to the sized fractions and to a composite bulk sample make up from the sized fractions. Table 16 gives similar results on the classified products.

Table 15

Electrostatic treatment of non-magnetic sized fraction

Test No.	Fraction	Product	% Wt.	U3O8 Assays %	U3O8 Distribution %
14	+36	Concentrate 1st	0.2	0.13	0.2
		" 2nd	44.2	0.16	43.8
		Mids	21.5	0.20	26.6
		Tails	34.1	0.14	29.5
		Composite	100.	0.16	100.

Table 15 Contd.

Test No.	Fraction	Product	% Wt.	U308 Assays %	U308 Distribution %
15	+52	Cons	20.0	0.20	9.5
		Mids	27.6	0.69	45.4
		Tails	52.4	0.36	45.0
		Composite	100.	0.42	99.9
16	+72	Cons	2.2	0.91	4.9
		Mids	6.5	0.46	7.4
		Tails	91.3	0.39	87.7
		Composite	100.	0.41	100.
17	+100	Cons	11.4	0.41	9.4
		Mids	21.0	0.44	18.6
		Tails	67.6	0.53	72.0
		Composite	100.	0.50	100.
18	-100 Coarse	Cons	2.8	1.00	3.8
		Mids	4.1	0.60	3.4
		Tails	93.1	0.73	92.8
		Composite	100.	0.73	100.
19	Composite all sizes	Concentrate 1st	0.8	1.27	2.3
		" 2nd	19.3	0.69	30.7
		Mids	24.5	0.44	24.8
		Tails	55.4	0.33	42.1
		Composite	100.	0.43	99.9

Table 16

Electrostatic treatment of non-magnetic classified fractions.

Test No.	Spigot No.	Product	% Wt.	U308 Assay %	U308 Distn. %
20	1	Cons	27.9	0.71	37.45
		Mids	29.4	0.53	29.45
		Tails	42.7	0.41	33.1
		Composite	100.	0.53	100.0
21	2	Cons	15.1	0.52	24.6
		Mids	16.0	0.47	23.55
		Tails	68.9	0.24	51.85
		Composite	100.	0.32	100.
22	3	Cons	5.3	1.48	22.1
		Mids	46.1	0.33	42.6
		Tails	48.6	0.26	35.4
		Composite	100.	0.36	100.
23	4	Cons 1st	1.3	0.96	3.2
		" 2nd	37.6	0.39	37.4
		Mids	8.6	0.33	7.2
		Tails	52.5	0.39	52.2
		Composite	100.	0.39	100.

All of the results in the above tabulations are very discouraging. The ratios of concentration are very low indeed and the grade of concentrates and recovery is altogether too low. Table 17 following shows the effect of a variation in the electrostatic conditions. In these cases a positively charged feed roller was used as formerly but the voltage was kept down in the region of 10,000. Under these conditions gangue minerals were deflected off the front of the roll purely by the motion of the roll and in a position governed by its speed. At this voltage and polarity there is a tendency for the uranium minerals to be held to the roll and removed by the wiper. Plus 36 and plus 52 non magnetic fractions were treated in this way the gangue product being re-passed several times. In table 17 the terminology has changed, in this case concentrate refers to the uranium bearing product carried to the back of the roll and tailing refers to the gangue thrown forward

Table 17

Electrostatic Treatment

Test No.	Product treated	Product	% Wt.	U_3O_8 Assay %	U_3O_8 Distn %
24	+36	Cons	5.65	0.84	31.05
	Non	Mids	9.85	0.30	19.3
	Mag	Tails	84.5	0.90	49.65
		Composite	100.	0.153	100.
25	+52	Cons	1.65	3.63	29.1
	Non	Mids	6.05	0.58	17.0
	Mag	Tails	92.3	0.12	53.9
		Composite	100.	0.206	100.

The above results are very much better than any of those shown in tables 15 and 16. In tests 24 and 25 uranium recoveries are around 50% in concentrates of 10% to 15% bulk. It would have been desirable to carry out more testing under the conditions applying to tests 24 and 25 but unfortunately, all original ore and sized products were exhausted.

d. Overall magnetic-electrostatic treatment. This has not been assessed for the East Painter ore as it was known that electrostatic conditions were not necessarily the optimum. For plus 36 and plus 52 mesh material overall recoveries are as follow.

Table 18

Magnetic-electrostatic recoveries

Size	Product	% Wt.	Assay %	Distn.%	Ratio of concentration
+36	Magnetic reject	29.05	0.08) 0.085	17.5	
	Electrostatic reject	59.9	0.09)	41.05	
	Cons and Mids	11.0	0.42	41.45	7.5
	Composite	99.95	0.13	100.	
+52	Magnetic Reject	46.25	0.1) 0.11	29.4	
	Electrostatic reject	49.6	0.12)	38.0	
	Cons and Mids	4.1	1.24	32.55	24.6
	Composite	99.95	0.156	99.95	

7. SUMMARY.

Two samples of ore from the Mt. Painter district were examined. The first sample was from No. 6 workings, it contained 0.35% of U_3O_8.

Crushing the ore caused the uranium minerals to concentrate in the finer sizes.

Preliminary magnetic treatment of the crushed and sized ore rejected 70.5% of magnetic material containing 0.15% of U_3O_8, this represents 30.5% of the total. Crusher fines which are too fine for the dry processing contained 0.91% of U_3O_8 or 23.6% of the total. This also would be lost in the proposed dry treatment making the overall crushing and magnetic loss 54.1%.

Electrostatic treatment of certain of the non magnetic products under the conditions detailed in the report were not very satisfactory. The work indicated that the ore should be crushed at least as fine as 25 mesh to secure adequate liberation of the uranium minerals. The electrostatic work also indicated that repeated retreatment of tailings was necessary in electrostatic work on this ore. The electrostatic recovery on the plus 52 mesh fractions was 80.2% in a 37.4% concentrate containing 2.44% of U_3O_8. Plus 72 mesh feed gave 76% recovery in a similar concentrate.

It is calculated that overall recovery of uranium by crushing to minus 25 mesh and rejection of lime, followed by magnetic and electrostatic treatment of sized fractions, would be around 28 to 29% in final concentrates containing 2% to 3% of U_3O_8. The testing and assumption on which this estimate is based are given in the report.

The second sample of ore was from East Mt. Painter. It contained 0.3% of U_3O_8. The sample contained more siliceous and less ferruginous gangue. Testing was carried out on minus 25 mesh ore split by classification and screening into various fractions. In this case also the uranium minerals concentrated in the finer fractions.

Magnetic treatment of the sized fractions rejected 44.2% of the gangue assaying 0.08% U_3O_8, a loss of 13.1%. Untreatable crusher fines raised the composite reject assay to 0.185% U_3O_8 and 49.7% by weight, a loss of 32.4% of the U_3O_8. Similar treatment of classified products rejected 45% of magnetics plus overflow containing 0.125% of U_3O_8 or 18.85% of the total.

Electrostatic treatment of the non magnetic portions was carried out under the conditions stated and with the results given in tables 15, 16 and 17. Under best conditions electrostatic recoveries ranged around 50%.

Overall magnetic-electrostatic treatment applied to the East Painter sample recovered from the minus 25 plus 36 fraction a total of 41.45% of the U_3O_8 in an 11% product by weight assaying 0.42% U_3O_8. In minus 36 plus 52 mesh material the recovery was 32.55% in a 4.1% concentrate containing 1.24% of U_3O_8.

8. CONCLUSIONS.

The work done on the ores was not successful in securing either high grade concentrates or high metal recoveries. Magnetic separation applied to ores of the type represented by the two samples submitted gave a useful rejection of ferruginous gangue. Electrostatic treatment, applied to non magnetics was only partially successful in concentrating the uranium minerals. It was felt that further exploration of electrostatic conditions could profitably be undertaken. However, any improvement in electrostatic conditions could only improve recoveries from that portion of the uranium mineral not already lost in magnetics and crusher fines.

14 June, 1945.

V11. THE RECOVERY OF URANIUM FROM THE TORBERNITE - AUTUNITE ORES OF MOUNT PAINTER, STH. AUSTRALIA

by

P. DIXON and T.R. SCOTT

Division of Industrial Chemistry
Council of Scientific and Industrial Research.

1. SUMMARY.

The importance of uranium as a source of atomic energy has stimulated an investigation into the possibility of recovering uranium from the autunite-torbernite ores of Mt. Painter, South Australia.

The poor grade of the ore and its location at a remote part of the State, required that every endeavour be made to develop some simple chemical process, whereby the uranium in the ore could be concentrated, or preferably extracted as pure uranium compounds, at or near the mine. By this means, transport of large quantities of low grade material over difficult country would be eliminated.

This paper deals with the development of a cyclic leaching process suited to the low grade ores (0.25 per cent U_3O_8) from Mt. Painter No.6 workings. The principal uranium mineral is torbernite (hydrated copper uranyl phosphate) which occurs in a brecciated matrix containing iron and manganese oxides with silica. The leaching agent is 5N (20 per cent) sulphuric acid and best results are obtained by leaching -1 inch ore, no preliminary concentration by conventional ore-dressing methods being required. Uranium is recovered from leach liquors by electrolytic reduction, followed by precipitation with hydrofluoric or oxalic acid. The residual liquors, after separation from the precipitate of uranous oxalate or fluoride, are used for leaching fresh lots of ore. The process is suited to the arid conditions and remote situation of the Mount Painter field, and gives overall recovery values of 65-70 per cent from an ore containing 0.25 per cent U_3O_8.

2. INTRODUCTION.

Of the numerous methods which have been in operation both in this and other countries for processing uranium ores, the majority have had as their primary objective the recovery of uranium.

Pitchblende ore such as that occurring at Great Bear Lake in Canada has been successfully treated by a process employing digestion with hydrochloric acid (1). Carnotite ores have been variously treated by leaching with hot hydrochloric, nitric or sulphuric acids (2). In some processes, the ore is first fused with soda ash while in others it is sintered with acid sulphate. The complex titaniferous ore from Olary, South Australia, has presented a peculiarly difficult problem and is discussed fully elsewhere (3). Both the torbernite and autunite ores from Mount Painter have been treated commercially over a limited period by the South Australian Radium Corporation which for the most part treated the ore with hot hydrochloric acid but also employed a hot solution of sodium carbonate as the extracting medium (3).

Owing to the nature of these extraction methods considerable amounts of iron and other impurities are dissolved along with the uranium and the subsequent recovery of uranium therefore involves several operations and somewhat elaborate chemical equipment.

In the present investigation, with the emphasis on uranium extraction, attention has been directed to those methods which achieve the greatest possible recovery of uranium from the ore. Radium recovery has not been entirely disregarded, but has remained subordinate to the main problem of uranium extraction.

The essential nature of the present problem was the extraction and concentration of uranium preferably at the mine, where, under arid conditions and in a remote situation, it became necessary to give due consideration to the associated factors of limited water supply and transport facilities, and the maintenance of chemical plant. With these factors in mind any extraction process had to be a fairly simple one using a minimum of complicated equipment. A process of selective leaching of the uranium mineral, preferably at air temperatures, seemed to be the ideal, with subsequent recovery of the uranium in a manner which would permit recycling the leaching agent and a minimum use of water.

The nature of the ore at Mt. Painter has been fully discussed elsewhere (4,5) and therefore only a brief description of the ores used in the present investigation will be given here. The investigations have been almost entirely confined to the leaching of ore from Mt. Painter No.6 workings. This material is a coarse breccia, impregnated with silica and containing considerable quantities of iron and manganese oxides. Torbernite ($Cu(UO_2)_2(PO_4)_2 \cdot 12H_2O$), meta-torbernite ($Cu(UO_2)_2(PO_4)_2 \cdot 8H_2O$) and autunite ($Ca(UO_2)_2(PO_4)_2 \cdot 8H_2O$) occur as surface encrustations or as flakes and platy crystals in cracks and joint planes of the gangue. Ores in a less complex matrix are found elsewhere in the area, but sufficient amounts were not available for extensive studies at the start of the investigation. The ore received at the laboratory consisted of lumps (3-6 inches), the uranium content varying from 0.11-0.25 percent U_3O_8, with torbernite as the major uranium mineral.

3. ANALYTICAL METHODS

A. Uranium.

At the outset it became evident that an investigation such as this would involve a very large number of uranium analyses and a simple and rapid method was essential. Gravimetric methods were slow and difficult, particularly when applied to solutions containing much iron and little uranium. To this end a colorimetric method based on the yellow color of uranyl solutions was devised and has been successfully used throughout this work in the analysis of ores and leach liquors. Since a full account of this method has been given elsewhere (6), it will suffice to give only a brief description here.

Uranium solutions of known concentration were prepared from A.R. $UO_2(NO_3)_2\ 6H_2O$, the uranium content of which was checked by ignition and weighing as U_3O_8. With these solutions a standard curve was obtained expressing the relation between color intensity and uranium content of the solution over the range 0.15 - 1.0 per cent U_3O_8, the curve being linear between 0.15 and 0.75 per cent U_3O_8. The color intensity is represented by the extinction (E), read directly on a Zeiss Pulfrich gradation photometer using cells of 20 mm. optical depth and employing specturm filter S 43 (filter gravity 4340 A°). Of the numerous contaminants found in leach liquors from Mount Painter No.6 ore, only copper and iron showed any tendency to interfere in the colorimetric method. Interference due to copper is entirely eliminated by the spectrum filter S.43 and the interference due to the yellow-brown color of small amounts of ferric sulphate is reduced to a negligible amount by the addition of up to 25 per cent of phosphoric acid (S.G.= 1.75). An empirically determined correction must be applied to the extinction in the presence of large amounts of iron.

B. Iron.

Iron determinations were likewise required in large numbers and a colorimetric method employing thioglycollic acid was used. In the presence of iron, thioglycollic acid produces a red-purple color which is unaffected by large amounts of phosphate and is directly proportional to the iron content over the range 2.5-7.5 mgm Fe/litre. To an aliquot of leach liquor containing a suitable weight of iron, thioglycollic acid solution (8 per cent thioglycollic acid in dilute ammonia) is added dropwise until a permanent red color is obtained. An additional 1.0 ml of thioglycollic acid solution is then added, followed by 10 ml. of 6 N ammonia and the solution made up to 100 ml. The resulting solution is then shaken vigorously and the color read immediately by the Zeiss Pulfrich photometer using the yellow filter S.57 (maximum transmission at 5720 A°).

When analyses of other contaminants were required, standard procedures were used.

4. SOLUBILITY OF TORBERNITE AND AUTUNITE.

It is known that both autunite and torbernite are soluble in mineral acids and concentrated alkali carbonate solutions. For the initial tests on solubility samples of high grade autunite (yellow) and torbernite (green) were obtained by hand-picking rich ore. Analyses showed these to contain 50.7 per cent and 50.5 per cent U_3O_8 compared with 61.6 per cent and 60.1 per cent respectively for pure minerals of the formula $R''(UO_2)_2(PO_4)_2\ 8H_2O$. Spectrographic analyses of these hand-picked samples, however, showed that the yellow mineral contained some copper, while the green mineral contained appreciable quantities of calcium.

Solubilities and rates of solution of the two minerals were determined in sulphuric, nitric and hydrochloric acids, at room temperature. At higher temperatures, the iron oxides present in the ore dissolve so readily that the solutions become heavily contaminated with iron salts. No tests were carried out with sodium carbonate solution which, evan at 100°C., is a less efficient solvent than the cold mineral acids.

A. Solubility in Acids.

An excess of the hand picked mineral was shaken with acids of various strengths until equilibrium was reached, five hours being more than sufficient for this purpose. The solvent powers of the acids were compared on the basis of normalities. The results are dhown graphically in Fig. 1.

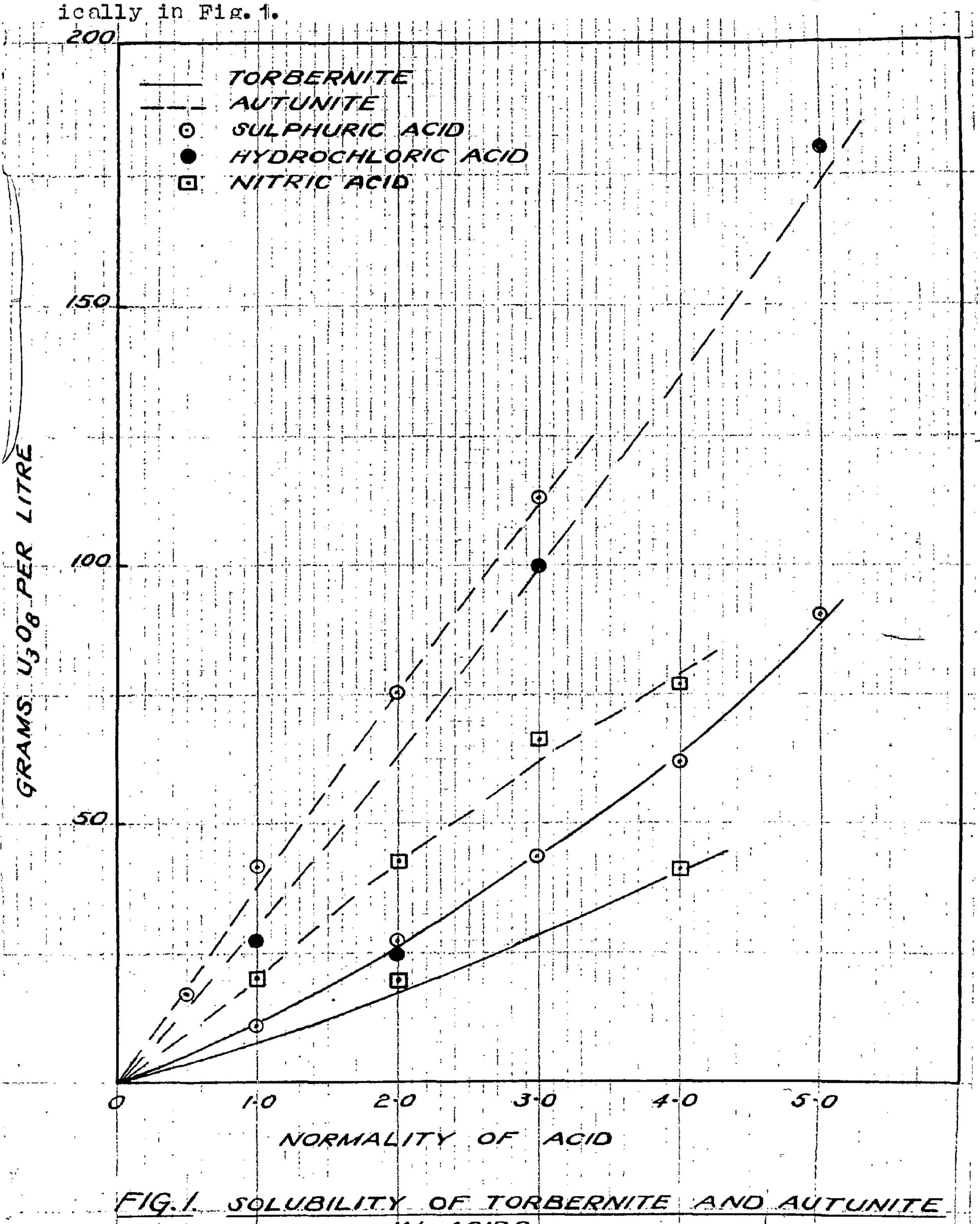


FIG. 1. SOLUBILITY OF TORBERNITE AND AUTUNITE IN ACIDS.

From these results the following conclusions may be drawn -

(i) Autunite is considerably more soluble than torbernite in all three acids.

(ii) Sulphuric acid is consistently better than the other two mineral acids, although hydrochloric acid is nearly as good.

(iii) Increase in strength of the acids does not generally result in important increases in the amount of uranium dissolved per equivalent of acid. The greatest increase occurs with sulphuric acid on torbernite where 1 N acid dissolves 12 per cent of the theoretically possible amount of mineral compared with 19.5 per cent for 5 N acid.

B. Mass Action Effects.

The usual reaction, viz.

$$3H_2SO_4 + R''(UO_2)_2(PO_4)_2 \rightleftharpoons 2H_3PO_4 + R''SO_4 + 2UO_2SO_4$$

represents an equilibrium and is thus subject to mass action effects.

When further large amounts of torbernite or autunite were added to the equilibrium mixtures and shaken for another five hours, a small increase in the amount of uranium in solution was generally observed (Table 1).

TABLE 1.

Acid	Mineral	Gm. U_3O_8/litre dissolved at equilibrium	
		With small excess	With large excess of mineral
2N H_2SO_4	Autunite	55	98
2N HNO_3	"	43.5	49.4
3N HCl	"	100	112
2N H_2SO_4	Torbernite	25.3	26.7
2N H_2SO_4	Torbernite + $Th(NO_3)_4$	(25.3)	50

With the exception of autunite in 2N sulphuric acid, these amounts were not sufficient to invalidate the results given in Fig. 1. In the case of autunite and sulphuric acid, the precipitation of calcium sulphate caused the equilibrium to be displaced to the right hand side of the above equation. This implies that the corresponding curve in Fig. 1 can have no pretentions to accuracy - it merely indicates that sulphuric acid is still a better leaching agent than the others.

Removal of phosphoric acid from solution by precipitation as insoluble phosphate should also affect the solubility of autunite and torbernite. The effect was demonstrated by adding 5 gm. of thorium nitrate to an equilibrium mixture of 2N sulphuric acid and excess torbernite, and shaking for five hours. By this means, the solubility of the torbernite was almost doubled (see Table 1). This phenomenon has no practical value, since the only phosphates insoluble in mineral acids are those of thorium, zirconium and tetravalent cerium.

If phosphoric acid be added to the solution instead of removed, a decrease in the amount of uranium in solution is noted; e.g. in the presence of seven times the usual amount of phosphoric acid, the solubility of torbernite was decreased by nearly 20 per cent. This decrease is not sufficient to affect significantly the efficiency of any leaching processes to be described, since such a large excess of phosphoric acid cannot accumulate in these processes.

C. Rate and Selectivity of Leaching.

Although no exact work was done on the rate of solution there is no doubt that autunite is more rapidly dissolved than is torbernite. Finely ground autunite dissolved to the maximum amount in less than one minute's shaking with 2N sulphuric acid, while the rate with the finest torbernite was always measurable. For the three mineral acids, the rate of solution of torbernite was approximately proportional to the solubility. Selectivity of leaching was surprisingly good for all acids. Although the samples of hand picked autunite and torbernite were sufficiently ferruginous to enable testing for iron contamination, in no instance did any of the solutions contain more than 1 part of iron to 100 parts of U_3O_8, and this obtained even when 7 gms. of finely ground ferruginous crude ore were shaken with a solution of autunite in 2N sulphuric acid for five hours. Hydrochloric acid at concentrations above 2N reacted in the cold with the manganese dioxide in the ore, producing noticeable quantities of chlorine, and causing contamination of the solutions with manganese.

5. LEACHING OF MOUNT PAINTER ORE.

A. Single Extraction Leaching.

(i) Choice of leaching agent.

From the foregoing study of the solution of the uranium minerals in the three mineral acids, it was evident that sulphuric acid was the most effective leaching agent. Other important factors favoring the use of sulphuric acid were: (a) its cheapness, (b) hydrochloric acid requires special corrosion - resisting equipment, and also reacts with the ore to produce chlorine, (c) nitric acid, besides being less effective than sulphuric acid, was found to be undesirable during the recovery of uranium compounds from the leach liquors.

It became clear at an early stage of this investigation that the method of leaching was to some extent influenced by the method used for the recovery of uranium from the leach liquor. It will be seen later that the method of recovery adopted necessitated that the acidity of leach liquors should be not less than 3.2 - 3.5 N and for this reason sulphuric acid of 4 - 5 N was chosen as the leaching agent. In Table 4 (p.9) the rates of leaching with various strengths of sulphuric acid are given, and lend support to the use of 4 N acid.

(ii) Methods of leaching.

These can be described under three heads:-

(a) Agitation leaching, where the ore and leaching agent may be placed in containers and subjected to rumbling, or where finely ground ore may be stirred. Such methods result in rapid leaching, but an unduly high proportion of iron is dissolved owing to the production of large amounts of very fine material by the grinding action.

(b) Circulation leaching, where the leaching agent is circulated continuously through the ore. In laboratory trials, liquor from the bottom of the leaching container was continuously removed and added to the top by means of an air-lift, but no significant improvement in the rate or completeness of leaching could be observed.

(c) Static leavhing, where the leaching agent is left in undisturbed contact with the ore. This method was used exclusively after the completion of the preliminary work described above.

(iii) Amount to leaching agent.

Because of the low grade of the ore, the amount of acid liquor was fixed at that required just to cover the ore sample. This volume of acid was extremely large in relation to the amount of uranium available in the ore sample and it therefore became imperative to leach more than one sample of ore with the same lot of acid in order not only to obtain a useful concentration of uranium in solution but also to make the maximum effective use of the acid. The extent to which this continued leaching could be carried out with the one lot of acid will be fully discussed later.

(iv) Analysis of ores and leached residues.

Even with the maximum care in sampling, it was felt that complete reliance could not be placed on analyses made on samples drawn from the relatively small lot (approx. 3 kg) of coarse ore used in the majority of leaching trials. The total uranium content of each lot of ore was therefore determined by adding the weight of undissolved uranium remaining in the ore to the weight of uranium present in solution in the leach liquor. The latter quantity was readily found by colorimetric analysis of the liquor but considerable difficulty was experienced in determining the former quantity by gravimetric analysis of the leached ore. It was found, however, that by thoroughly washing the leached ore with water, drying and grinding to a sufficiently fine size (-1/8 inch), all the residual uranium could be extracted by re-leaching the fine material with 4N sulphuric acid. This re-leaching was done for five days, at the end of which the uranium content of the leach liquor was determined colorimetrically. Assuming 80 per cent extraction in five days (Table 4), the uranium content of the leached ore could then be calculated. In this way, all sampling errors were eliminated. The method was checked in several cases by washing the re-leached ore, re-grinding to even smaller size, and leaching for a third time with acid.

The only uranium recovered in such instances was the 20 per cent unleached at the end of five days in the first re-leaching.

≠ On this assumption, values obtained for the uranium contents of leached ores are probably slightly greater than the true values, since, in the majority of instances, it is probable that somewhat more than 80 per cent. of the uranium is leached in five days. It should also be noted that, even if this figure varies from 75-85 per cent in different estimations, the corresponding variation in total uranium content of the original ore is only of the order of 2.0-2.5 per cent.

(v) Size of ore.

Factors which must be considered in determining the optimum size of ore for leaching include the extent to which grain size influences (a) the accessibility of the uranium content of the ore to leaching agents, (b), the retention of leach liquor by the ore, and (c) the rate of leaching.

(a) Accessibility of uranium minerals to leaching agents. Several lots of lump ore were leached until practically all visible uranium minerals had been dissolved. The total accessible uranium was then determined from the leach liquor, adding the amount represented by the small weight of visible torbernite which could be removed by hand from the lumps. The ore was washed, ground to pass 1 inch screen, and re-leached until the uranium content of the acid showed no further increase. The additional uranium which had been rendered accessible by reducing the size of ore was then determined in the leach liquor by the usual colorimetric method. The process was repeated on the same ore with further reduction of particle size.

The results are given in Table 2 and have been calculated on the assumption (see page 7) that practically all uranium is accessible in -1/8 inch ore.

TABLE 2.

Size of ore	Percentage of total uranium accessible %
Lump (3-6 inch)	42-50
- 1 inch	75-85
- 1/4 inch	90-96
- 1/8 inch	Approx. 100

(b) Retention of leach liquor. Since the aim throughout the investigation has been to devise a leaching procedure which produces liquors of a strength (1.5-2.0 per cent U_3O_8) suitable for subsequent recovery processes, it is evident that excessive quantities of wash liquor from the spent ore cannot be tolerated, for their concentration to a reasonable strength by evaporation would be costly and tedious. (The possible shortage of water in the Mount Painter area must also be taken into consideration). Moreover, when operating a cyclic leaching process (q.v.), it is essential to maintain a constant minimum volume and strength of acid in circulation, and this cannot be achieved with low grade ore (0.25 per cent U_3O_8) without restricting the volume of wash water to the lowest possible figure.

The volume of liquor retained by the ore after gravity draining is thus of great importance, since it has been necessary in laboratory trials to washwith the same volume of water to recover approximately half of the retained uranium salts. Figures showing the progressive increase in retention of liquor, as the particle size of the ore is reduced, are given in table 3.

TABLE 3.

Size of ore	Percentage of liquor retained when ore is just covered with liquid %	Volume of liquid required just to cover 1000 gm. ml.
Lump	6	320
- 1 inch	22-27	245
- 1/4 "	45	200
- 1/8 "	65	185
- 22 #	85	210
-100 #	> 90	240

(c) Rate of leaching. Pure torbernite dissolves more rapidly in acids when finely ground and a similar effect might be expected with finely-ground ore. Experience has shown, however, that although solution of iron from the ore is accelerated, the accessible uranium from -1/4 inch ore is dissolved less rapidly than that in - 1 inch ore. Table 4 shows this effect for various strengths of sulphuric acid.

TABLE 4.

	PERCENTAGE OF TOTAL ACCESSIBLE URANIUM LEACHED								
Acid Normality	-1" ore (7% - 14 #)				-1/4" ore (29% - 14 #				
Days of leaching	5.0	3.8	2.6	1.4	5.0	4.5	3.9	2.6	1.4
1	73	83	59	52	48	52	52	46	46
2	90	91	-	69	62	73	65	60	63
4	94	93	85	74	79	84	78	71	71
7	96	98	92	85	85	92	84	78	81
11	100	100	100	99	97	100	93	96	96
15				100	100		100	100	100
Wt. U3O8 leached (gm.)	4.06	4.47	4.57	4.21	2.68	3.29	3.20	3.15	4.47

The ores in question were carefully sampled by riffling and leached with equal volumes of acid until a constant concentration of uranium was obtained in the liquors. Leaching was static except for the agitation of the leach required to ensure proper mixing before drawing daily samples of the liquor.

The results show that the most suitable strength of acid lies in the range 3.8-5.0 N, though the decrease in rate of leaching with weaker acid is by no means as great as might be expected. For - 1 inch ores, it appears that a two-day leaching period is necessary to obtain a good recovery of uranium (90 per cent), but the same leaching period gives less than 70 per cent recovery with - 1/4 inch ores.

It is interesting to note the variation in the total amounts of uranium extracted from each lot of ore, despite the careful sampling. Since equal weights of the two sizes were taken, the lower uranium content of the -1/4 inch ore can hardly be ascribed to sampling errors, but is more probably due to loss of torbernite dust during grinding and riffling, particularly the latter. Measures were taken to minimise such losses when samples were being ground for analysis.

From the above considerations, it is evident that, although all the uranium present in -1/8 inch ore is accessible to leaching, when compared with coarser ore, the rate of leaching is slower, the degree of iron contamination in the liquor is greater, and the volume of leach liquor retained by the ore presents serious difficulties. These three disadvantages do not arise in the leaching of lump ore, but less than 50 per cent of the total uranium is accessible to leaching and the volume of acid required to cover a given weight of ore is disproportionately large. In consequence, up to five leaches with the one lot of acid are necessary to give solutions containing a reasonable amount of uranium. Moreover, although the leaching efficiency is reasonably good in the first two leaches of such a series, mass action effects appear in the remaining leaches, as shown by a serious decrease in the proportional of accessible uranium extracted by the leach liquor.

The optimum size of ore must therefore lie between the extremes just mentioned. More precise information as to the optimum size was obtained from leaching runs with -1 inch and -1/4 inch ores. In the latter instance (see page 16), the conclusions reached from Table 4 were borne out, only 84 per cent of the accessible uranium being leached. With -1 inch ore, however, due to the greater rate of leaching, a higher percentage of the accessible uranium was recovered, and this size of ore has therefore been taken as the optimum, although it is possible that the true optimum size of ore may be nearer to $-\frac{3}{4}$ inch. An alternative procedure would be to reduce the ore to at least 10 mesh, leach out nearly all the uranium and recover the rich liquor by centrifuging. Tests have shown that over 80% of the liquor can be recovered in this way and washing thus presents no particular difficulties. Only two leachings with the one lot of acid would be required to obtain 2% U_3O_8 solutions. Such a method would be particularly suitable for the treatment of enriched fine material produced by ore-dressing operations. Apart from the special apparatus required however, two new problems appear. Iron contamination becomes greater with such fine material and recovery of radium from the huge bulk of fines would be most laborious, if not impossible. By contrast, fines associated with -1 inch ore amount to only 7-8 per cent of the total material.

(vi) Selectivity of leaching.

In order to determine the nature and extent of contamination of the leach liquors, two liquors were prepared, one with 4N sulphuric and the other with 4N nitric acid. In both instances three lots of ore were leached, for one day each, without any dilution of the leachate by washing. The purpose of the nitric acid leach was to detect those metallic constituents having insoluble sulphates. Analyses are given in Table 5.

TABLE 5.

Gm./100 ml.	HNO_3 leach	H_2SO_4 leach
U_3O_8	2.16	3.45
Fe	0.52	0.88
Al	0.1	0.2-0.3
Mn	0.37	0.7
Rare Earths	0.3	$<$ 0.2
Mg	$<$ 0.05	$<$ 0.1
Ca	$<$ 0.1	-
V_2O_5	-	$<$ 0.01

The presence of rare earths (approx. 50 per cent CeO_2) was interesting and suggested that they might be derived from monazite. A sample of 98 per cent monazite (-100 #) left in contact with 4N nitric acid at room temperature for three weeks was observed to undergo partial solution. These results are in accord with the observations of Stillwell & Edwards(5) that finely disseminated monazite is present in the ore from No.6 workings at Mount Painter.

The following elements were not detected in either leach liquor:

Ag, Hg, Sn, As, Bi, Cd, Cr, Th, Zn, Ni, Co, Sr, Na, K. Small amounts of Pb, Ba and Ca were detected in the nitric acid leach but only Ca was present in measurable amount.

The virtual absence of arsenic and vanadium is noteworthy since these elements are frequently associated with uranium minerals.

Spectrographic examination of ore including handpicked specimens of autunite and torbernite and the solids obtained by evaporation of leach liquors to dryness, confirmed the above results, and tin was also detected.

From the results of the gravimetric analysis of the contaminants, it can be estimated that approximately 18 per cent of each lot of 4N sulphuric acid is used in dissolving the various metallic oxides in the course of the non-cyclic leaching scheme shown in Fig. 2 (see p.13). It is clear, however, that when the large preponderance of gangue material is taken into account, the leaching process is still quite selective with respect to uranium.

(vii) Reduction of acid strength during leaching.

Where three lots of ore were treated with one lot of 4N acid, it was found that each water washing diluted the acid liquor to such an extent that the leaching efficiency on the third lot of ore became greatly reduced. Reduction in acid strength from this cause can be eliminated by washing the ore with 4N acid instead of water.

Reduction in acid strength also occurs through solution of contaminants such as iron, manganese, aluminium, etc. The degree of this reduction in acidity has been determined by titrating samples of leach liquors with 6N alkali, the first permanent turbidity being taken as the end-point. In the presence of fluorides which subsequently react with the gangue during leaching to form fluosilicic acid, a precipitate of sodium silico-fluoride is formed during the titration and gives a false end-point. This error can be overcome by filtering the solution before completing the titration. The method was checked by titrating synthetic leach liquors of known acidity. Titration with decinormal alkali to various end-points (as shown by indicators) did not give satisfactory results, due to the very slow change in acidity in the region of pH 3-4. The titration technique is sufficiently accurate for control purposes but cannot be regarded as precise. By the above procedure the reduction in acidity of actual leach liquors from three lots of ore, where acid washing has been employed, amounts to about 0.7-1.0N. This value is slightly greater than can be explained on the basis of known contaminants. In order to recover leach liquors having a final free acid normality of 3.2-3.5N, it is therefore necessary to start leaching with 4.5N acid. Recycled acid (see p. 18) contains ferrous salts which react with the manganese dioxide in the ore, causing further contamination and requiring the acid at the start of each leaching cycle to be 5N.

B. Counter-current Leaching.

During "single extraction" leaching of Mt. Painter ore, it was shown (Table 4) that the rate of leaching decreased very considerably after the first day. In an attempt to overcome this difficulty, methods of "counter-current" leaching were devised, in which each lot of ore was leached with at least two lots of acid before being discarded. In this way, the mass action effect, which presumably slows down the rate of leaching in a single extraction leach, is very much reduced, since the uranium not dissolved by the first lot of acid is subsequently leached by fresh acid or acid containing only minor amounts of phosphoric acid and uranium salts.

The counter-current methods were tested by a series of leaches on -1 inch and -¼ inch ore.

(i) Counter-current leaching of -1 inch ore.

The scheme is shown in Fig. 2. In each cycle, four lots of ore are leached with one lot of acid, which remains in contact with each ore sample for one day only. Each lot of ore is leached twice, so that in one cycle two lots of fresh ore and two lots of previously leached ore are treated.

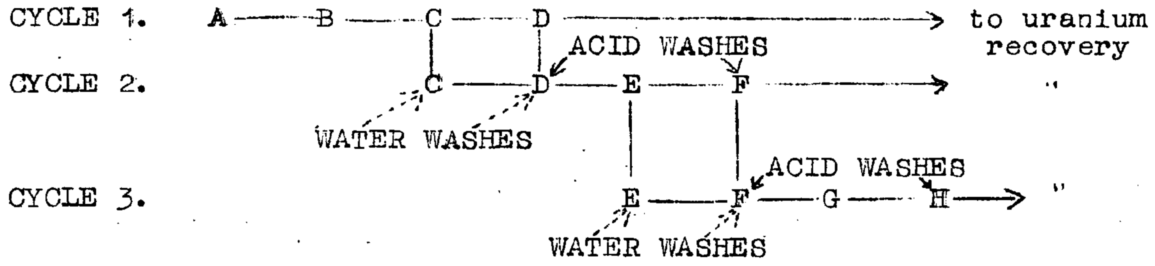


FIG. 2.

A description of the procedure for cycle 2 will serve to explain the method.

Ores C and D have been leached once, and drained but not washed. Each therefore retains approximately 25 per cent of acid liquor (Table 3). To C is added sufficient fresh acid to cover the ore completely. After leaching for one day, the liquor is drained from C and added to D, being again just sufficient to cover the ore. The liquor eventually drained from D is transferred to E, but since the latter has not been previously leached, this liquor is not sufficient to cover it entirely. Ore D is therefore given an acid wash, using a volume of acid equal to the average of that usually retained by these ores, and this liquor is also added to E. After leaching E, the liquor which can be drained off is added to F and the same volume of acid as was used in the previous wash is also added to F to ensure that the ore is covered. (Alternatively, E may be washed with this volume of acid before adding it to F). The liquor finally drained from F is filtered (or the suspended material allowed to settle) and treated to recover pure uranium compounds. Ores C and D are subsequently given a water wash, equal in volume to the acid wash mentioned above. These ores are then discarded and the combined water washes used to prepare fresh acid for the next leaching cycle. Ores E and F are not washed with water, since they are to be leached again in Cycle 3.

Samples of ore were prepared by passing two bags (approximately 100 lbs.) of lump ore through a jaw crusher until the material all passed 1 inch mesh. Since this procedure concentrates the uranium into the fines, the latter were separated by screening at 14 mesh, and both fine and coarse materials were divided into sixteen approximately equal samples by riffling. Samples were then weighed separately and each brought to the same weight (within 2 gm.). The fines and the coarse material were finally mixed to produce sixteen equal composite samples. This procedure was repeated on several occasions, the sample weight varying from 2700-3000 gm., and the percentage of fines from 7-9 per cent. To cover one sample, 750 ml. of 4-5N sulphuric acid were required, of which 150-200 ml. were retained by the ore after draining. Variation of the acidity within the limits given did not appear to influence the leaching efficiency.

The total amount of uranium remaining in the ores after leaching was determined as described on page 7. Uranium present in the final liquors and the water washings was determined colorimetrically. The amount lost by not washing the ore more thoroughly was calculated on the assumption, borne out by later washing experiments, that the uranium found in the wash is equal to that left in the liquor retained by the ore after one washing. Results are shown in Table 6.

TABLE 6.

Cycle	Weight of U_3O_8			Percentage of Total Uranium Leached. %	Percentage of Total Uranium Recovered. %
	In final liquor gm.	Recovered in washings gm.	Not leached gm.		
1	8.3	-	-	-	-
2	9.0	1.0	3.20	77.5	70.4
3	6.6	0.87	3.46	70.7	63.3
4	9.8	0.86	2.50	82.3	76.1
5	9.8	0.73	3.50	76.3	71.3
6	8.5	0.68	3.26	75.2	70.0
Average				76.4	70.2

Comparison of the average percentage of uranium leached (76.4 per cent) with the percentage range for accessible uranium in -1 inch ore (Table 2) shows that this method of leaching is a great improvement on single extraction leaching. The difference (6 per cent) between the percentage leached and that recovered represents the uranium leached but not recovered by the washing technique adopted.

As an indication of the improvements effected by acid washing and counter-current leaching, the results given in Table 7 are of interest.

TABLE 7.

Leaching Process	Duration of each leach. days.	Washing liquid	Lots of ore leached	Concentration of U_3O_8 in final liquor %	Percentage of total uranium leached %
Single extraction.	1	Water	3	1.70	56
"	2	"	3	1.82	62
"	1	Acid	3	1.48	64
Counter current	1	"	2	1.58	76

In the first two single extraction leaches, successive lots of ore were leached with the same lot of acid, keeping the volume of the latter constant by the addition of the water wash from the ore previously leached. The leaching efficiencies were low, even when the acid was left on each lot of ore for two days. Better results were obtained with acid washing combined with one-day leaching. In all the single extraction leaches, however, the third lot of ore was inefficiently leached, due in all probability to the mass action effect of uranium and phosphoric acid already in solution. That counter-current leaching overcomes this problem is shown by the greatly increased extraction of uranium, the concentration in solutions obtained from two lots of ore being nearly as great as that from three lots of ore leached by the single extraction method.

(ii) Counter-current leaching of -¼ inch ore.

It was possible that by applying some form of counter-current leaching to a more finely ground ore, advantage might be taken of the more complete availability of the uranium, at the same time overcoming the difficulties arising from high retention of leach liquor and consequent loss of uranium by incomplete washing and slower solution rate of the uranium mineral in such fine ore.

To this end a procedure was developed which required three lots of ore to be leached with the one lot of acid. One lot of fresh ore was to be introduced at the end of each cycle, each ore sample thus being leached three times instead of twice as in the scheme discussed for -1 inch ore. The effect of this procedure was also virtually to provide three washings for each ore sample. Acid washing was employed. A flow sheet of this system is given in Fig. 3 from which the operations can be seen in greater detail. For example, in cycle 3, sample C has had two leachings in previous cycles and sample D one leaching. Fresh 4N acid sufficient just to cover the ore is added to C which is leached for one day. The liquor drained off is added to D, and C then receives a water wash equal in volume to the liquor retained. After leaching D for one day, the liquor is drained off and D given an acid wash equal in volume to the retained liquor. The combined leachate and acid wash from D is then used for leaching a fresh ore sample E. The acid liquor is drained from E after a day's leaching and E given an acid wash. The combined leach liquor and acid wash from E is then ready for treatment to recover uranium. The water wash obtained from C is used in the preparation of fresh acid for the next cycle and the procedure repeated with D, E and F, F being a fresh ore sample.

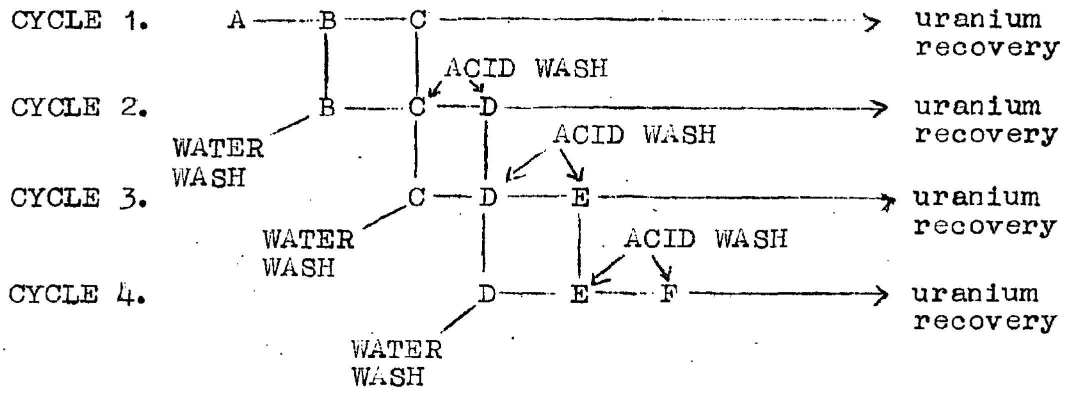


FIG. 3.

In following out this counter-current scheme, two bags of ore were crushed to -$\frac{1}{4}$ inch, and divided into sixteen approximately equal portions by the procedure which has already been described for -1 inch ore. The weights of all samples were finally adjusted by hand to 1480 gms. The above leaching procedure was then followed over five cycles, the 1480 gm. samples of ore being covered with 300 ml. 4N sulphuric acid. A careful check of the uranium extracted was made by colorimetric analyses of each leach liquor. Each ore sample as it was discarded from the leaching series was thoroughly washed with water and dried at 110°C. Fresh 4N acid was then put on the dried ore and releaching continued for a week to determine the amount of accessible uranium which had not been leached.

The results from this leaching programme are shown in Table 8.

TABLE 8.

Ore Sample	Total U_3O_8 in ore gm.	U_3O_8 accessible gm.	U_3O_8 leached gm.	U_3O_8 recovered gm.	Percentage of total U_3O_8	
					Leached %	Recovered %
C	3.05	2.90	2.60	2.25	85.2	73.8
D	3.23	3.07	2.75	2.30	85.1	71.2
E	3.16	3.00	2.55	2.07	80.7	65.5
F	3.98	3.78	3.32	2.57	83.4	64.6
G	3.20	3.04	2.69	2.09	84.1	65.3

The average leaching efficiency value of 83.7% is quite good, comparing favourably with the values given in Table 4 for single extraction leaching. However, the leaching recovery values are much lower and it is clearly seen that the theoretical advantages of this scheme are not realised in practice as a considerable amount of the uranium extracted is not recovered in the washing. This is due to the excessive and variable amounts of liquor retained by such fine ore, a feature which lends added weight to the arguments already set forth in favor of leaching -1 inch ore.

The method adopted for -1" ore cannot be applied satisfactorily to -$\frac{1}{4}$" ore since washing losses are greatly increased. On the other hand, the method for -$\frac{1}{4}$" ore gives solutions much poorer in uranium when applied to -1" ore, due to the greater volume of acid needed to cover -1 inch ore and the lower percentage of uranium which is accessible in such relatively coarse ore.

C. Cyclic Process with Counter-current Leaching.

Reference has already been made to the close dependence of leaching on the method adopted for recovery of uranium from the final leach liquors. This is particularly so in a cyclic process and the ability to recycle the acid liquor after recovery of the uranium either as the tetrafluoride or the oxalate depends on a number of factors. Of these it has been found that increase in both iron and fluosilicic acid content do not appear to have any adverse effect either on the recovery of the uranium or on electrolytic reduction except for the extra consumption of electrical energy required to reduce the iron. It was nevertheless reasonable to expect that the contaminants such as iron, manganese, etc. would ultimately build up in the recycled acid to an extent which would render further recycling unworkable. However a stage is reached when the amounts of iron, manganese, aluminium, phosphoric acid, etc. lost from the cycle by the necessarily incomplete washing of discarded ore become equal to the amounts of the same contaminants dissolved in one cycle. Continuation of the cyclic scheme beyond this stage would not result in any further increase of the contaminant concentration of the liquor and, provided that satisfactory leaching could be achieved with such contaminated acid, recycling could be continued indefinitely.

For any particular scheme of counter-current leaching using ore of constant size and having a constant retention value, it is possible to calculate the maximum contamination value for each element, if the rate at which contamination increases with each lot of ore is known. This figure has been repeatedly determined for iron in leaching with acid of different strengths and has been fixed at 0.12 gm. Fe per 100 ml. of 4.5N leach liquor per day, using -1 inch ore. It has been assumed that the concentrations of other contaminants (manganese, aluminium etc.) in the leach liquor bear a constant ratio to the iron concentration and this ratio has been determined by complete analysis of several leach liquors. Actually it has been observed in counter-current leaching that the rate of solution of manganese and aluminium does not remain constant but tends to fall off rapidly. The calculated maximum concentration of contaminants therefore gives a correct value for iron but a somewhat exaggerated figure for the other contaminants. It was felt that if such a "maximum contamination" liquor gave satisfactory leaching, the leach liquor of maximum contamination found in practice would give equally satisfactory results. Since it was calculated that approximately 20 cycles are required before the recycled liquor reached this maximum contamination value, it was difficult to check calculated contamination with actual figures, but over five cycles close agreement between theoretical and observed iron values was obtained.

For 4.5N acid on -1 inch ore, the liquor of maximum contamination is calculated to contain, in one litre, 18 gm. Fe, 11 Gm. Mn, 5 gm. Al and 24.5 gm. conc. H_3PO_4 (S.G. = 1.75). No account is taken of the other elements present in minor amounts in an actual leach liquor.

It was found in practice, however, that further contamination of recycled liquors occurs as a result of the oxidation of ferrous iron (produced by the electrolytic reduction) by manganese dioxide in the ore, according to the reaction,

$$MnO_2 + 2FeSO_4 + 2H_2SO_4 \longrightarrow Fe_2(SO_4)_3 + MnSO_4 + 2H_2O$$

Moreover, according to the method adopted for the precipitation of uranium from the reduced liquors, further contamination occurs. If the oxalate is precipitated, the excess of oxalic acid remaining in the liquor reacts with the manganese dioxide in the ore, according to the reaction,

$$MnO_2 + H_2C_2O_4 + H_2SO_4 \longrightarrow MnSO_4 + 2H_2O + 2CO_2$$

To allow for this increase in contamination, the amount of manganese in synthetic liquors of maximum contamination must be increased from 11 gm./litre to 44 gm./litre. If hydrofluoric acid is used to precipitate the uranium, excess of the reagent reacts with the silica in the ore to form fluosilicic acid or with the aluminium salts in the liquor to form complex fluosulphates. The maximum fluorine concentration attained during recycling can be calculated as described above and gives a value of 27 gm. fluorine per litre.

In preparing synthetic liquors, therefore, the following contaminants were added per litre :

88 gm. $FeSO_4, 7H_2O$

50 gm. $Al_2(SO_4)_3, 15H_2O$

170 gm. $MnSO_4, 4H_2O$

13.5 ml. H_3PO_4 (S.G. = 1.75),

and when hydrofluoric acid was used as precipitant, the appropriate amount of 35% hydrofluoric acid was also added.

The practicability of operating a cyclic process with such contaminated liquors was investigated by a series of leachings on -1 inch ore, using the counter-current method shown in Fig. 2. At the end of each cycle, the liquor was reduced electrolytically and uranium precipitated, as the oxalate in the first eight cycles, and as the fluoride in the next four. The filtrate from each precipitation was combined with the washings from the spent ore in the next cycle, and sufficient concentrated sulphuric acid was added to give a recycled acid of the desired normality, which was then used as the leaching agent in a subsequent cycle. Apart from these modifications, the techniques used and the quantities of material were the same as for the counter-current scheme described on page 13.

Results of this leaching program are summarized in Table 9. In the first four cycles, it was found that the ore was of much lower grade than usual and a fresh lot of hand-picked ore was used for the succeeding cycles. Moreover, it was only during the first four cycles that the extra contamination due to oxidation of ferrous iron and oxalic acid was noted, and the liquors of appropriately increased manganese content were consequently not used until the fifth cycle.

TABLE 9.

Ore		Leach Liquor		Precipitant	Percentage of total uranium	
Lots leached	Uranium content % U_3O_8	Contamination	Normality of free H_2SO_4		Leached %	Recovered %
A - H	0.113	Maximum	4.5	Oxalic acid	68	-
I - P	0.276	"	5.0	"	75	65
Q - V	"	"	4.5	Hydrofluoric acid	74	
W - X	"	Nil	4.5	"	81	72

From these results, it might be concluded that uncontaminated acid is a more efficient leaching agent than fully contaminated liquor, but if the result for ores W and X be considered in conjunction with the comparable results in Table 6, the difference between the two leaching agents becomes negligible.

For ores containing less than 0.20 per cent U_3O_8, this leaching technique is less satisfactory, since a reduced leaching efficiency is obtained and high yields of uranium oxalate or fluoride can only be obtained from the more dilute solutions produced, by using a much greater excess of precipitant.

Washing losses, represented by the differences between the percentages of uranium leached and recovered, are unavoidably higher in a cyclic process, since the liquor added to the first ore in each cycle is not fresh acid, but contains the uranium not precipitated from the previous leach liquor as well as that recovered in the wash liquors from the previous cycle. In non-cyclic leaching (Table 6), the latter quantity of uranium is all that is present in the acid used for each new cycle. In all forms of counter-current leaching, washing losses can be reduced by 1 to 2 per cent by adopting the alternative scheme of washing the third ore in each cycle with acid, instead of adding the acid directly to the fourth ore, as has been the practice in all experiments described here. If, instead, of washing spent ores (such as C and D, Fig. 2) separately, C is first washed, the drainings used to wash D, and the same process be repeated with a fresh lot of water, the total volume of washings is the same but only 33 per cent instead of 50 per cent of the uranium in the retained liquor is left unrecovered. Furthermore, in precipitations with oxalic acid, the volume of the liquor is decreased by evaporation during heating, and it may therefore be possible to increase the volume of wash liquor and still maintain the total volume of recycled liquor within the desired limits. It is claimed, therefore, that a 65-70 per cent overall recovery of uranium may be obtained in the cyclic process described above.

The merits of the various leaching processes which have been described will be discussed later in conjunction with the methods used for recovery of uranium from leach liquors.

6. RECOVERY OF URANIUM FROM LEACH LIQUORS.

In the past, it has been customary to recover uranium from acid solutions by the sodium carbonate process. To the acid liquor, sodium carbonate is added in sufficient amount to neutralize the acid and dissolve the uranium as sodium uranyl tricarbonate. The carbonates of iron, copper, etc. are precipitated. The solution is separated from the precipitate by decantation and filtration, acidified and boiled to remove carbon dioxide. By careful control of the acidity, sodium uranate may be precipitated at this stage; alternatively, if the solution be too acid, precipitation is effected by re-neutralizing, this time with sodium hydroxide. The precipitate is finally washed, filtered and dried. (Ammonium uranate may be prepared by an analogous method).

Theoretically the above process results in complete recovery of the uranium but it suffers from several serious disadvantages. One disadvantage lies in the nature of the two precipitates obtained. The first of these, a mixture of copper, iron and other carbonates, forms a bulky gelatinous mass which has to be very thoroughly washed in order to realise the theoretical recovery of the uranium. This implies generous use of water, which is not desirable under the conditions obtaining at Mount Painter.

The second precipitate (sodium uranate) is also gelatinous and presents a difficult filtration problem.

Perhaps the most important disadvantage is the complete neutralisation of the acid. Even under the optimum leaching conditions developed in this investigation, the amount of acid effectively used in dissolving the uranium minerals is but a small part of the total acid necessary for leaching. Consequently the adoption of the conventional recovery process would result in an excessive waste of acid.

For these reasons attention has been directed to the possibility of precipitating the uranium from strongly acid solutions, thereby facilitating the recycling of the acid liquor to subsequent leachings and also reducing to a minimum the use of water for washing purposes.

All salts of hexavalent uranium appear to be soluble in acids, but tetravalent uranium shows close analogies with thorium, in that the phosphate, oxalate, and fluoride are only slightly soluble in sulphuric acid. Moreover, these salts can be precipitated in a readily filterable form.

The process of uranium recovery can therefore be divided into two stages, reduction and precipitation, which will be dealt with in that order.

A. Reduction.

It is well known that reduction of hexavalent uranium (uranyl ion) to the tetravalent form can be effected in many ways, either by formic acid in the presence of sunlight, by reaction with a more electronegative metal such as magnesium, zinc or iron, or by electrolysis.

Reduction of uranyl sulphate solutions proceeds easily and quantitatively by either the metallic or electrolytic methods, such reduction actually forming the basis of a volumetric method for the estimation of uranium. However torbernite or autunite solutions contain an appreciable amount of phosphoric acid and reduction of uranium in such solutions leads to the precipitation of gelatinous uranous phosphate unless the acidity is maintained above a certain minimum value. As shown in Table 10, this critical acid concentration depends on the concentration of uranium.

TABLE 10.

Concentration % U_3O_8	0.5	0.75	1.0	1.5	2.0	3.0	4.0	6.0
Highest normality at which phosphate precipitates.	1.85	2.5	2.9	3.15	3.4	3.5	3.5	3.75

Increase in the concentration of phosphoric acid above that ratio normally existing in torbernite or autunite causes an increase in the critical acid normality. In a recycled liquor in which the phosphoric acid content has reached the maximum contamination level, the critical acid normality becomes greater by approximately 0.4N. Near the critical value however, precipitation of uranous phosphate is very slow and it is possible to reduce solutions having slightly less than the critical acid concentration without encountering any trouble. At the concentration of uranium found in actual leach liquors, i.e. 1.5-2 per cent U_3O_8, it is necessary to maintain the concentration of free acid at a value preferably greater than 3.25N in order to prevent concurrent precipitation of uranous phosphate during reduction. For this reason the use of 4 - 5N acid for initial leaching purposes is important. Admittedly the acid concentration of more weakly acid liquors could be adjusted prior to reduction but the use of stronger acid for leaching is justified in the more rapid extraction of the uranium from the ore.

(i) Metallic reduction.

Experiments showed that complete reduction of synthetic torbernite solutions (1.5 per cent U_3O_8) could be effected by a minimum of four equivalents of either zinc or iron.

In these and subsequent experiments on reduction and precipitation, synthetic torbernite solutions have been used. Such solutions contained uranium, copper and phosphoric acid in the proportions found in the mineral torbernite. The uranium was introduced as ammonium uranate dissolved in sulphuric acid and the copper as copper sulphate. These synthetic solutions consequently contained ammonium sulphate derived from the solution of ammonium uranate in sulphuric acid. Synthetic torbernite solutions made from the oxide, U_3O_8, dissolved in sulphuric acid showed no differences in behaviour from those solutions containing the ammonium sulphate and it therefore appeared that the ammonium salt had little effect on the results obtained from the latter solutions.

During metallic reduction a considerable amount of acid is consumed. Consequently, metallic reduction can be applied only to a batch process. However, precipitations of uranium tetrafluoride, oxalate or phosphate can be made successfully following reduction by iron. The large excess of iron in the solution after reduction does not cause any significant contamination of the product. Precipitation of uranium oxalate is unsatisfactory from solutions reduced with zinc since zinc oxalate contaminates the precipitate to an extent which reduces the uranium content from 50 per cent U_3O_8 to 30 per cent U_3O_8 or less.

(ii) Electrolytic reduction.

Electrolytic reduction although involving additional plant, possesses advantages which make it the only method applicable to a cyclic process.

The composition of the actual leach liquors introduces some difficulty in the interpretation of the results obtained, particularly in calculating current efficiencies for uranium reduction in the presence of other reducible salts such as ferric and copper sulphates. The reduction of both iron and copper has been assumed to occur in theoretical time and consequently the apparent current efficiencies for uranium reduction are in all probability lower than the actual values operating. Nevertheless the results permit definite conclusions to be drawn, particularly with regard to optimum current density.

The electrolyses were carried out in cells consisting of one litre lead beakers, the inner bottoms of which were coated with a clear lacquer. Porous pots were used to contain the anolyte. The walls of the lead beakers formed the cathode while a lead strip in the porous pots was used as the anode. The anolyte consisted of sulphuric acid of the same normality as the free acid in the catholyte. The cells had a catholyte capacity of 300 ml. and were kept cool by surrounding them with water in a large container. The course of the reduction was followed by removing a small aliquot of the catholyte and titrating the reduced uranium with 0.05N potassium permanganate, but in the presence of iron, titration with 0.04N ferric sulphate solution, using thiocyanate as indicator, was employed. This latter technique would form a valuable control method in any large scale operation. The titration with permanganate was straithfforward and excellent results were obtained provided the titrations were carried out rapidly. Slow titrations enabled air oxidation of the uranous solutions to introduce variable errors. Titration of reduced uranium with ferric sulphate solution was satisfactory when the aliquot of the catholyte was diluted to 100 ml. with hot water. In cold solution and in a small volume the oxidation by ferric ions was too slow to enable the titration to be performed.

In calculating current densities, an accurate measure of the cathodic surface could be obtained from the copper deposit resulting from the electrolysis of torbernite solutions, the upper edge of the deposit being sharply defined against the lead of the cathode.

Although the cathode was initially lead, the deposition of copper virtually resulted in the uranium being reduced at a copper cathode. This effect naturally lowers the current efficiency since the over-voltage for hydrogen is very much less at a copper than at a lead electrode.

The optimum current density was determined by electrolysing a synthetic torbernite solution (1.5 per cent U_3O_8 and 3.25N in free acid) over a range of current densities and measuring the current efficiency at 95 per cent reduction. For each value of current density a graph showing per cent reduction against time was prepared and the actual time for 95 per cent reduction read from the curve. Owing to the possible reduction of uranium to the trivalent stage the rate of reduction does not fall off as sharply when approaching the completion of the tetravalent stage as might otherwise be expected, and up to 95 per cent reduction the time required is always readily defined. Towards 100 per cent reduction the exact time is not definable with certainty and, for this reason, the current efficiencies have been determined at 95 per cent reduction.

From the plot of current efficiency against current density as shown in curve A, Fig. 4, it is seen that the best current efficiencies are obtained at low values of current density - not greater than 2 amps. per sq. ft. At such a current density, a current efficiency of 68 per cent obtains for the synthetic torbernite solution used.

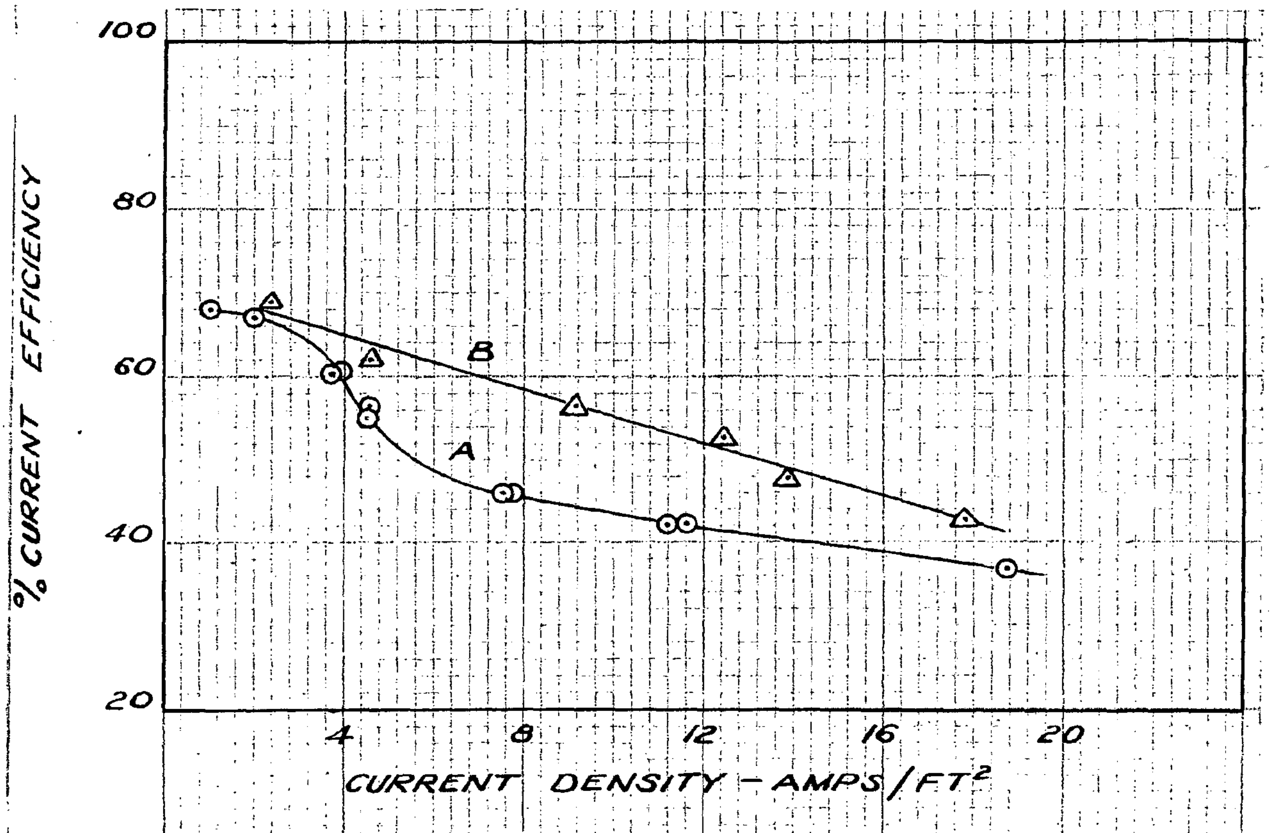


FIG. 4 VARIATION OF CURRENT EFFICIENCY WITH CURRENT DENSITY

The concentration of uranium has a pronounced influence on the current efficiency and this is clearly evident in Table 11, which shows the current efficiencies obtained with varying concentrations of uranium at a current density of 2 amps./sq.ft. A concentration greater than 2.8 per cent U_3O_8 was not employed since this value is not likely to be exceeded in liquors from low grade ores. From the same table it is seen that, except for the dilute solution, acidity has not such a marked effect as might be anticipated. The 2.8 per cent solution electrolysed in 3.25N free acid deposited phosphate, confirming the earlier statement regarding critical normality and this precipitation was probably responsible for the lower current efficiency recorded.

TABLE 11.

Concentration % U_3O_8	Free Acid Normality	Current efficiency (at 95 per cent reduction).
2.8	3.25	76 (deposited $U_3(PO_4)_4$)
2.8	4.00	80
2.8	5.00	81
1.4	3.25	68
1.4	4.00	70
1.4	5.00	70
0.93	3.25	47
0.93	4.00	57
0.93	5.00	61

Variation in cell design may have a beneficial effect on the operating efficiencies and a few electrolyses were carried out with a cell of different construction in which a lead sheet anode is covered with a closely fitting diaphragm of asbestos cloth impregnated with silica. In a cell of this type the volume of the anolyte is considerably reduced and the total electrical resistance of the cell becomes more nearly that of the catholyte which, because of the greater ionic concentration, is appreciably lower. In such a cell the energy efficiency for reduction is somewhat greater, e.g. the operating voltage for similar current density was 2.40 volts as against 2.55 volts for the porous pot type. Curve B in fig. 4 represents the plot of current efficiency against current density for this cell and it is seen that although the efficiency at low current densities is of the same order as from the first type of cell, the efficiency does not fall off as rapidly with increasing current density.

Observations on the individual effects of contaminants were limited to fluosilicic acid and iron, the latter being by far the most important. As stated previously, theoretical current efficiency had to be assumed for the reduction of iron as well as for copper and the current efficiency for uranium reduction under these conditions is based on the sum of the theoretical reduction times for all three elements. In Table 12 are the results from electrolytic reduction of synthetic torbernite solutions (3.25N in free acid) which contained the maximum contamination concentration of iron (1.8 per cent iron) and of fluosilicic acid (2.7 per cent fluorine). From these results it is seen that even with the assumptions made in calculating current efficiency referred to above, the actual current efficiencies are somewhat better than those obtained previously for the same concentration of uranium in synthetic torbernite solutions. Included in Table 12 is the result obtained for the reduction of a true leach liquor containing the maximum concentration of all contaminants; this shows good agreement with the other results.

TABLE 12.

Solution	Concentration % U_3O_8	Contamination	Current density amp./ft².	Current efficiency at 95 per cent reduction of uranium. %
Synthetic torbernite	1.31	1.8 per cent iron.	1.92	82
"	1.40	2.7 per cent fluorine (as H_2SiF_6)	1.92	75
Leach liquor ex V, table 9.	1.86	Maximum of all contaminants	1.92	83

The somewhat higher efficiencies obtained in the presence of contaminants appear to be due to the presence of iron. Even allowing theoretical efficiency for iron reduction, the efficiencies for the uranium in the presence of iron are higher than in corresponding solutions where iron is absent but other contaminants are present. Moreover, it has been observed that, with maximum contamination liquors, current efficiencies for uranium reduction of the order of 70 per cent are obtainable at very much higher current densities than that quoted as the optimum for synthetic torbernite solutions.

The presence of such large amounts of iron in actual leach liquors introduces a certain reduction in the electrical economy of the process since a large proportion of the electrical energy is used in each cycle to perform no useful work. Unfortunately the iron becomes reoxidised during passage of the leach liquor through the leaching cycle but there seems to be no means of overcoming this problem.

Some preliminary reductions were attempted using synthetic torbernite solutions in which the uranium was present as uranyl nitrate. With these solutions 100 per cent reduction could not be attained and the limited reduction possible was exceedingly slow. Such incomplete reduction was undoubtedly due to the oxidising character of the nitric acid present and it is essential to the success of the process developed in this work that nitrates or nitric acid be absent from the leach liquors. This aspect of the problem further emphasises the unsuitability of nitric acid as a leaching agent.

Stirring of the catholyte was not used during the above laboratory experiments but would doubtless be advantageous. Such refinements as stirring and variations in cell design are properly in the domain of pilot plant technique and it is sufficient to record here the orders of optimum current densities and corresponding current efficiencies.

B. Precipitation.

Four methods, involving the precipitation of sodium uranous fluoride, and uranous fluoride, oxalate and phosphate, have been investigated and are described in detail below.

(i) Sodium uranous fluoride.

When sodium fluoride is added to a solution of a uranous salt, a green precipitate of sodium uranous fluoride ($NaUF_5$, H_2O) is formed, the physical properties of which are characterized by rapid settling, which facilitates decantation of the mother liquor, and a granular form which permits rapid filtration. It was found in preliminary experiments with this compound that it could be precipitated from quite strongly acid solutions of uranyl sulphate and in moderately good yield.

The conditions for optimum yield of this compound were determined by precipitating the salt from a reduced synthetic torbernite solution by the addition of solid sodium fluoride.

The synthetic torbernite solution, 2 per cent U_3O_8 and 4N in free acid, was chosen of such concentration in uranium and free acid as to represent that of an average leach liquor. The addition of sodium fluoride in the solid form was necessary in order to avoid dilution of the acid liquor, an important factor in any cyclic process.

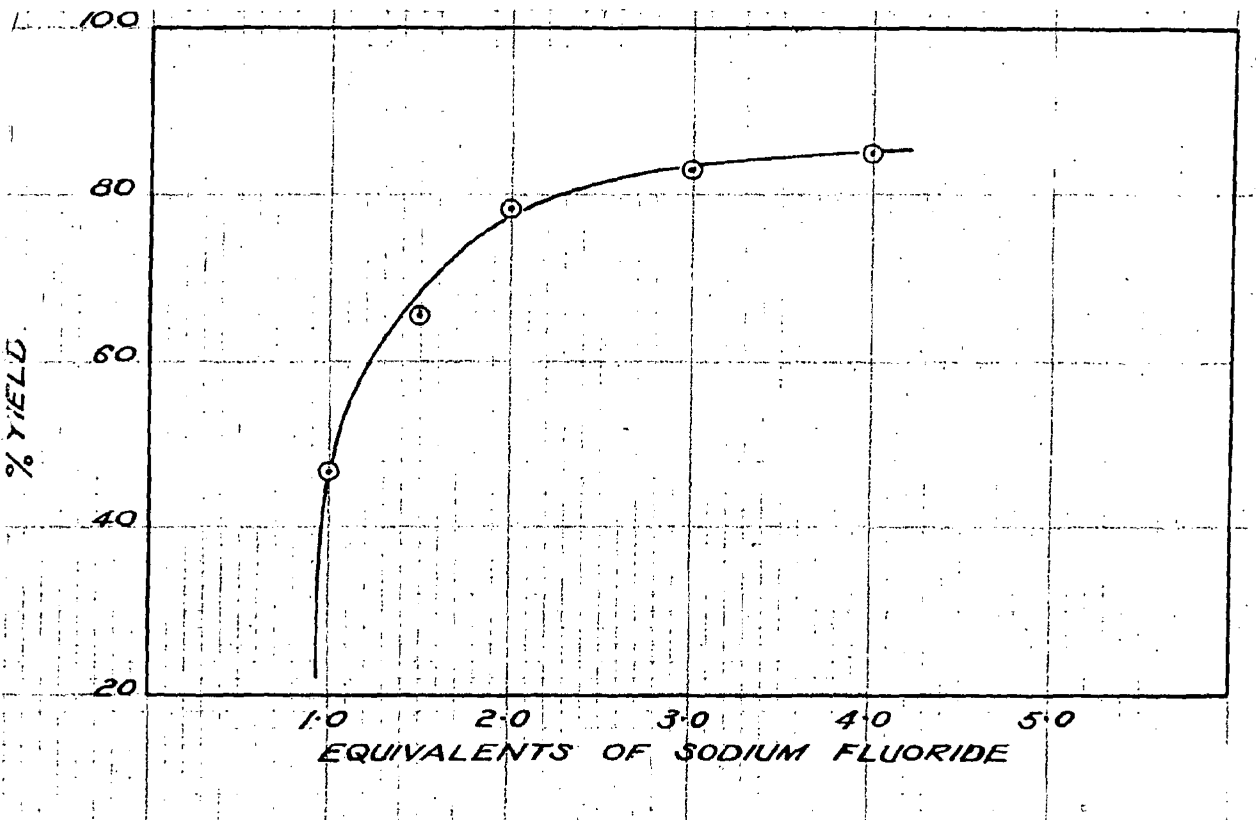


FIG.5 PRECIPITATION OF SODIUM URANIUM FLUORIDE

It was found that no precipitation of the complex fluoride occurred until at least one equivalent of sodium fluoride had been added, at which stage a yield of 47 per cent was obtained. As shown in Fig. 5, further additions of the precipitant increased the yield and also improved the physical properties of the precipitate. A yield of 84 per cent was obtained with three equivalents of sodium fluoride but more than this amount resulted in very little further increase in yield.

Other factors such as low acidity of the solution had a marked influence on the yield but since the merits of the recovery of uranium as this compound lay in the preservation of the acid liquor, no good purpose could be served by reducing the acid concentration in an attempt to improve yields.

When this method of recovery was applied to true leach liquors it was found that the product was contaminated with aluminium in the form of sodium aluminium fluoride. The extent of this contamination was such as to reduce the U_3O_8 content of the precipitate from 75 per cent to 35 per cent, although the percentage of aluminium was not large, being only about 6 per cent.

(ii) Uranium tetrafluoride.

In an endeavour to overcome the difficulty of co-precipitation of aluminium with uranium, hydrofluoric acid was substituted for sodium fluoride as the precipitating agent. A green precipitate of uranium tetrafluoride is thus obtained which, in the pure form, has the formula $UF_4.H_2O$. This compound also has the desirable property of quick settling which made sodium uranous fluoride initially so attractive. Furthermore, since aluminium fluoride remains soluble the product is substantially free from aluminium.

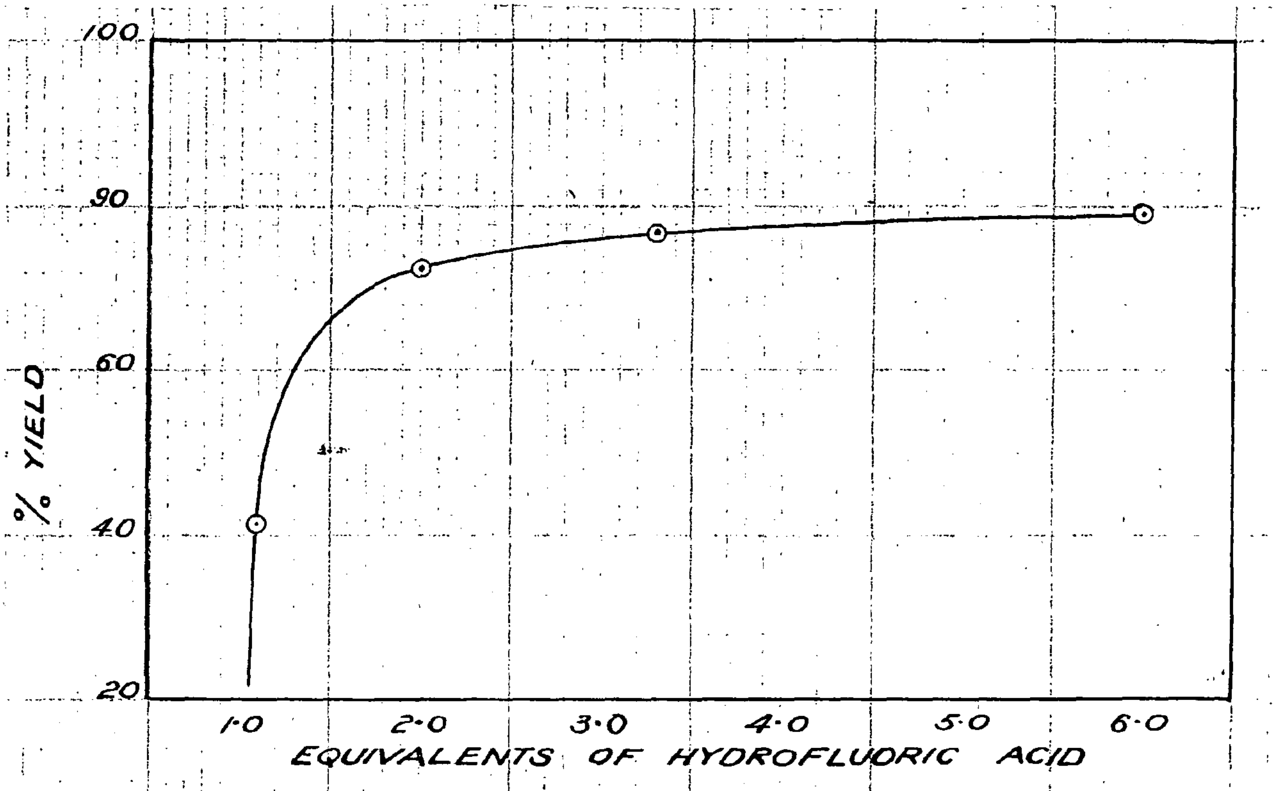


FIG. 6. PRECIPITATION OF URANIUM TETRAFLUORIDE

The conditions for optimum yield of uranous fluoride are almost identical with those obtaining for the complex sodium salt. It was found (Fig. 6) that 3 equivalents of hydrofluoric acid as the 38 per cent commercial grade were required to give the optimum yield. From synthetic torbernite solutions 2 per cent in U_3O_8 and 4N in free acid, the yield was 76 per cent, the uranium content of the product, expressed for convenience as U_3O_8, being 77 per cent. Acid normality appeared to exert little influence on yield, but variations in the concentration of uranium had a marked effect. This is shown in Table 13, which gives the yields, expressed in terms of U_3O_8, obtained from synthetic torbernite solutions over the range 0.93-2.8 per cent U_3O_8 and in 4 and 5N free acid, three equivalents of the precipitant being used.

TABLE 13.

Synthetic torbernite solution		Yield of UF_4H_2O from 200 ml. solution gm.	Uranium content of UF_4H_2O. % U_3O_8	Yield of uranium	
Concentration % U_3O_8	Free acid normality			gm. U_3O_8	%
2.8	4	5.52	77.8	4.29	76.7
1.4	5	2.53	77.4	1.96	73.6
1.4	4	2.62	77.3	2.02	73.7
0.93	5	1.57	77.4	1.21	65.0
0.93	4	1.48	80.5	1.19	64.0

From this it is evident that production of the more concentrated leach solutions is desirable particularly from the point of view of a cyclic process, otherwise too great a proportion of the uranium remains in the cycle with consequent increase in washing losses. This again emphasises the need for leaching an ore with low retention characteristics.

From actual leach liquors the recovery of uranium as the tetrafluoride is almost as good as from the synthetic torbernite solutions. From six leach liquors an average yield of 73 per cent was obtained, the average composition of the product being 72 per cent U_3O_8.

(iii) Uranous phosphate.

When synthetic torbernite solutions containing 1.5 per cent U_3O_8 are electrolytically reduced, gelatinous uranous phosphate ($U_3(PO_4)_4$) is precipitated unless the acid normality exceeds 3.15N. These precipitates, even when set to a gel, become crystalline on boiling or even on heating for an hour at 80-90°C. By reducing solutions of slightly greater acidity, no precipitation occurs in the cold, but the same crystalline precipitate is obtained on heating. The precipitate settles and filters extremely well, but analysis shows that considerable co-precipitation of sulphate ion occurs, the formula of the material approximating to $U_3(PO_4)_3(SO_4)_{1.5}$, $7H_2O$ after drying to constant weight at 110°C.

The phosphoric acid present in autunite and torbernite is only sufficient to combine with 75 per cent of the uranium when the latter is reduced to the tetravalent state (assuming that uranous orthophosphate is formed). Precipitations were therefore carried out in synthetic torbernite solutions (1.5 per cent U_3O_8) in the presence of up to four times the amount of phosphoric acid usually associated with uranium in torbernite. In all instances, solutions were heated rapidly to the boiling point, boiled for 1-2 minutes, cooled and filtered. From a colorimetric estimation of the uranium in the filtrate, the percentage precipitated could be determined. Results are shown in Table 14.

TABLE 14.

Normality of free sulphuric acid	Equivalents of phosphoric acid present	Percentage of uranium precipitated %
3.15	0.75	28
"	1.07	60
"	1.16	70
"	1.31	77
"	1.50	90
"	2.25	90
"	3.00	90
3.90	1.50	47
3.60	"	58
3.40	"	68
3.27	"	82
3.15	"	90
3.90	3.0	71
4.50	3.0	59

In this table, results are also given for the percentage of uranium precipitated when the acidity of the solution is varied. It is evident that the method is most useful when the acidity is controlled within rather narrow limits. Despite this handicap, the use of phosphoric acid as precipitant is worthy of further consideration, because it does not give rise to the additional contamination of leach liquors which occurs when hydrofluoric or oxalic acids are used. Moreover, according to the formula given above, the phosphate precipitates only contain about 75 per cent of the theoretical amount of phosphorus pentoxide. Hence, if 0.75 equivalents of phosphoric acid be added to a reduced leach liquor already containing 0.75 equivalents derived from the solution of torbernite, nearly 90 per cent of the uranium will be precipitated and the filtrate will still contain about 0.75 equivalents of phosphoric acid. By recycling this filtrate over a second lot of ore, a further 0.75 equivalents will be dissolved with the uranium and no further addition of reagent to the reduced leach liquor will be necessary, i.e. during each subsequent cycle, sufficient phosphoric acid will be leached to obviate the need for adding any further acid from extraneous sources.

Attempts were therefore made to apply this process to the precipitation of uranium from synthetic leach liquors containing the maximum quantity of contaminating salts. (It is important to note that such solution are approximately 30 per cent stronger than those prepared for leaching purposes (see p. 18), since the latter dissolve additional iron, manganese, etc. during the leaching of the ore, before they are finally drained off and treated for the recovery of uranium). Although the method employed was identical with that used for the earlier precipitations from synthetic torbernite solutions, yields were often below 25 per cent, and, even in the presence of four equivalents of phosphoric acid, did not exceed 60 per cent. Furthermore, great difficulty was experienced in obtaining reasonable agreement between replicate precipitations.

A study of the variables affecting the yield indicated that the contaminating salts present in solution tended to inhibit the precipitation of uranous phosphate and that only by controlling the rate and duration of heating could this effect be even partly overcome. Even with slow heating, and by holding the hot solution at temperatures close to the boiling point for several hours, the yield could not be consistently raised beyond 70-75 per cent. To achieve this, the addition of 3-4 equivalents of phosphoric acid was necessary. In a cyclic process this excess of phosphoric acid would build up sufficiently to require a higher normality of sulphuric acid during electrolytic reduction; hence careful control of acidity would be required to prevent the yield of phosphate falling below 70 per cent. The method is thus less attractive when applied to contaminated leach liquors, particularly as phosphoric acid must be added at the end of each leaching cycle in order to obtain the maximum yield of uranium.

(iv) Uranous oxalate.

Uranous oxalate, like thorium oxalate, is only slightly soluble in strongly acid solutions. The precipitate is bulky and gelatinous when formed in the cold, but becomes crystalline on heating, and is then readily washed and filtered. In view of the misleading results obtained in the phosphoric acid precipitation from synthetic torbernite solutions, practically all oxalic acid precipitations were done in synthetic leach liquors containing the maximum amount of contaminants, assuming the cyclic leaching scheme described on p. 16 et seq. to be operating.

Variables investigated included the technique of precipitation, acidity of solutions, number of equivalents of oxalic and phosphoric acids, concentration of uranium, and composition of the precipitate. These will be discussed in order.

(a) Technique of precipitation.

Although it was known from the outset that very good yields could be obtained with this method, great difficulty was experienced in obtaining reproducible results for comparative purposes until the conditions of precipitation were standardized. Despite the fact that the gelatinous oxalates, precipitated in the cold, can be made crystalline by heating soon after precipitation, there are indications that after ageing for an hour or more, the precipitate may remain non-crystalline even with prolonged heating. This danger was avoided by adopting, as standard procedure, the addition of granular oxalic acid to the cold solutions; under these circumstances, most of the oxalic acid does not dissolve until the solution is warm enough to produce a crystalline precipitate. In accord with this procedure, solutions were not stirred until the temperature reached 70-80°C. The final temperature and the rate of heating are also of importance, and slowly heated solutions which are finally boiled for a

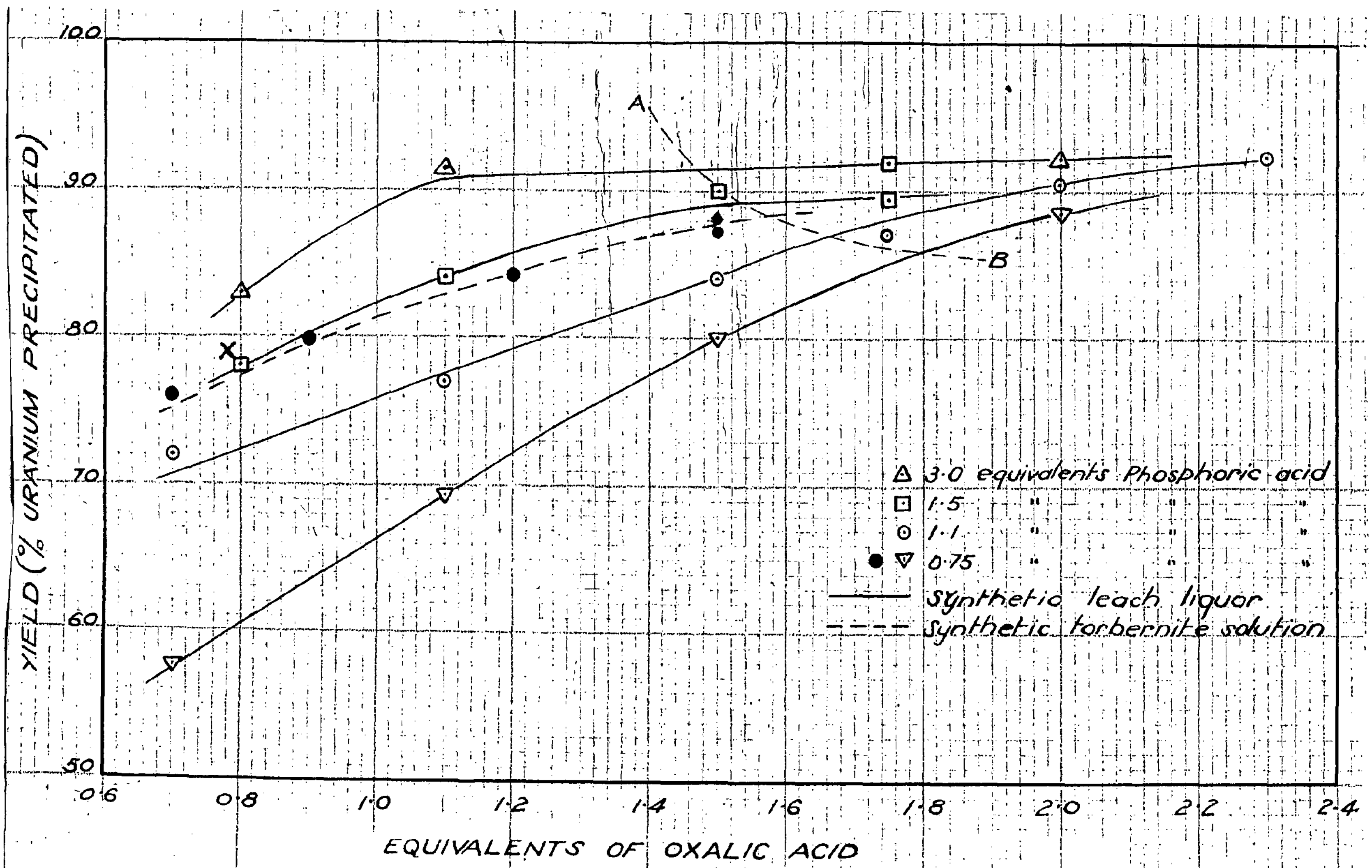


FIG. 7.

VARIATION OF YIELD WITH NUMBER OF EQUIVALENTS OF OXALIC AND PHOSPHORIC ACID few minutes (ca. 108°C.) may give up to 5 per cent higher yields than solutions heated to 95°C. All experimental precipitations were therefore controlled by heating a constant volume (100 ml.) of reduced synthetic leach liquor for 50-75 mins. on a hot plate, starting from the cold, and reaching a final temperature of 95-100°C. These conditions give a somewhat lower yield than may be obtained under optimum conditions, but this ensures that results, at least as good as those shown in Fig. 7, will be reproducible without difficulty in practice.

The oxalate precipitates are much more soluble in hot than in cold acid, and the hot solutions readily become supersaturated on cooling. The minor difficulties introduced by this phenomenon will be dealt with at a later stage.

(c) Acidity of solutions.

Yields from solutions containing 3.25N sulphuric acid differed by less than 2 per cent from those carried out in 5N acid. For convenience, most precipitations were done in 4N sulphuric acid.

(c) Equivalents of oxalic and phosphoric acid.

The effects of these two acids are so closely related that it was impossible to consider them separately. Precipitations were therefore made to cover the ranges 0.7-2.3 equivalents of oxalic acid and 0.75-3.0 equivalents of phosphoric acid, the maximum figure in the latter instance being greater than would ever occur in an actual cyclic leaching program, using oxalic acid for precipitating the uranium.

Since colorimetric estimations of the small amounts of uranium left in the heavily contaminated filtrates were unsatisfactory, the precipitates were filtered, washed with water, dried for two hours at 110°C., and weighed. The uranium content could then be determined colorimetrically in the solution obtained by dissolving a known weight of precipitate in nitric acid (which also oxidised the uranium) and diluting to the required volume with sulphuric acid. Results are shown graphically in Fig. 7, for solutions containing 1.5 per cent U_3O_8.

Although high yields were obtained with relatively small amounts of oxalic acid, the precipitations over the range 0.6-1.1 equivalents of oxalic acid were difficult to reproduce, particularly in the presence of large amounts of phosphoric acid, and the figures given must therefore be accepted with caution. Moreover, in this range, supersaturation troubles were so great that any method of precipitation used in practice should use at least 1.5 equivalents of oxalic acid. In some cases, seeding and vigorous stirring of the solutions did not appreciably increase the rate of crystallization of the oxalates and it was necessary to allow solutions to stand for several days before precipitation was completed. The solution corresponding to point X (Fig.7) was perfectly clear five hours after removal from the hot plate, despite continuous stirring. On the other hand, with solutions lying to the right of the dotted line AB (Fig. 7), no appreciable supersaturation is evident after the solutions have been stirred for two hours while cooling, and this is consequently the range in which it is most desirable to work.

Fig. 7 also shows the results obtained by precipitating uranous oxalate from reduced synthetic torbernite solutions. It is obvious that these results are of little value when applied to fully contaminated leach liquors.

(d) Concentration of uranium.

Fig. 8 indicates the variation in yield as the uranium concentration is varied over the range 0.75-3.0 per cent U308. Most precipitations were done in solutions containing 1.5 per cent U308, since this is the order of concentration obtained in actual leach liquors from 0.25 per cent ore using the cyclic method described on p. 16 et seq.

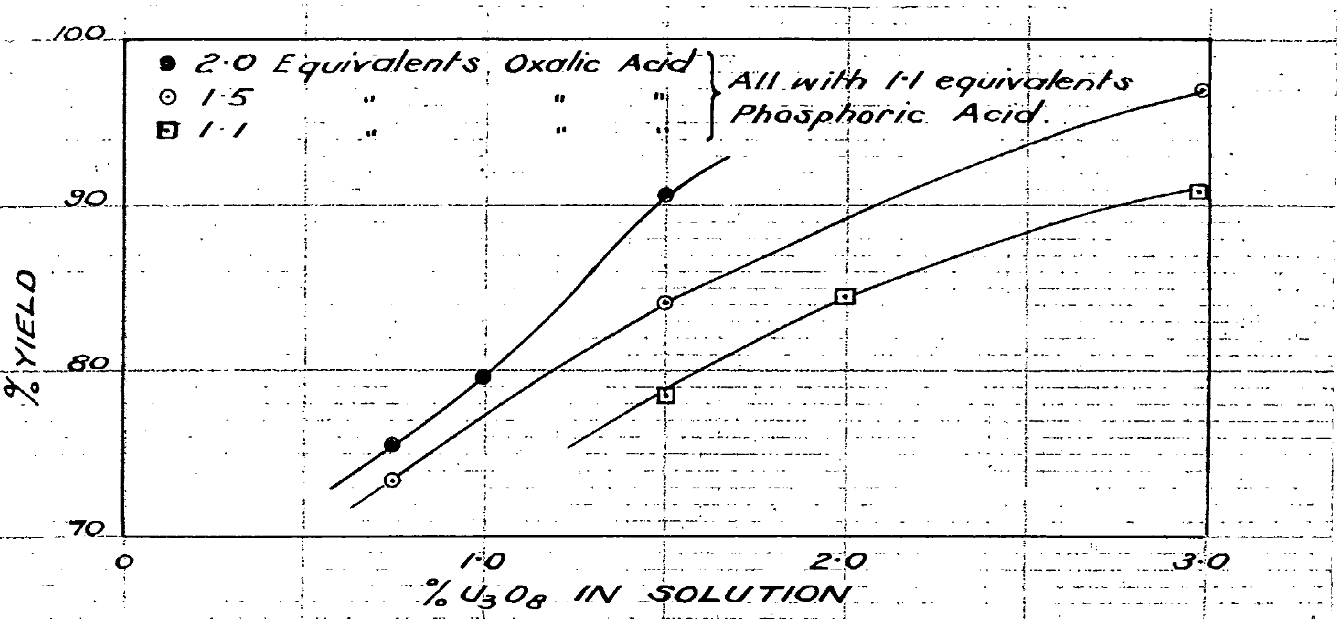


FIG. 8. VARIATION OF YIELD WITH URANIUM CONCENTRATION IN OXALIC ACID PRECIPITATION.

(e) Composition of precipitate.

Analysis of the oxalate precipitates, dried at 110°C., shows them to contain up to 13.5 per cent P2O5 and 2.4 per cent SO3, the amounts of these impurities varying with the circumstances of precipitation. Thorough washing of the precipitates with water does not remove appreciable quantities of these contaminants. The sulphate impurity is unimportant, but the amount of phosphoric acid combined in a precipitate affects the concentration of phosphoric acid left in the filtrate, and this in turn affects the maximum concentration obtainable in a cyclic leaching process. It thus became necessary to analyse a variety of precipitates in order to determine the magnitude of this effect, and the results are shown in Fig. 9.

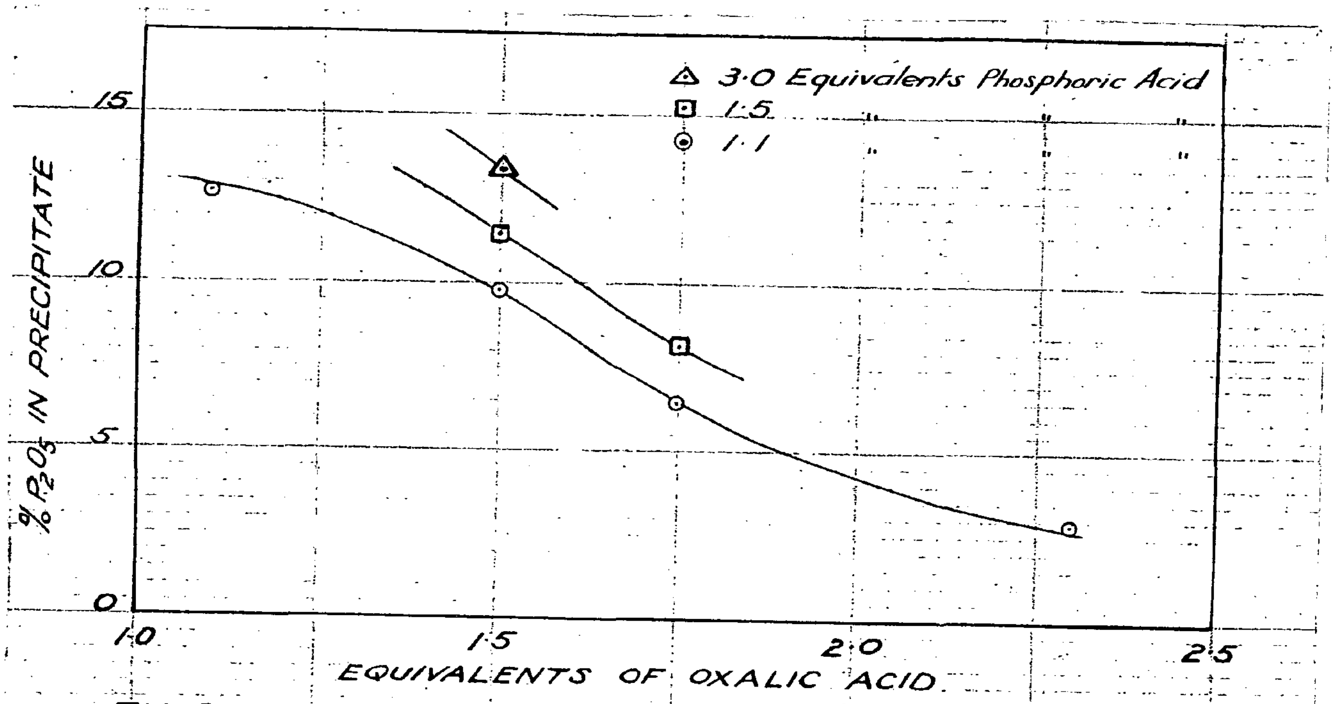


FIG. 9 PHOSPHATE CONTAMINATION OF OXALATE PRECIPITATES

From these results, and those of Fig. 7, it is possible to calculate that the maximum phosphoric acid contamination in the cyclic leaching of -1 inch ore, when precipitating with 1.5 equivalents of oxalic acid, will correspond to 1.1 equivalents of phosphoric acid (in leach liquors containing 1.5 per cent U_3O_8). If the precipitation be done with 1.75 equivalents of oxalic acid, the precipitate contains less phosphate impurity, more phosphoric acid remains in the filtrate, and the calculated maximum concentration is increased to nearly 1.5 equivalents. This leads to a higher yield, with no supersaturation troubles, at the expense of only 0.25 extra equivalents of oxalic acid.

1.75 equivalents of phosphoric acid is therefore taken as the optimum for solutions containing 1.5 per cent U_3O_8. With more concentrated solutions, the phosphate content of the precipitates is even smaller for the same number of equivalents of oxalic acid, yields are higher, and it is possible to get 90 per cent yields with considerably less oxalic acid, if desired. With weaker solutions (< 1 per cent U_3O_8), the reverse is true and the whole process is less satisfactory and more expensive.

7. THE RECOVERY OF RADIUM.

Throughout this investigation the possibility of radium recovery from Mount Painter torbernite and autunite ore has been subordinated to the recovery of uranium. The use of sulphuric acid as a leaching agent for the uranium has the effect of converting the associated radium to the highly insoluble radium sulphate. This salt may be recovered by washing the ore after leaching has been completed and collecting the fines which settle out from the wash liquors. The amount of washing required to recover a reasonable proportion of the radium has not been determined, but during the cyclic leaching process described on page 16 et seq. each lot of spent ore was washed at least three times with water. The fines so obtained, when combined with those which settled out from the leach liquors, were regarded as radium concentrates. From a total of 70 kg. of ore, 1.5 kg. of these concentrates was obtained. This represented 2.1% of the total ore, compared with 8% of fine material (-14 #) originally present.

For each per cent of U_3O_8 contained in unweathered torbernite ore, 2.35 milligrams of radium element are present per ton of ore. This is somewhat less than the equilibrium value and the amount actually recoverable would be considerably less. The methods for recovery of the radium, from the fines or sludge, in a relatively concentrated form present no serious difficulties and, if desired, the preliminary stages could be done in conjunction with the uranium recovery process.

At least two methods are available.

Hot concentrated sulphuric acid may be employed to dissolve out the radium sulphate from the sludge, the radium sulphate being then reprecipitated together with the sulphates of lead and barium on dilution.

Alternatively, the fines and sludge may be carbonated by boiling with sodium carbonate solution and the washed residue leached with dilute hydrochloric acid. Addition of dilute sulphuric acid to the filtrate will then precipitate the enriched insoluble sulphates of radium, barium and lead. In either case, the product obtained would be suitable for elaboration to high grade radium salts by subsequent laboratory manipulations.

8. DISCUSSION OF METHODS.

Of the leaching methods described in this paper, the cyclic process is preferred because it achieves a 65-70 per cent recovery of uranium from the ore while using the minimum amounts of sulphuric acid and water. The only outstanding disadvantage of this process is the extra electrical energy required to reduce all the iron in the leach liquor at the end of every cycle. Although this difficulty may be overcome by adopting a non-cyclic process, the consumption of acid and water is much greater and it is not possible to recover all the uranium from the final leach liquors unless the relatively unsatisfactory sodium carbonate process, described on p. 20, is adopted.

The same arguments favor the use of electrolytic reduction, since the simpler process using scrap iron as the reducing agent uses unduly large quantities of acid and contaminates the liquor with ferrous sulphate to such an extent that it cannot be satisfactorily recycled. Metallic zinc is even less satisfactory if oxalic acid be used as the precipitant, since large quantities of zinc oxalate are found in the uranous oxalate precipitate.

By conventional ore-dressing methods, it is possible to prepare concentrates from the low-grade ore which contain up to 85 per cent of the total uranium in the original material. Such enriched fines may be leached with much smaller volumes of acid, but the overall recovery of uranium is only slightly better than that achieved with -1 inch ore and, unless the leach liquor is removed by centrifuging, so much water is required to recover it that the original saving of water is almost entirely offset. The extra equipment required for this method does not therefore appear to be justifiable, unless the saving of water is the paramount consideration.

Of the methods of precipitation described at an earlier stage, only those using hydrofluoric or oxalic acid are regarded as satisfactory. In the phosphoric acid precipitation, the yield is not sufficiently high to justify the particularly careful control which is required, while the precipitate formed by adding sodium fluoride to reduced solutions is too heavily contaminated with cryolite (Na_3AlF_6) to be of much value. By comparison, the oxalate precipitate from liquors of maximum contamination contains 50-52 per cent U_3O_8 compared with 56 per cent for the precipitate from synthetic torbernite solutions. Uranous fluoride precipitated under the most unfavourable conditions still contains 70 per cent U_3O_8.

It is difficult to choose between the fluoride and oxalate precipitates. Much lower yields are obtained with the former (leading to increased washing losses in a cyclic process), and it tends to become gelatinous in the presence of contaminating salts, although it settles rapidly and washes well by decantation when precipitated from synthetic torbernite solutions. The amount of hydrofluoric acid required for this method is also more expensive than the required amount of oxalic acid. However, if it be desired to prepare metallic uranium, or uranium hexafluoride, from the compounds produced in the cyclic process, the fluoride appears to be the more suitable compound despite the disadvantages which have been mentioned, but up to the present it has not been possible to obtain information on this score. If the chemical nature of the compound produced by the cyclic process is of little significance, the oxalic acid method is preferred, since it produces a high yield of a precipitate which is easy to wash and filter. This conclusion takes account of the fact that it is necessary to heat solutions in the oxalic acid method, since this disadvantage is balanced by that arising from the need for special apparatus and precautions in a method using hydrofluoric acid.

9. CONCLUSIONS.

The low grade uranium ore from Mount Painter, No.6 workings, can be successfully treated by a cyclic, acid-leaching process.

The procedure developed is suited to mine conditions since no elaborate chemical equipment is involved and the water requirements of the process are relatively small. The only special items of equipment necessary are electrolytic cells and a source of direct-current voltage, and these could be installed and maintained at the mine with little difficulty.

After being crushed to -1 inch size the ore can be leached directly without any previous up-grading and an over-all yield of 65-70 per cent of the uranium content is readily obtained. Acid consumption is kept at a minimum by recycling the leach liquors after uranium has been removed.

The uranium is finally recovered either as the tetrafluoride or as the oxalate thereby representing a concentration of the uranium of between 200 and 300 : 1 from the low grade ore.

Concurrent recovery of radium is possible though in somewhat reduced yields.

10. ACKNOWLEDGEMENTS.

The authors are greatly indebted to Mr. R.G. Thomas of the Division of Industrial Chemistry, C.S.I.R., who, from his knowledge of the Mount Painter field has been able to give invaluable advice and criticism during this investigation.

Acknowledgment is also due to Mr. S.B. Dickinson, Director of Mines, South Australia, for the supply of Mount Painter ore used in this work.

11. REFERENCES.

(1) TRAILL, R.J. and McCLELLAND, W.R. - Canada: Dept. of Mines; Investigations in Ore Dressing amd Metallurgy, 138-148 (1931).

(2) BARKER, H.H. and SCHLUNDT, H. - Univ. Missouri Bull., 24: No.26, (1923).

(3) THOMAS, R.G. - Aust. Chem. Inst. J. & Proc., 9: 122 (1942).

(4) MAWSON, D. - Trans. Roy. Soc. South Aust. 68: 334 (1944).

(5) STILLWELL, F.L. and EDWARDS, A.B. - C.S.I.R. Mineragraphic Investigations Report No.315.

(6) SCOTT, T.R. and DIXON, P. - (in publication).

METHOD OF ASSAY ADOPTED FOR THE ESTIMATION OF URANIUM IN MOUNT PAINTER ORE

by

T. W. DALWOOD

Analyst and Assayer Department of Mines

South Australian School of Mines and Industries

Adelaide.

---

1. INTRODUCTION.

The uranium minerals commonly encountered in this area consist almost entirely of copper-uranium phosphates (torbernite and meta-torbernite) and calcium-uranium phosphate (autunite). Investigation showed that the ore contained, at most, only small traces of non-acid soluble uranium of no economic interest and the absence of an interfering amount of vanadium.

Assaying presented no particular problem, except that it was found necessary to purify the uranium phosphate which is generally accepted as the final product in the usual methods of assay. Small errors due to impurities in the final precipitate which would have little significance in ore of reasonable uranium content, became serious when dealing with the extremely low-grade material for which the modification was adopted.

2. PROCEDURE.

Boil from 0.5 gram to 5 grams of finely ground ore (according to the amount of uranium present) with 10 ml. of HNO_3 in a 150 ml. squat beaker. Evaporate to dryness but do not bake. Boil the residue with 5 ml. of HCl and 40 ml. of water. Filter into a 250 ml. beaker and wash with hot water to a bulk of approximately 100 ml. Warm the filtrate and pass H_2S until all the iron is reduced. Again warm and pass a steady stream of H_2S for not less than five minutes. Filter into a 300 ml. beaker and wash with H_2S water containing a little HCl. Boil off the H_2S from the filtrate. Oxidize the iron by adding a little $KClO_3$ and boiling for some minutes. Pour the solution into 50 ml. of hot 10% Na_2CO_3 solution contained in a 400 ml. beaker and boil for about ten minutes. Remove from the heat and add 5 grams of NH_4Cl. Stir and set aside without further heating for several hours (preferably overnight). Filter into a 600 ml. beaker and wash once with 2% $(NH_4)_2CO_3$ solution. Retain the filtrate. Transfer the filter paper and precipitate back into the 400 ml. beaker and dissolve the precipitate by heating with 50 ml. of 10% HCl. Pour the solution and pulp into 50 ml. of hot 10% Na_2CO_3 solution, boil for ten minutes and add 5 grams of NH_4Cl as before. Allow to stand without heating for not less than one hour. Filter into the filtrate from the first precipitation and wash five or six times with 2% $(NH_4)_2CO_3$ solution. Boil the filtrate for some minutes to decompose most of the $(NH_4)_2CO_3$. Slightly acidify the solution with HNO_3 and boil off the CO_2. Add microcosmic salt equal to not less than ten times the weight of uranium present. Add HNO_3 drop by drop until the precipitate formed is just dissolved. Now add 10 grams of $Na_2S_2O_3$ and 25 ml. of acetic acid. Boil for five minutes and add 10 ml. of 25% ammonium acetate solution. Continue the boiling for ten minutes and then allow to stand until the precipitate has settled. Filter and wash thoroughly with hot water. Ignite wet in a porcelain crucible, cautiously at first and finally over the full heat of a bunsen flame. Cool and moisten the residue with HNO_3. Carefully evaporate to dryness and then ignite over a bunsen flame.

It has been found that the uranium phosphate at this stage is invariably impure, being contaminated with small amounts of alumina, iron, etc. The following procedure for purification has proved satisfactory:-

Dissolve the phosphates in the crucible in a very little hot HCl and evaporate nearly to dryness. Add a few ml. of water and warm to dissolve. Wash into a 150 ml. beaker containing 2 grams of pure Na_2CO_3 using as little water as possible. Boil for a few minutes and add 10 ml. of 20% NH_4Cl solution. Disregard any precipitate. Pass H_2S through the solution for about five minutes and allow to stand without heating for at least one hour. Add a few ml. of strong $(NH_4)_2CO_3$ and then filter and wash with H_2S water containing 2% $(NH_4)_2CO_3$. Acidify the filtrate with HCl and boil off CO_2 and H_2S. Add about 1 ml. of HNO_3 and boil well to oxidize. Add 2 ml. of microcosmic salt solution (41 grams per litre), 2 drops of methyl red indicator, then NH_4OH drop by drop to neutral point and 2 drops in excess. Add a little filter pulp and boil for two minutes. Allow to stand warm for about one hour. Add 1 drop of NH_4OH and filter and wash with 2% NH_4NO_3 solution made just ammoniacal. Ignite wet in a weighed porcelain crucible cautiously at first and finally over the full heat of a bunsen flame. Cool, add a little HNO_3 and evaporate to dryness. Ignite, cool in a desiccator and weigh as $(UO_2)_2P_2O_7$. Weight of $(UO_2)_2P_2O_7$ multiplied by 0.7863 equals weight of U_3O_8 found.

PART VI - MISCELLANEOUS REPORTS.

I. PRELIMINARY REPORT ON SPECTROGRAPHIC ANALYSES OF URANIFEROUS ORES.

by
PROFESSOR KERR GRANT and K. B. MATHER,
Physics Department, University of Adelaide.

This report summarises work carried out in the Physics Department of the University of Adelaide, in connection with the spectrographic estimation of uranium in its ores.

The uranium spectrum is notoriously rich in lines, but few persist from ores containing less than 0.5% uranium.

Preliminary analyses using chemically analysed ores from Mount Painter and elsewhere (supplied by Mr. Dalwood, Analyst & Assayer, South Australian School of Mines) indicated that the electric D.C. arc between graphite electrodes would permit detection of about 0.1% of uranium. However, the presence of a fully developed iron spectrum added considerable difficulty, with the spectragraph used. Masking of important uranium lines by iron lines frequently occurred.

Estimates of the most intense arc lines recorded between 5,000 and 2,500 A.U. contradicted the published "Raies Ultimes" of Pollok and Leonard (who used spark methods).

It was therefore decided, at this stage, to undertake a complete examination of the uranium spectrum in respect to true line intensities, line sensitivities to concentration, and interferences of uranium lines with lines of elements present in typical ores. This work aims at determining the essential data required for any spectrographic analyses of uranium. It is not yet complete.

A complementary aspect has been the development of a suitable instrumental technique for such analyses. In summary it may be said that:-

(i) of the spectrograph itself, only large instruments are suitable. A dispersion of 5 A.U./mm. can suffice.
(ii) the uranium ore spectrum is considerably enhanced in the vicinity of the cathode of the D.C. arc. The so-called "cathode layer" method is therefore to be advocated.
(iii) the most useful spectral range is 4500 - 3000 A.U.

Little work has been done on actual analyses beyond the preliminary experiments. However, the difficulty of 'sampling' has arisen. It is likely to contribute the major error. (The 'cathode layer' method is most sensitive when some 10 mgm sample is employed)

Eye estimates of the reproducibility of spectra so far recorded, considered in conjunction with published estimates of accuracy in similar problems suggest an accuracy certainly better than 20% of the value at a concentration of 0.1% uranium.

II. NOTES ON THE PHOTOGRAPHIC ACTION OF RADIO-ACTIVE MATERIALS.

by
PROFESSOR KERR GRANT and K. B. MATHER,
Physics Department, University of Adelaide.

Some preliminary work here has suggested that use might be made of the fogging of photographic plates by gamma-rays.

Heavy, double emulsion, X-ray films have been used to give maximum absorption, and samples of Mt. Painter ores placed in contact with the film. The ores must be finely ground.

Exposures of 2 weeks recorded photographic fogging by samples reported as "below 0.1%" by chemical assay. The exposed portion of the film, on development, reveals localised points of activation, together with a general background fog for the higher concentrations. The most promising quantitative method would appear to consist of counting the number of points of activity in a definite field, samples to be compared being ground to similar consistency.

The method, if developed, would suffer from the disadvantages attendant upon use of the gamma-emission as a measure of uranium content, (as in Geiger counters, etc.) However, virtually no apparatus is required.

Dated 30/10/45.

III. TOPOGRAPHIC MAP OF MT. PAINTER AREA, SOUTH AUSTRALIA.

REPORT ON COMPILATION METHODS USED.

BY

A. C. BOOTH

DEPARTMENT OF THE INTERIOR.

1. INTRODUCTION:

A survey map of the Mount Painter area on a scale of 20 chains to an inch with contour form lines at intervals of about 250 feet was compiled in the Commonwealth Department of the Interior, Canberra. It is presented in four (4) sheets (plans Nos. 3288, 3289, 3290, 3291, Sheets 45A, 45B, 45C, 45D) and covers an area of about 154 square miles (11 miles by 14 miles). It was prepared from aerial photographs taken by the R.A.A.F. and from a skeleton ground control survey made by Mr. A.A.H. Davison of the Department of Lands, Adelaide.

The compilation of the map was an extremely tedious operation on account of the inadequate number (25) of ground control stations and their somewhat awkward distribution for nearly 500 small photographs (5 inches square). There were some runs of 30 photographs each with no control points at all between the terminals. The lack of ground control points also limited the expressions of topographic relief to form lines at approximately 250 feet intervals.

The following procedure was designed essentially to cope with these difficulties.

2. PROCEDURE:

The plotting of principal point traverses and their transfer at common scale to the master compilation board was quite orthodox. During the subsequent process of adjustment, where ground control stations were missing it was necessary to use join points derived from adjacent runs in the vicinity of control stations.

The determination of useful form lines was a more doubtful undertaking than the integration of photographs from the point of view of the work involved and the prospects of success. The basis of the procedure that was followed was to assume that the aircraft would be comparatively stable for a short time in the direction of its line of flight although it would certainly have varying lateral tilts. Paralla measurements were therefore decided on along the flight line at points close to each principal point. The levels thus determined were to be adjusted between control points at each end of a short section of the run. However the runs were not all broken up into sections by intermediate trigs. and cross traverses of parallax measurements were necessary between trigs., (e.g. Mt. Pitt to N.L.P.) to provide such points. These cross traverses took in points on the lateral overlaps of adjacent runs and an arbitrary adjustment for misclose between the levels of the trigs. had to be adopted at this stage.

Weak as this arrangement was it was considered better than adopting an arbitrary adjustment along the flight line over an excessive number of photographs.

A section containing up to ten pictures appeared to be the maximum acceptable.

The selection of parallax points on the photographs was made on the basis that two high and two low points per overlap should suffice for form lining at intervals of 250 feet. These points might or might not include the centre line points which provided the levelling traverse along the flight line. In general one or two of the side points were common with the adjacent overlap in the same run or in the adjoining run thus providing lock points for the subsequent general levelling adjustments. All selected levelling points were then shown with their identification numbers on paper tracings of the master compilation sheet which also showed the principal point traverses. Trig. stations were shown with their elevations and also the rough courses of the main streams.

The sequence of level adjustment was as follows:-

The gradient at Arkaroola Creek was obtained by determining its level at points near trigs. H, F, D, A, & C, by reference to the elevations of those trigs. In this way it was found that the elevatio of point A by survey appeared to be about 400 feet. too high and it was therefore abandoned. A gradient was adopted of about 40 ft. to the mile for Arkaroola Creek and this served as a guide later to estimating the gradient of other creeks.

As the parallax measurements were completed the cross traverses were run out and adjusted.

Then the sections of the centre line traverses between cross traverses or between trigs. were also run out and adjusted. These adjusted values in the vicinity of each principal point served as a datum on each overlap to give a value to the side points. All values were then entered against the corresponding level point on the paper tracing of the master compilation.

Points appearing on more than one overlap showed a value derived from each of the overlaps. These values were then studied in conjunction with a steroscopic examination of the photographs so that a set of single values could be arrived at which would be consistent with the visible topographical form.

Where the values on the margin of a run required depressing they were often accompanied by a corresponding rise on the other margin. Where both margins required depressing or raising it suggested that the centre line traverse was too high or too low and an adjustment of this nature was investigated.

At this stage the values of points along the main creeks were compared with the estimated gradients of the same creeks and discrepancies were eliminated. This provided a reasonable measure of control where no control existed.

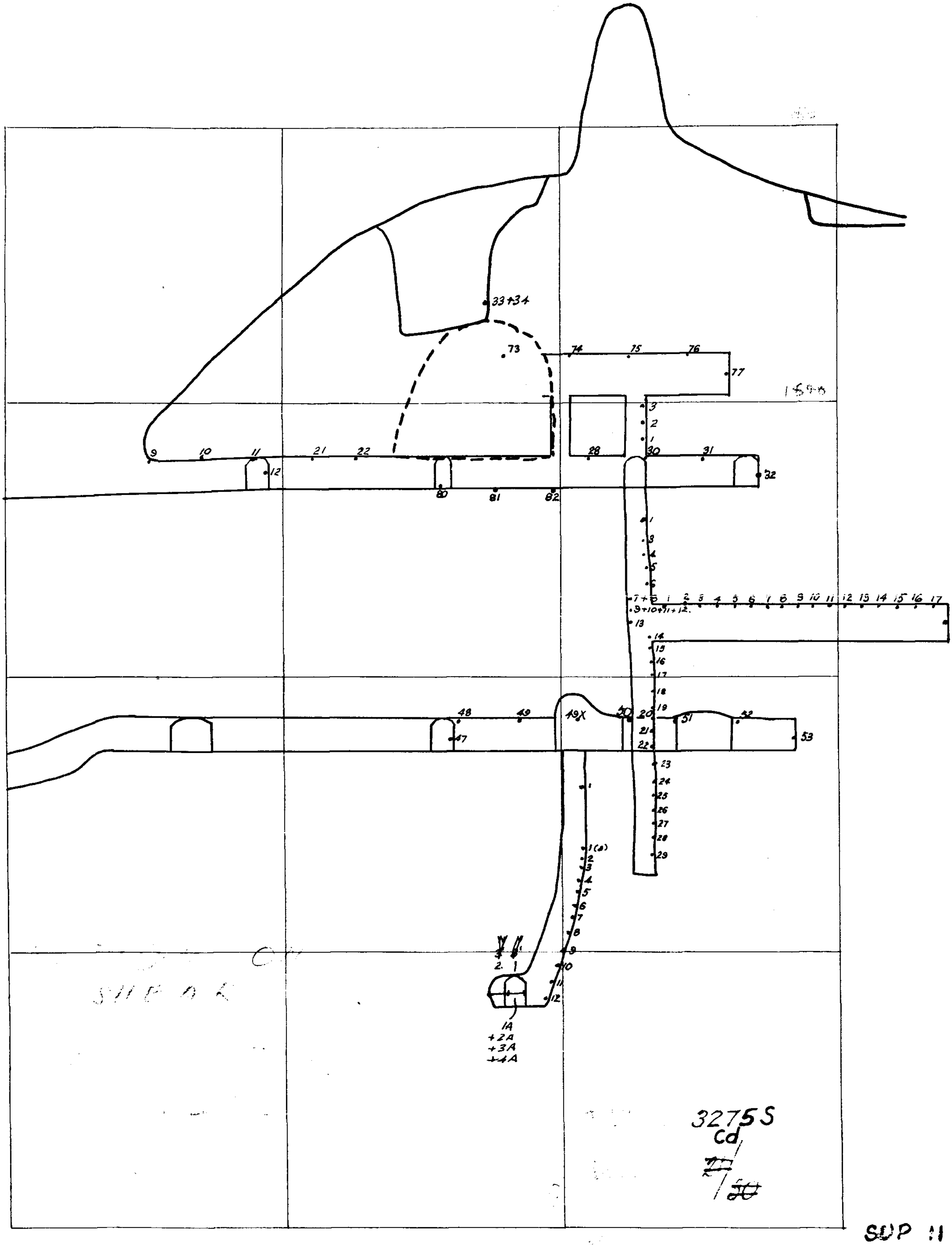
The single set of elevations for all levelling points was finally transferred to the photographs and the "controlled" form lines interpolated under the steroscope.

3. CONCLUSION:

It is considered that the form lines have a good chance anywhere of being correct within the value of the contour interval. The value of the interval has been maintained fairly well and over a small area the horizontal plane will be well enough defined so that interpolations at closer intervals for geological purposes should have considerable value despite a possible departure from the true datum.

In the final stage of plotting the detail from the photographs some slight local adjustments were made to eliminate discrepancies remaining in the compilation of the principal point traverses.

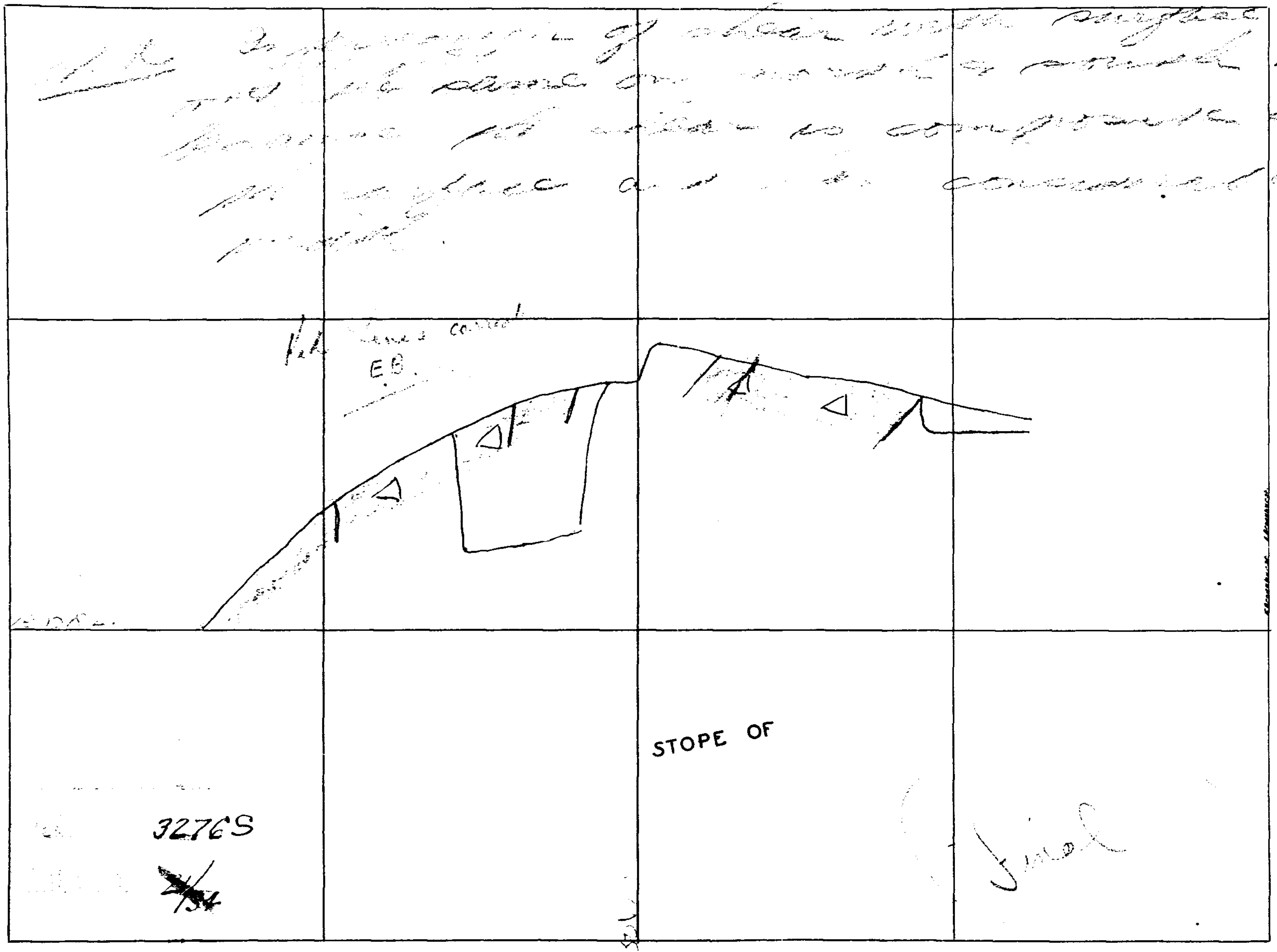
Dated 5/11/1945.



3275S
Cd
~~II~~
/~~50~~

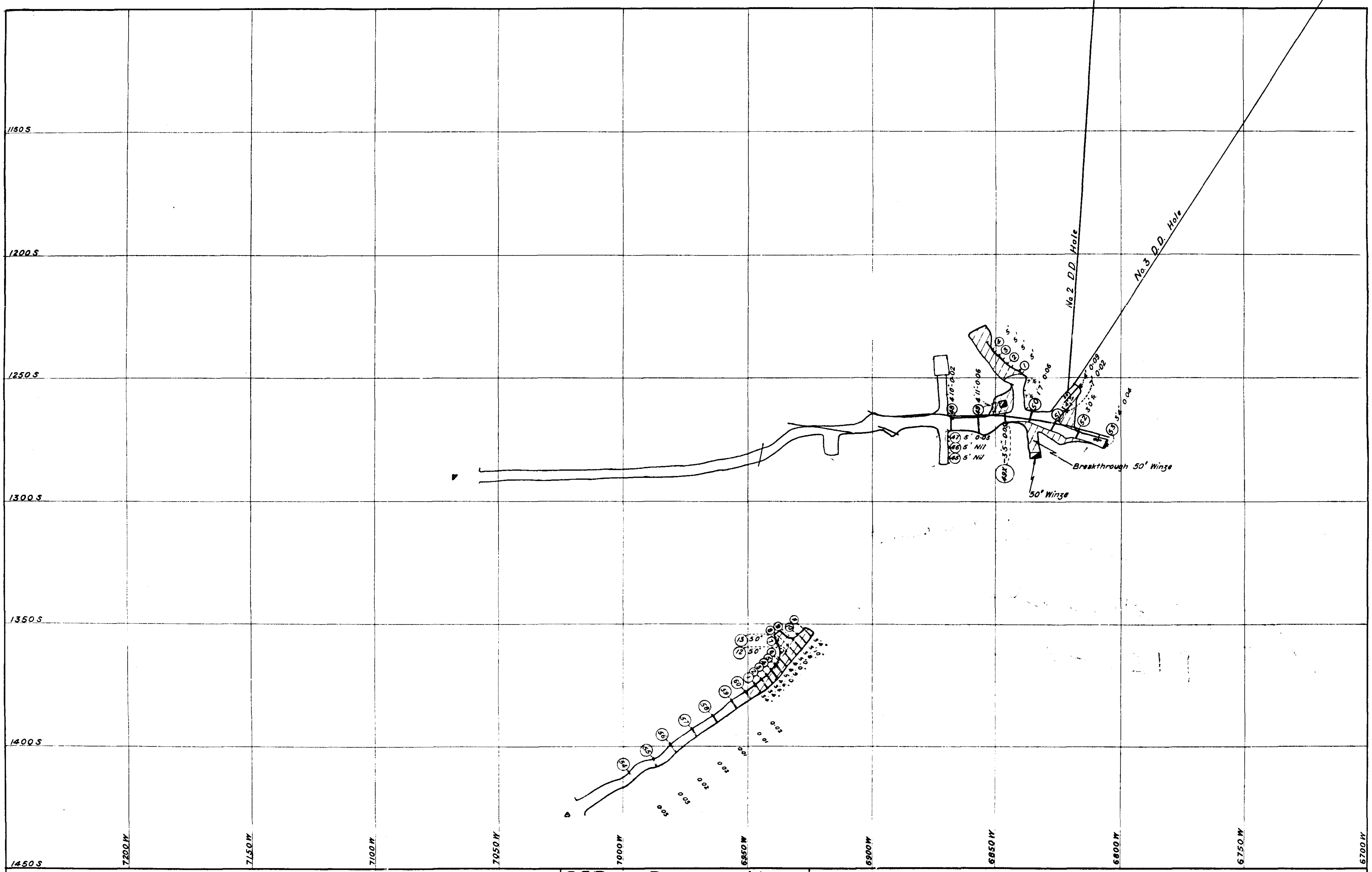
SUP 11
Cd

[illegible]



SUP 12
Cd.

W-E Projection on Main Lode
Showing gully on South side (Hanging wall)



S.A.G. DEPT. OF MINES

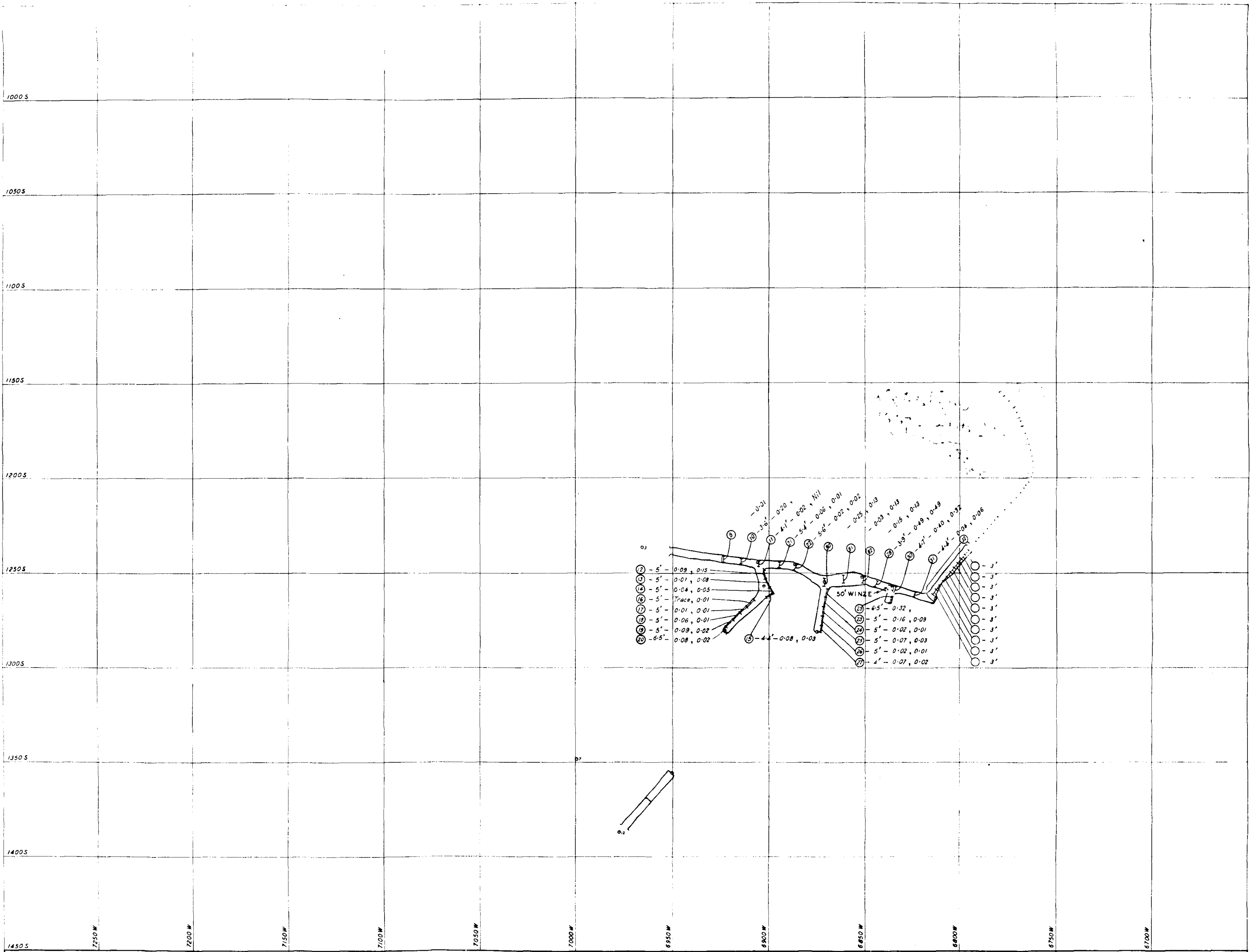
MT. PAINTER
No 6 WORKINGS
100' LEVEL
ASSAY PLAN

SCALE	1" = 20 Ft.	DATE
Drawn		~~RACK Nº R21/30~~
Traced		
Checked		~~PLAN No 2946~~
Approved		

2946
Cd

U.

UP 33
Cd



NOTE

SAMPLES SHOWN THUS:

Nº Width U_3O_8 %
⑫ - 5' - 0·09, 0·13

ASSISTANT GOVERNMENT GEOLOGIST

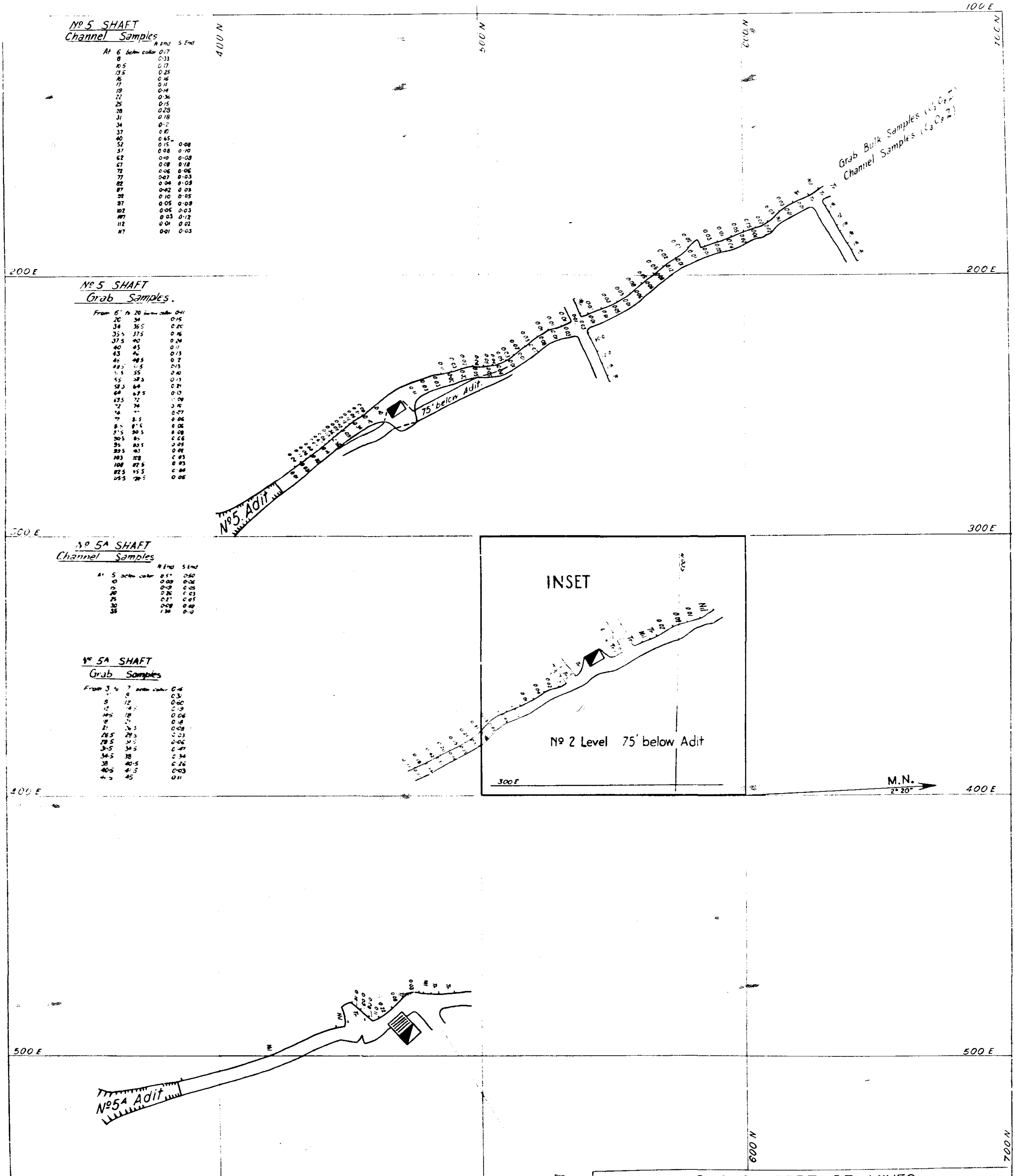
GOVERNMENT GEOLOGIST

S. A. G.—DEPT. OF MINES

MT. PAINTER

Nº 6 WORKINGS
50' LEVEL
ASSAY PLAN

Date	25 - 1 - 45	Scale 1" = 20'
Drawn	E.B.	PLAN Nº UP-34
Traced	J.G.P.	
Checked	J T MacD.	
Approved		



No 5 SHAFT
Channel Samples

	N End	S End
At 6' below collar	0·17	
8	0·33	
10·5	0·17	
13·5	0·25	
16	0·16	
17	0·11	
19	0·14	
22	0·36	
25	0·15	
28	0·29	
31	0·18	
34	0·12	
37	0·10	
40	0·65	
52	0·15	0·08
57	0·08	0·10
62	0·10	0·09
67	0·08	0·18
72	0·06	0·06
77	0·07	0·03
82	0·04	0·09
87	0·02	0·03
92	0·10	0·05
97	0·05	0·09
102	0·05	0·03
107	0·03	0·12
112	0·01	0·02
117	0·01	0·03

No 5 SHAFT
Grab Samples.

From 6' to	20 below collar	0·11
20	34	0·15
34	35·5	0·20
35·5	37·5	0·16
37·5	40	0·24
40	43	0·11
43	46	0·13
46	48·5	0·17
48·5	51·5	0·13
51·5	55	0·10
55	58·5	0·13
58·5	64	0·21
64	67·5	0·13
67·5	72	0·08
72	74	0·16
74	77	0·07
77	81·5	0·06
81·5	87·5	0·06
87·5	90·5	0·08
90·5	95	0·06
95	99·5	0·05
99·5	103	0·08
103	108	0·03
108	112·5	0·03
112·5	115·5	0·04
115·5	120·5	0·06

No 5A SHAFT
Channel Samples

	N End	S End
At 5' below collar	0·57	0·50
10	0·09	0·06
15	0·19	0·05
20	0·26	0·03
25	0·27	0·05
30	0·09	0·02
35	1·30	0·10

No 5A SHAFT
Grab Samples

From 3 to	7 below collar	0·46
7	9	0·31
9	12	0·60
12	14·5	0·19
14·5	18	0·06
18	21	0·18
21	26·5	0·08
26·5	28·5	0·03
28·5	31·5	0·06
31·5	34·5	0·47
34·5	38	0·34
38	40·5	0·26
40·5	41·5	0·03
41·5	45	0·11

Reference Drawings	Reg. No.
E.P. No 4 Adit. Assay Plan	48-20
E.P. No 1 Adit Assay Plan	S 110

No.	Authority	Date	Amendment	Req. No. 39	D.M.

S. A. G. DEPT OF MINES.

EAST PAINTER
No 5. AND No 5A. ADITS
ASSAY PLAN

Approved

Director of Mines

Drawn B.T.
Traced G.P.K.
Checked J.B.A

Passed

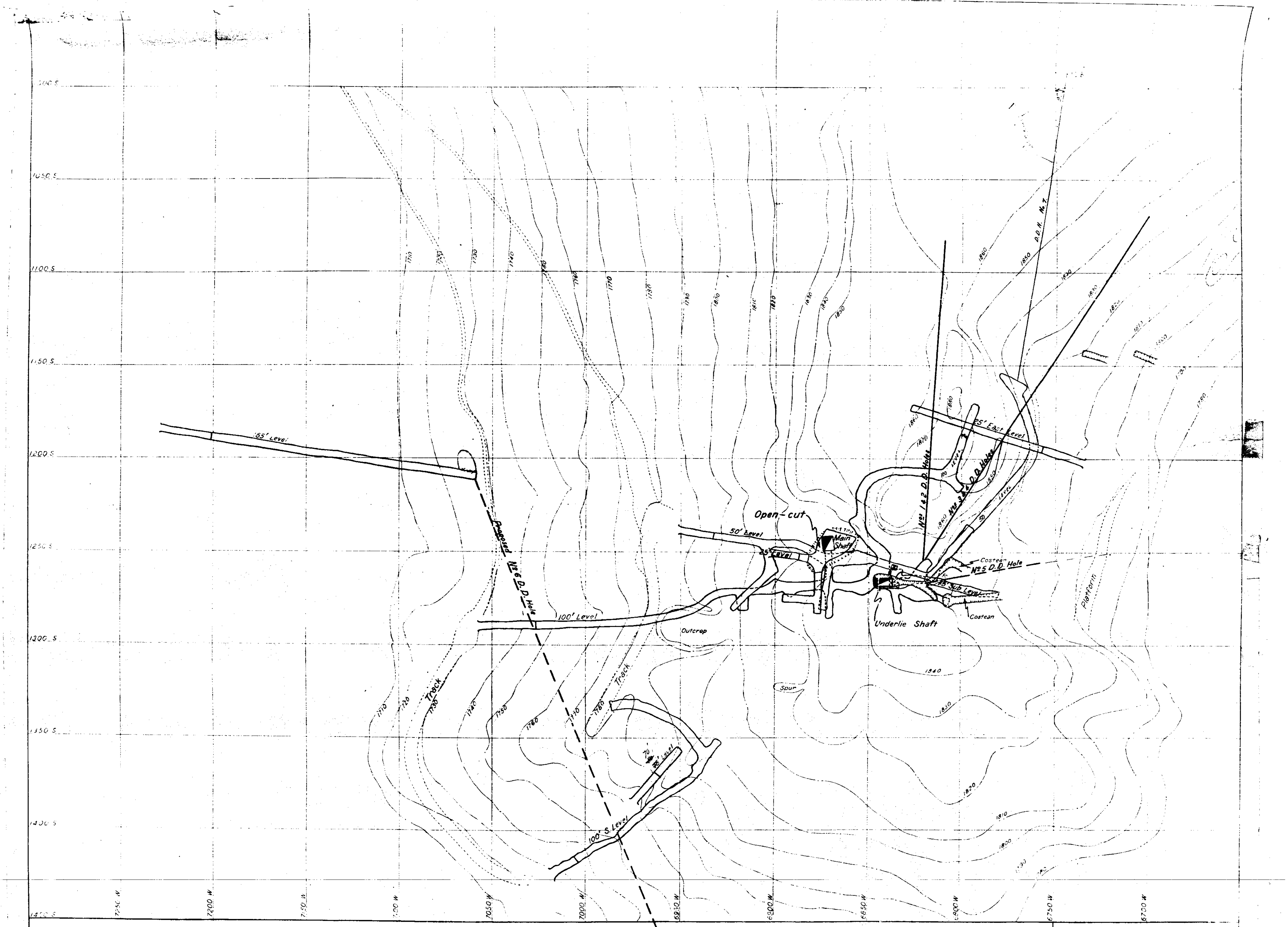
Chief Draughtsman

SCALE 20 feet to 1 inch.

~~48-21~~ Cd

DATE 10-2-48.

U.E.P.-36 cd



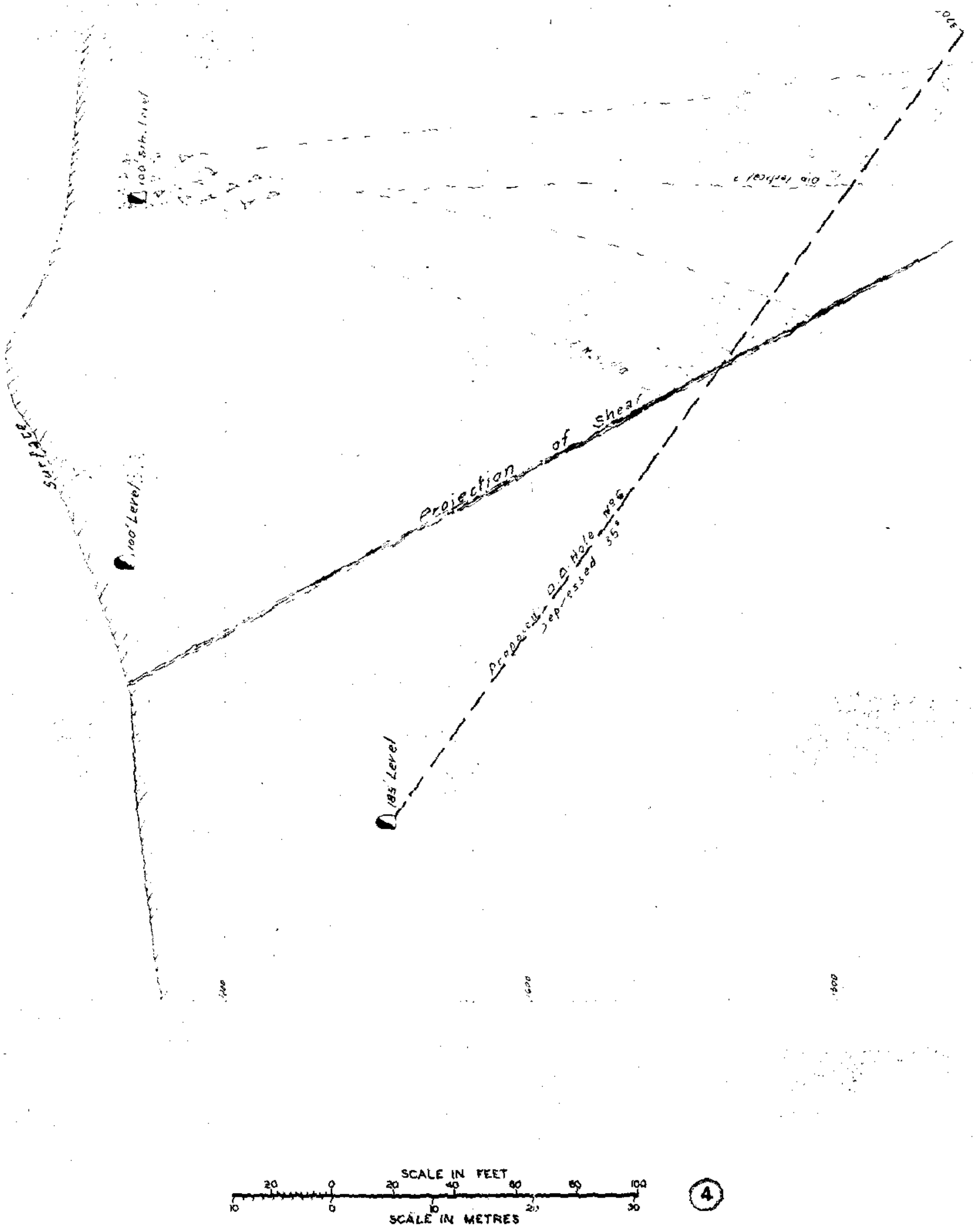
S. A. G. - DEPT. OF MINES

MT. PAINTER
Nº 6 WORKINGS
COMPOSITE PLAN
U.P. 39

C. Broadhurst
ASSISTANT GOVERNMENT GEOLOGIST

Up to date 24/8/45

Approved	Date	28-3-45	Scale	1" = 20'
	Drawn	E. B.	PLAN Nº 200	
Passed	Traced	J. G. P.		
	Checked	J.T.MacD.		



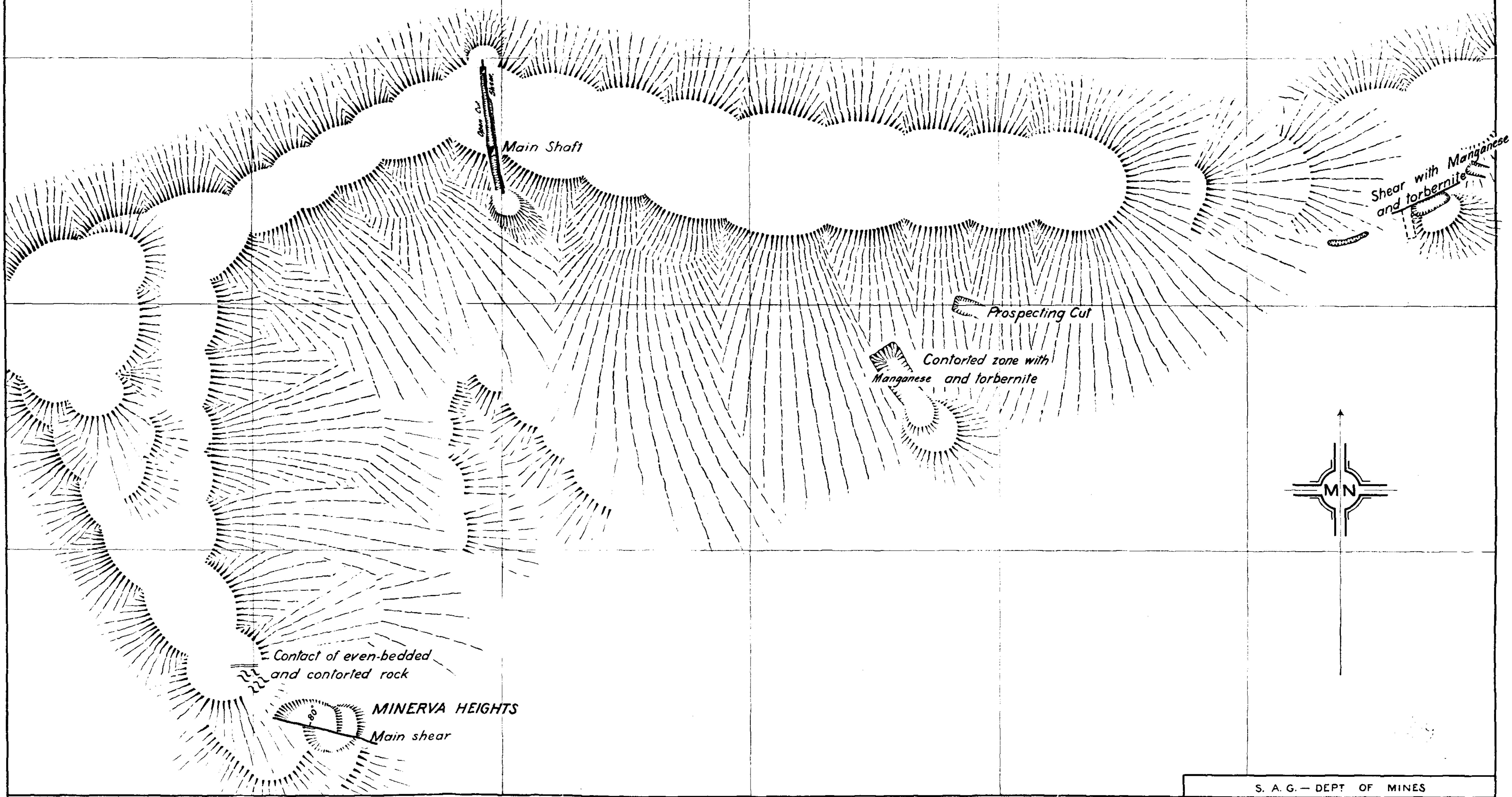
④

MT PAINTER
Section thro' proposed No. 6 D.D. Hole
Scale 1" = 20'

Plan No 2994

~~2994~~
~~Cd~~

UP-40
Cd



S. Broadhurst
ASSIST GOVERNMENT GEOLOGIST.

S.B. Dickinson.
GOVERNMENT GEOLOGIST.

S. A. G. — DEPT OF MINES

MT PAINTER

Nº 7 WORKINGS

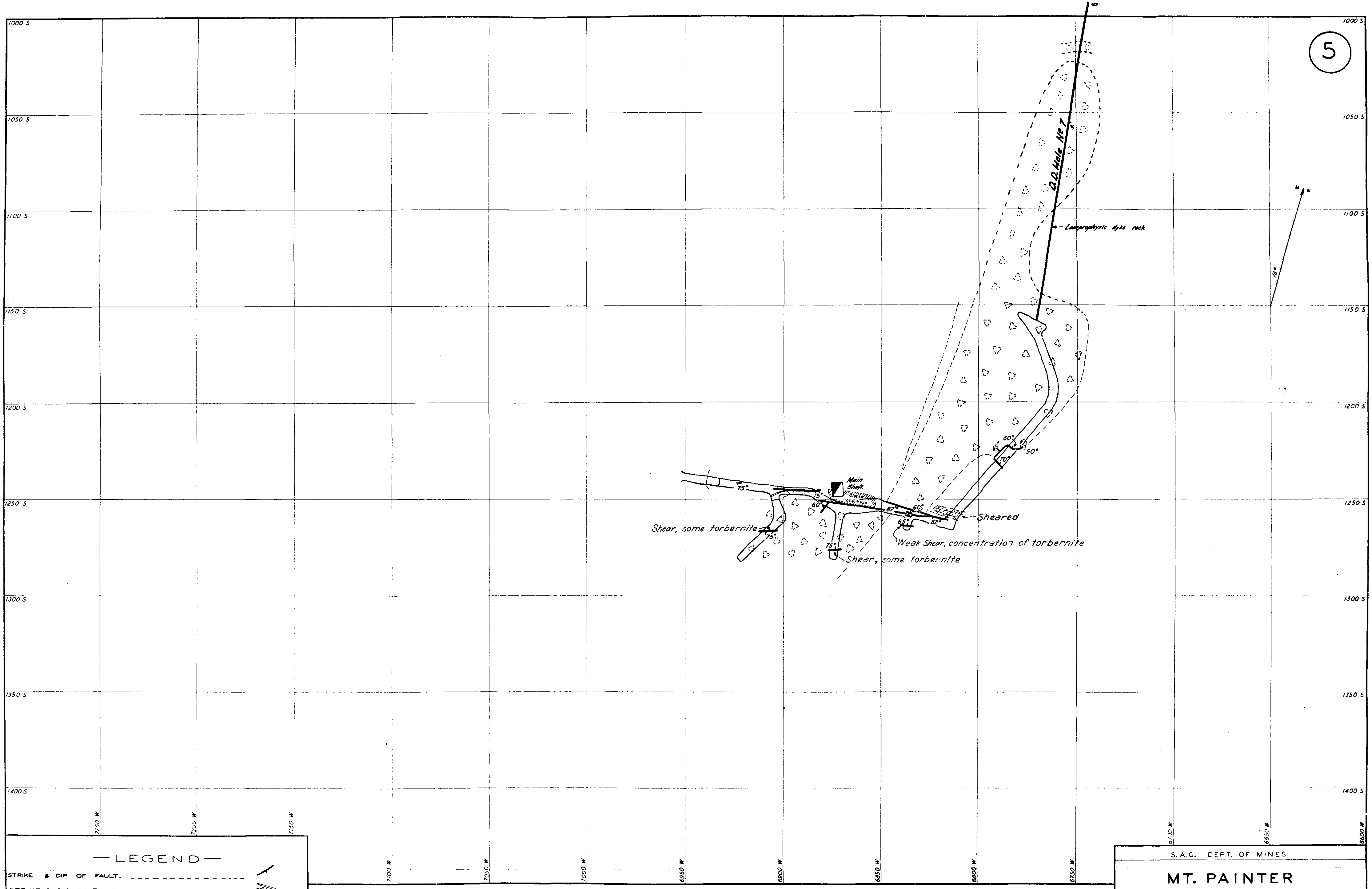
SURFACE PLAN

Approved
S.B.
Passed

Date	16·3·45	Scale 1" = 40'
Drawn	E.B.	PLAN Nº UP-41
Traced	J.T.MacD.	
Checked	J.G.P.	

Cd.

⑤



—LEGEND—

STRIKE & DIP OF FAULT	
STRIKE & DIP OF BANDING	70°
BANDED GNEISS	☐
PEGMATITIC GRANITE	☐
SILICIFIED GNEISS	☐
HAEMATITE BRECCIA	△ ▽
MANGANIFEROUS IRONSTONE	☐

C. Broadhurst
ASSISTANT GOVERNMENT GEOLOGIST

S.B. Dickinson
GOVERNMENT GEOLOGIST

Drawing References
Composite Plan No. 3185
50' Level Assay Plan No. 3183.

Approved
C. Broadhurst.

S.A.G. DEPT. OF MINES

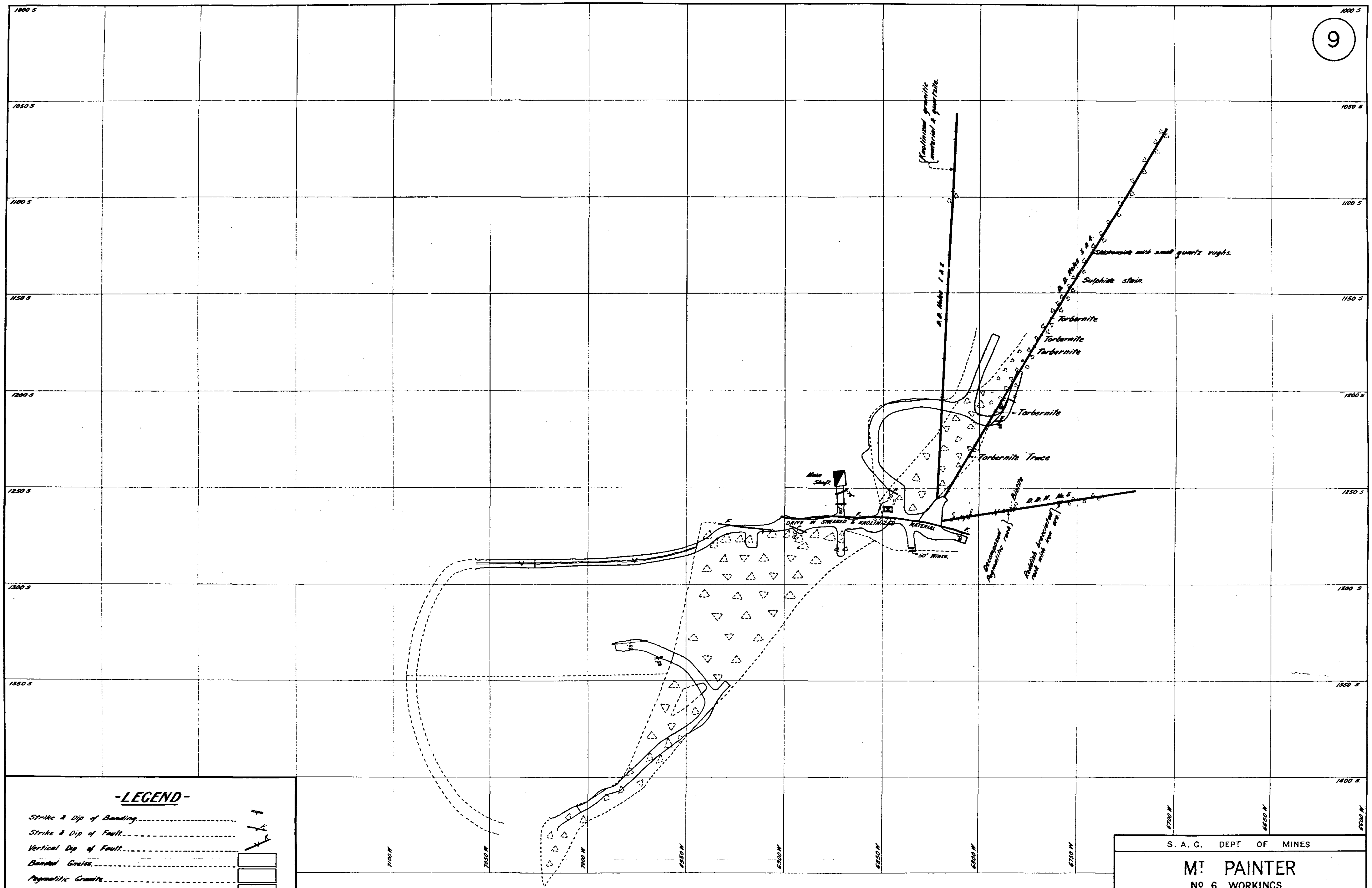
MT. PAINTER
Nº 6 WORKINGS
50 FEET LEVEL
GEOLOGICAL PLAN

SCALE	1 INCH = 20 FEET		
Passed		Drawn	E. B.
N.E. Lock		Traced	J.G.P. & B.S.G.
Date	22-8-45.	Checked	B S G

PLAN Nº UP-49

Cd

⑨



C. Broadhurst
Assistant Government Geologist.

S. B. Dickinson
Government Geologist.

Drawing Reference.
Composite Plan (No. 3185)
100' Assay Plan (No. 3182)

Approved.
C. Broadhurst

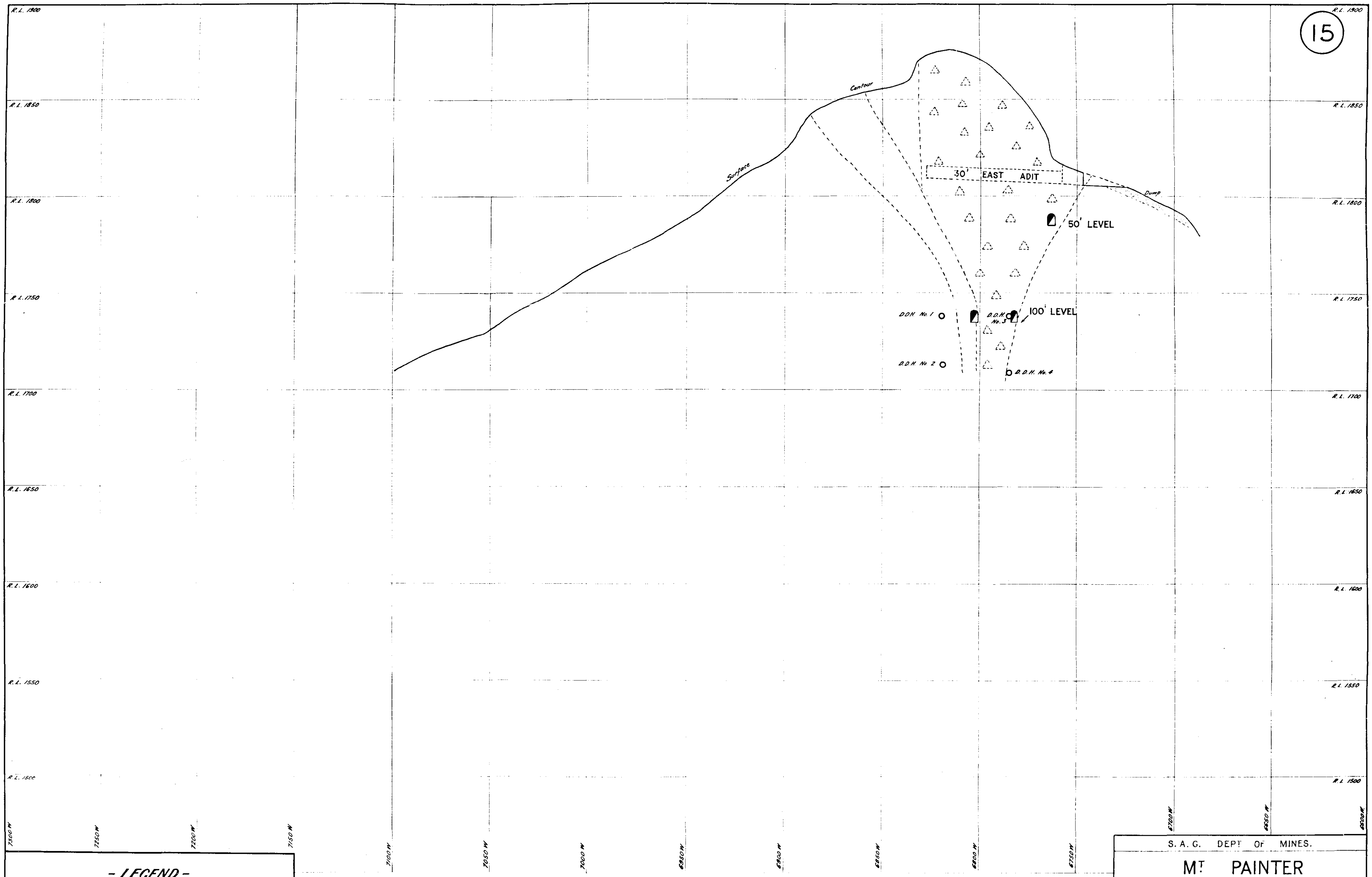
S. A. G. DEPT OF MINES

MT PAINTER
Nº 6 WORKINGS
100′ LEVEL
GEOLOGICAL PLAN

SCALE	ONE INCH = 20 FEET		
Passed		Drawn	E.B.
		Traced	B.S.G.
Date.	7. 9. 45	Checked	J.G.P. & B.C.B.

PLAN Nº UP-52

⑮



- LEGEND -

Banded Gneiss	☐
Silicified Gneiss	☐
Haematite Rock	☐
Haematite Breccia	△ △ △

C. Broadhurst
Assistant Government Geologist.

S.B. Dickinson.
Government Geologist.

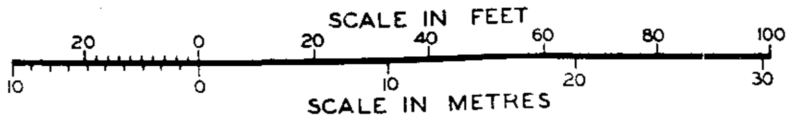
Approved
C. Broadhurst

S.A.G. DEPT. OF MINES.

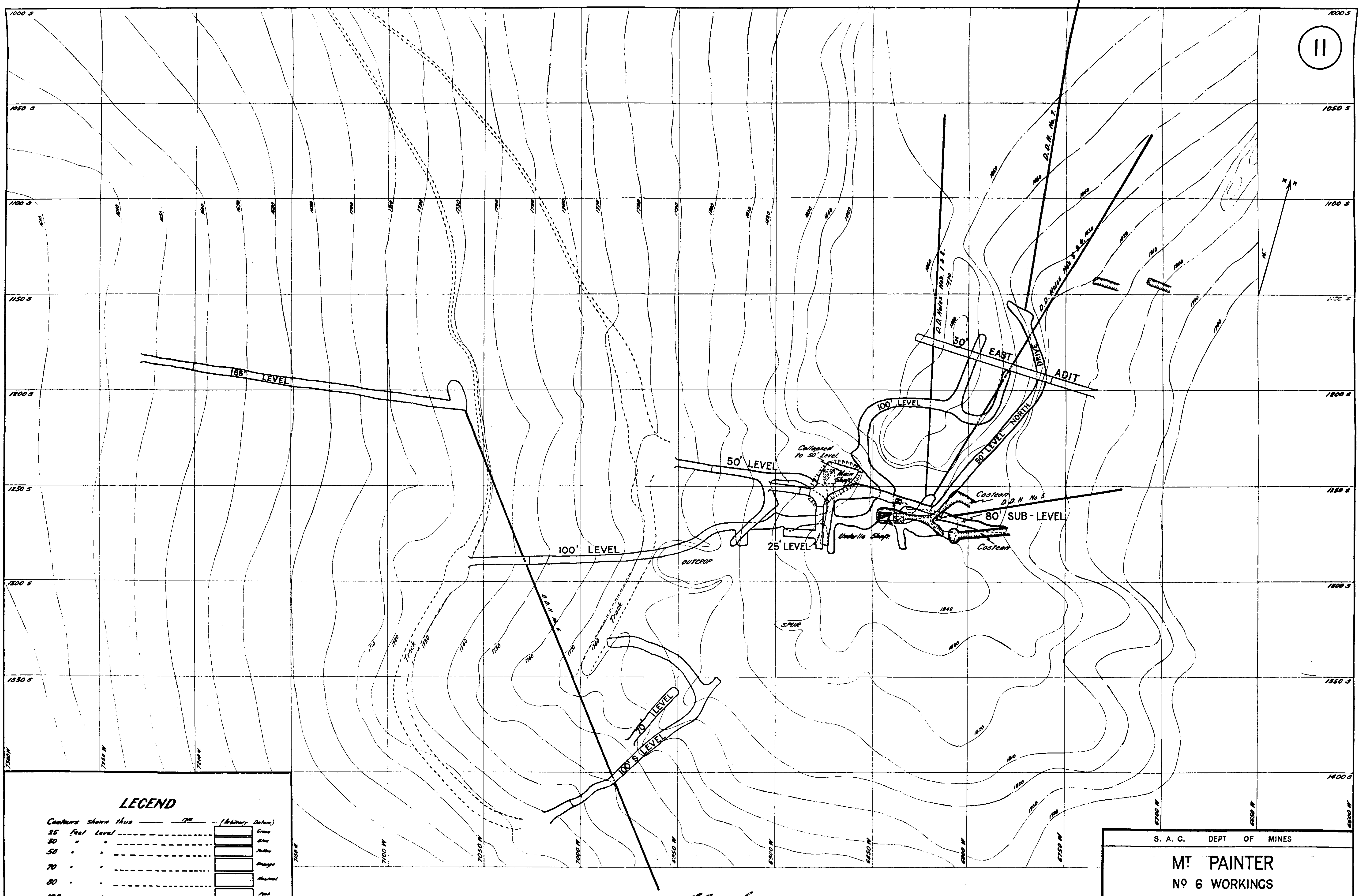
M.T PAINTER
Nº 6 WORKINGS
W-E SECTION AT 1200 S

SCALE	ONE INCH = 20 FEET		
Passed.		Drawn	E.B.
W.E. [illegible]		Traced	B.S.C
Date	22-8-45	Checked	W.E.L.

PLAN Nº UP-55



④



LEGEND

Contours shown thus ——— 1710 ——— (Arbitrary Datum)

25 Feet Level	☐	Green
30 " "	☐	Blue
50 " "	☐	Yellow
70 " "	☐	Orange
80 " "	☐	Neutral
100 " "	☐	Pink
100 S " "	☐	Pink
185 " "	☐	Brown

L. Broadhurst
Assistant Government Geologist.

S. B. Dickinson
Government Geologist.

Drawing Reference	Approved.
Composite (Old) Nº 2855 Surface Geological Nº 3184 Dumps & Surface Assay Plan Nº 2851	L. Broadhurst

S. A. G. DEPT OF MINES

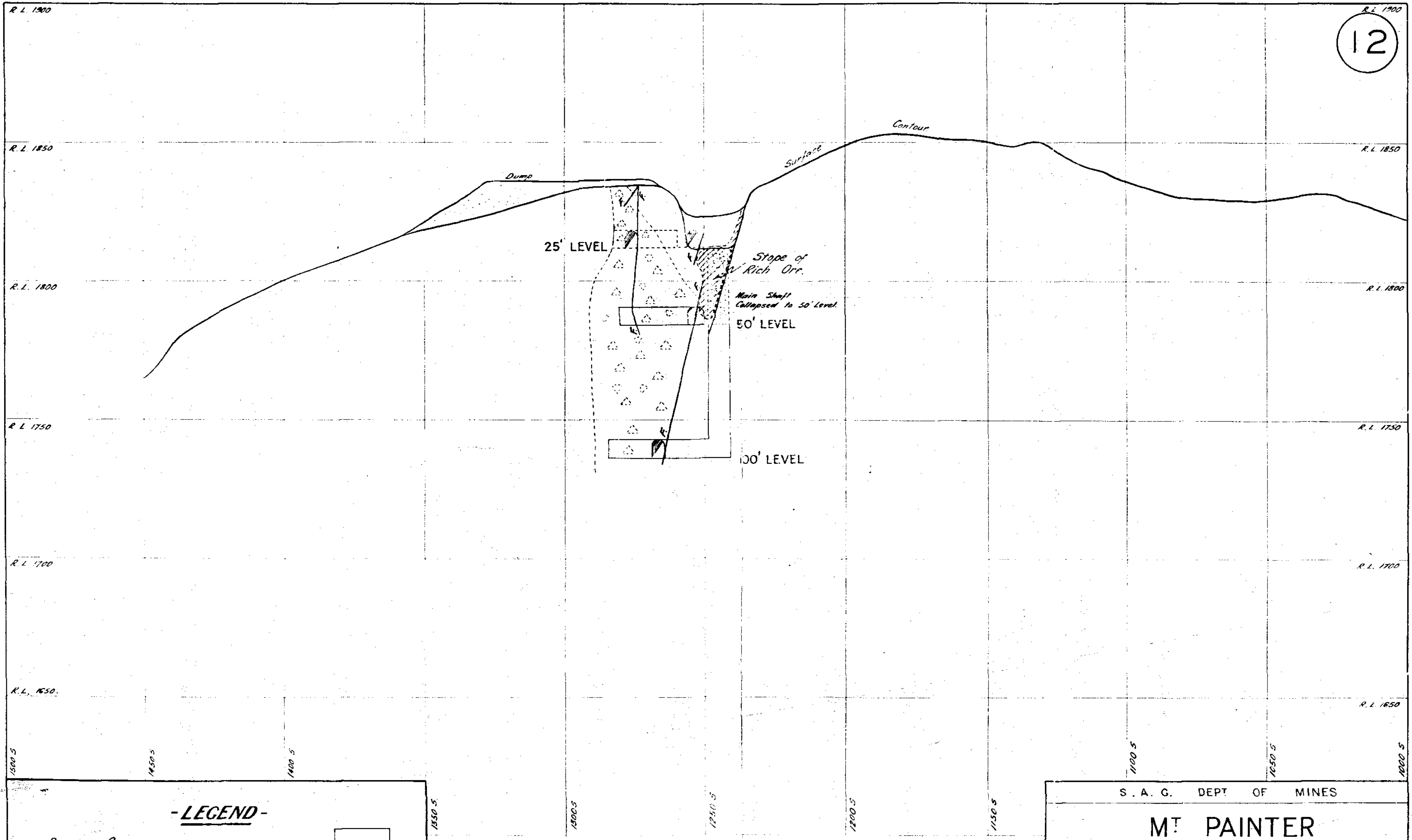
Mᵀ PAINTER

Nº 6 WORKINGS

COMPOSITE PLAN
(SHOWING SURFACE CONTOURS)

SCALE	ONE INCH – 20 FEET			PLAN Nº
Passed		Drawn	E. B.	
		Traced	B. S. G.	
Date	18-10-45	Checked	J. G. P.	ed. UP-58

⑫



-LEGEND-

Banded Gneiss ☐
Silicified Gneiss ☐
Haematite Breccia ▨
Manganiferous Ironstone ☐
Positions of Faults shown thus F/

C. Broadhurst
Assistant Government Geologist

S.B. Dickinson
Government Geologist

Drawing Reference
Composite No. 3185
Levels 25' No. 3176
50' No. 3175
100' No. 3173

Approved.
C. Broadhurst

S.A.G. DEPT OF MINES

MT PAINTER
Nº 6 WORKINGS
N-S SECTION THRO' MAIN SHAFT

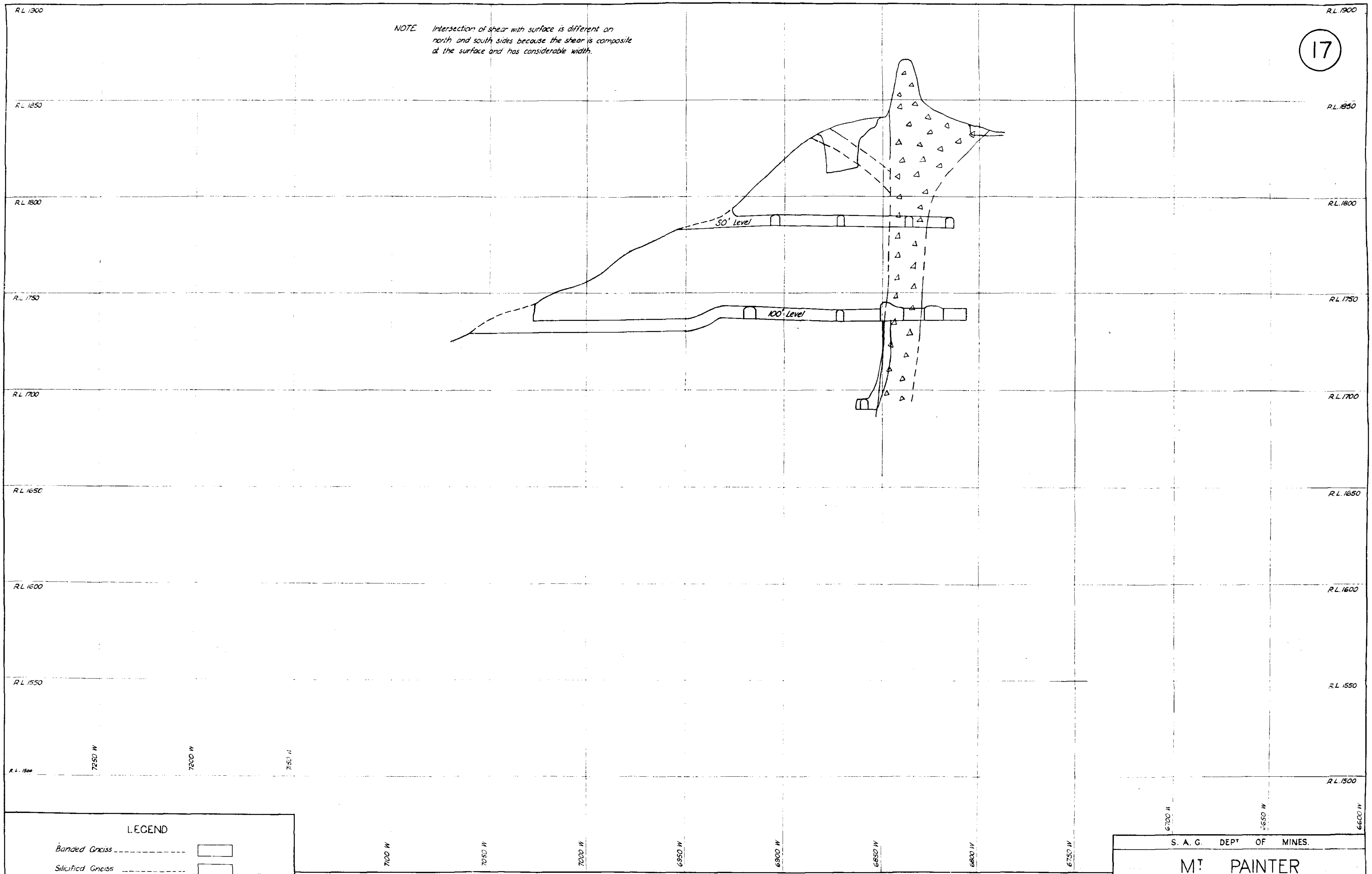
SCALE ONE INCH = 20 FEET

Passed.	Drawn.	E. B
	Traced.	B. S. G
Date. 7. 9. 45	Checked.	E. B.

PLAN Nº
UP-60
Cd.

⑰

NOTE Intersection of shear with surface is different on north and south sides because the shear is composite at the surface and has considerable width.



LEGEND

Banded Gneiss	☐
Silicified Gneiss	☐
Haematite Rock	☐
Haematite Breccia	△ △ △

S. Broadhurst
Assistant Government Geologist

S.B. Dickinson
Government Geologist

Drawing Reference	Approved
Composite No 3185 Levels 50' (No 3175) 80' (No 3177) 100' (No 3178)	S. Broadhurst

S. A. G. DEPT OF MINES.

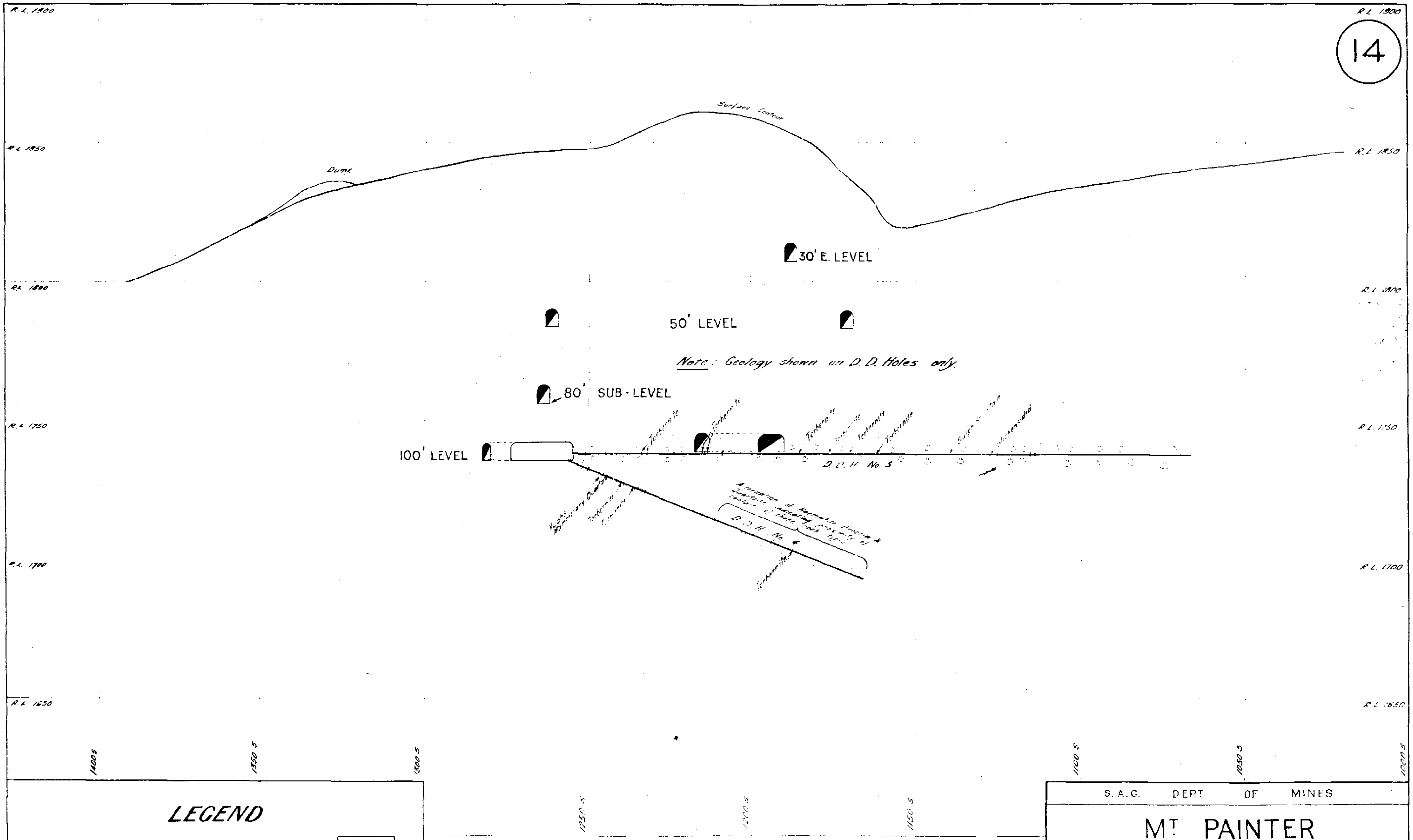
MT PAINTER
No 6 WORKINGS
W-E PROJECTION ON MAIN LODE
GEOLOGY NORTH OF SHEAR

SCALE	ONE INCH = 20 FEET		
Passed		Drawn	E B
		Traced	G.T.J.
Date	4-9-45	Checked	E. B

PLAN No. UP-62
Cd.

SCALE IN FEET
20 0 20 40 60 80 100
10 0 10 20 30

④



LEGEND

Silicified Gneiss	☐
Siliceous Breccia	▨
Haematite Rock	☐
Haematite Breccia	▨

C. Broadhurst
Assistant Government Geologist.

S. B. Dickinson
Government Geologist.

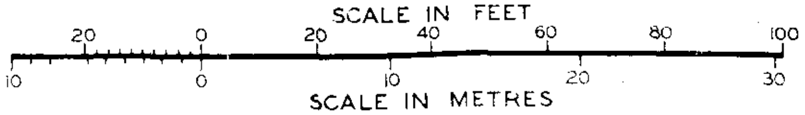
Drawing Reference
Composite No 3185
Levels 25 No 3176
50 No 3175
80 No 3177
100 No 3178

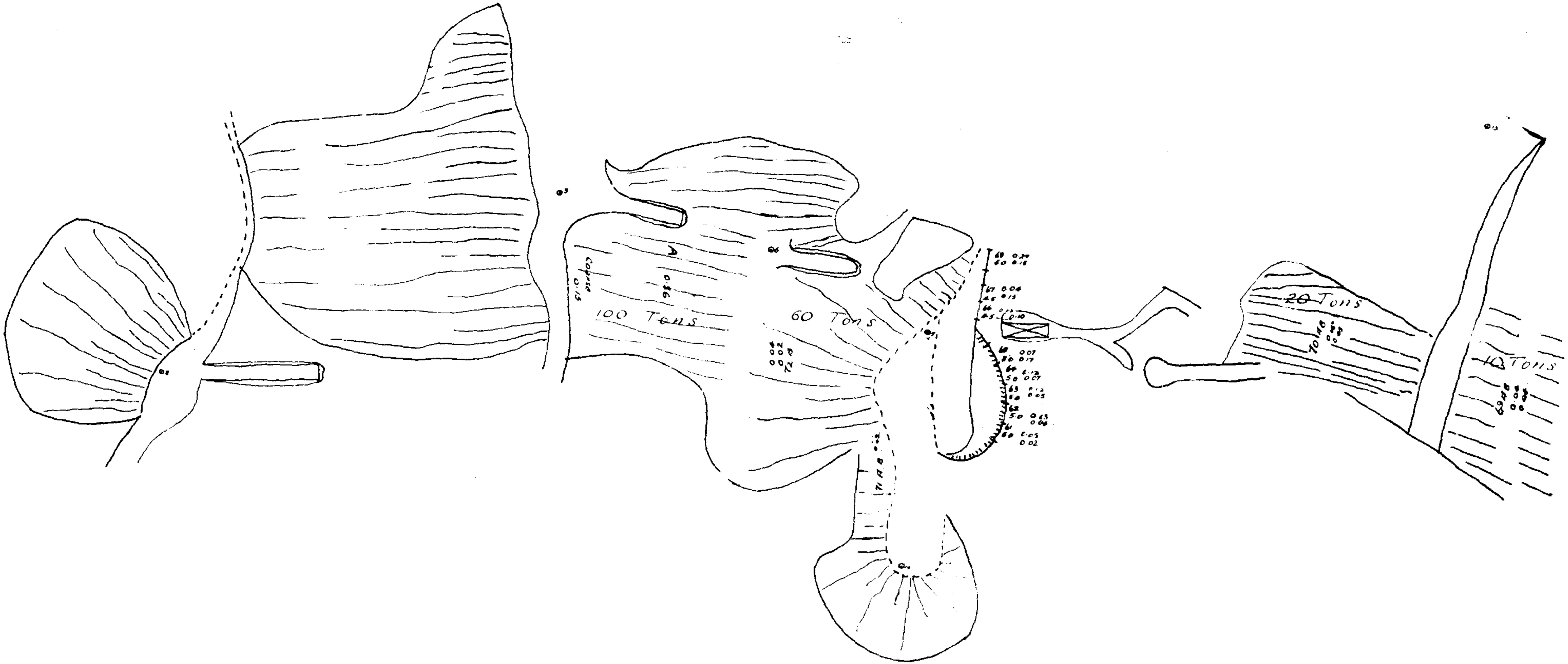
Approved
C Broadhurst

S.A.G. DEPT OF MINES

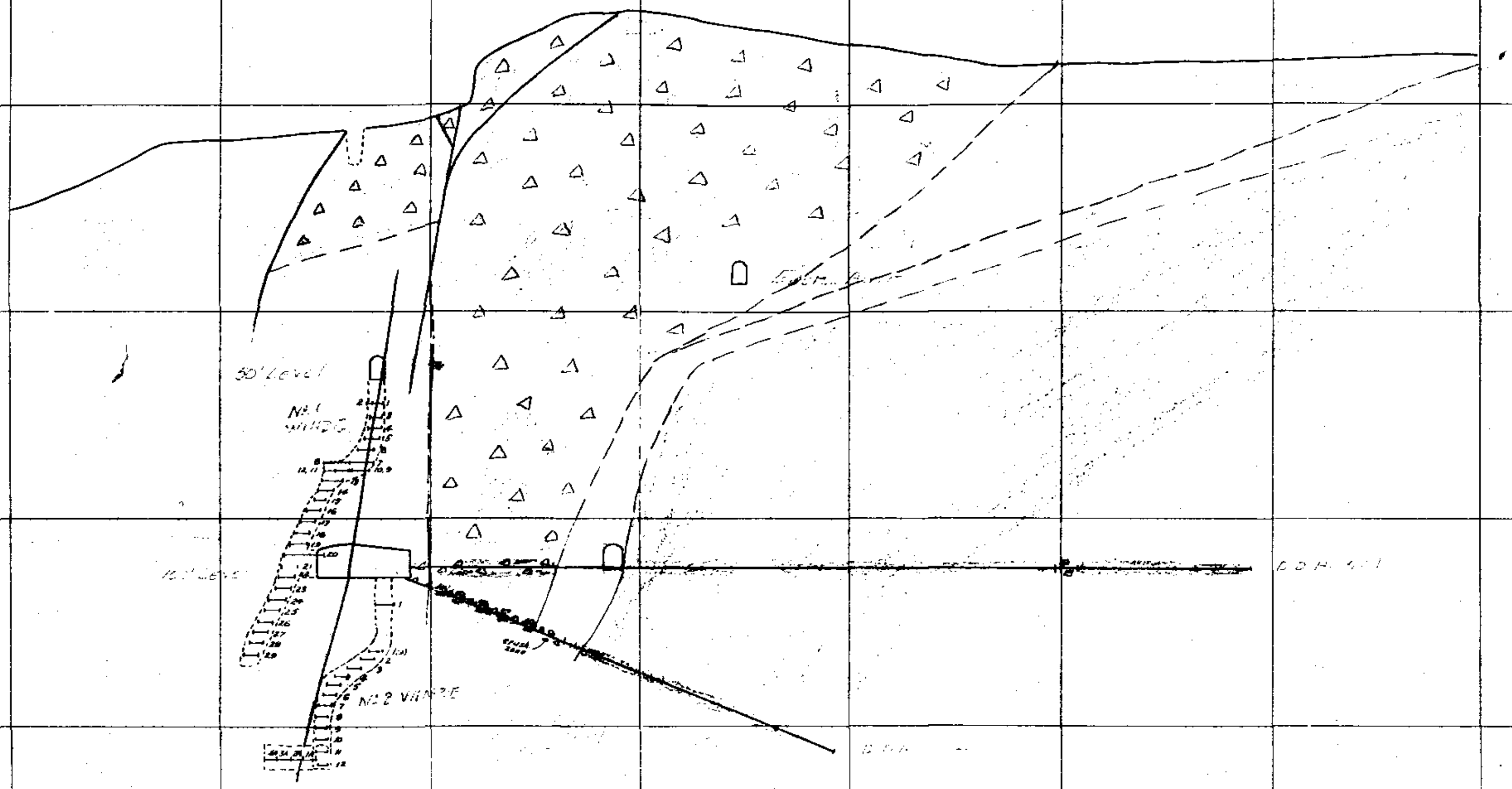
Mt PAINTER
No 6 WORKINGS
SECTION THRO' Nos 3 & 4 D.D. HOLES

SCALE	ONE INCH = 20 FEET		PLAN No
Passed	Drawn	F.B.	UP-63
	Traced	B.S.G.	
Date 7-9-45	Checked	J.G.P.	



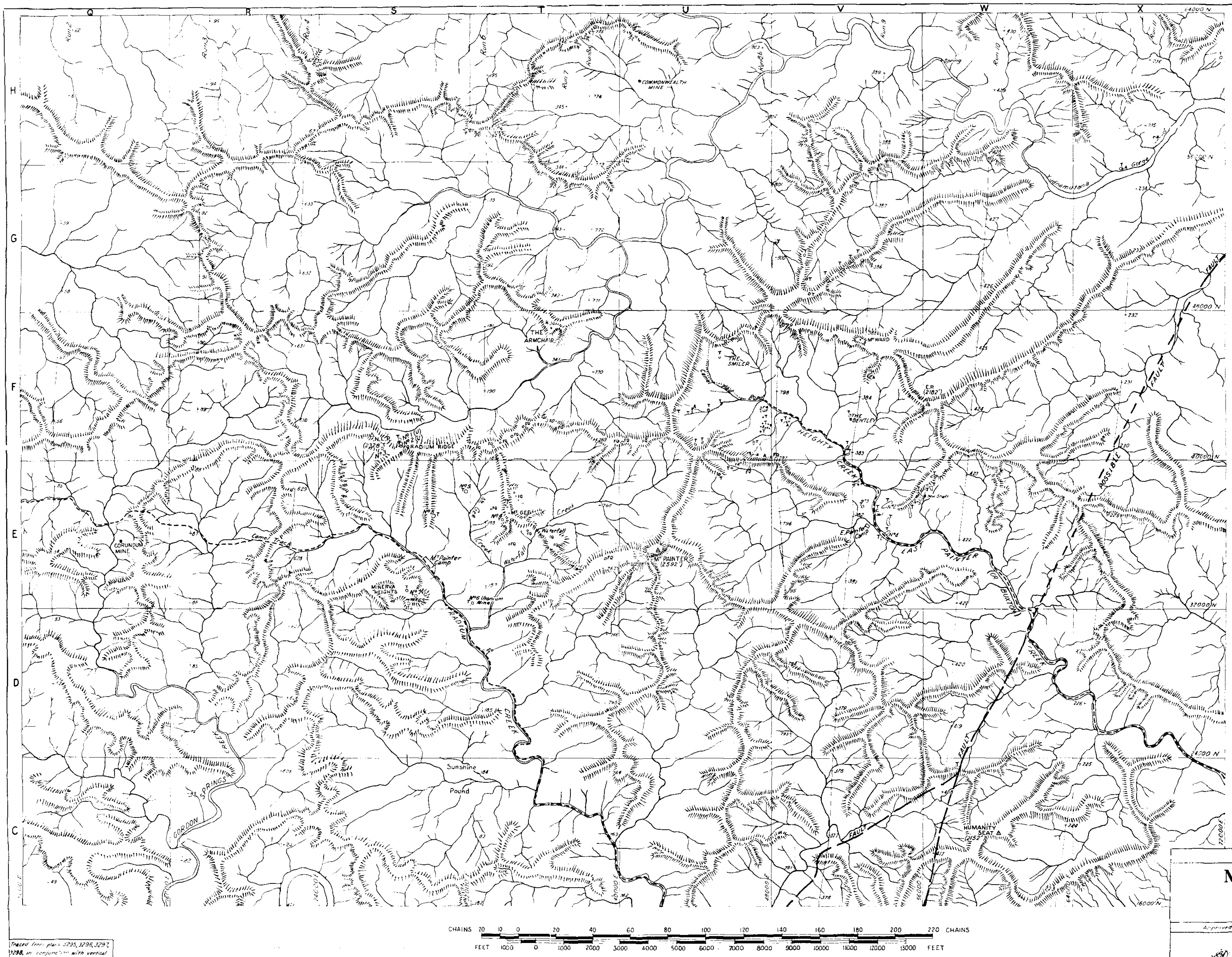

Coarse 0.15
A
0.36
100 Tons
60 Tons
0.04
0.02
72.0
20 Tons
70 A.B
0.08
0.08
10 Tons
69 A.B
0.04
0.08
71 A.B 0.12
68 0.24
6.0 0.18
67 0.04
4.5 0.15
66 0.13
4.5 0.10
65 0.07
8.0 0.17
64 0.12
5.0 0.07
63 0.12
5.0 0.05
62
5.0 0.13
0.06
61
6.0 0.05
0.02

U.P.-76



HAEMATITE BRECCIA —
SPECULAR HAEMATITE ROCK

UP-81



T.N.

LEGEND

CREEKS
MAIN DIVIDES
TRIG. STATIONS △
TRACKS
SHAFTS ■

<u>URANIUM OCCURRENCES</u>: (as at Sept. 1948)

Autunite	(orange)	A○
Torbernite	(green)	T○
Uranophane	(yellow)	UR○
Fluorescent Quartz	(purple)	FQ○
Radio-active ?	(red)	RA○

Note: *The grid shown is based on an arbitrary datum for Humanity Seat △ of 60,000E, 20,000N, and is in squares of 1 mile sides. Co-ordinate distances are in links.*

S. A. G. DEPT OF MINES.

Mt. PAINTER AREA

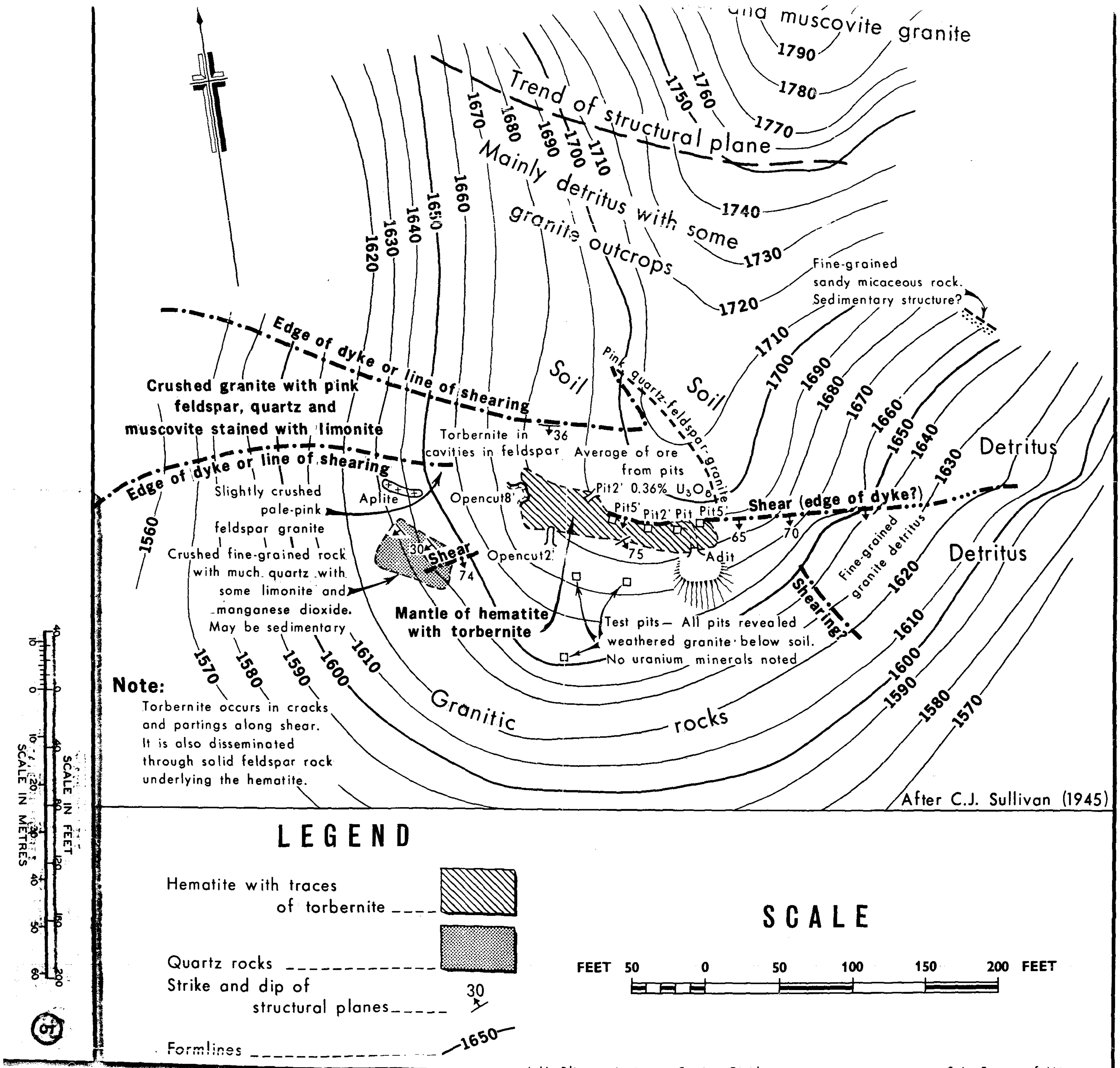
SHOWING ALL KNOWN
URANIUM OCCURRENCES
(INCLUDING FLUORESCENT QUARTZ)

Approved	Uranium Occurrences determined from aerial photos and field observations by D. J. Simes.	SCALE 20 Chains to 1 inch.
Director of Mines	Drawn E.F.R. Passed J.T.MacD	UP-93 Cd
		DATE December 1947

Traced from plans 3295, 3296, 3297, 3298, in conjunction with vertical aerial photographs.

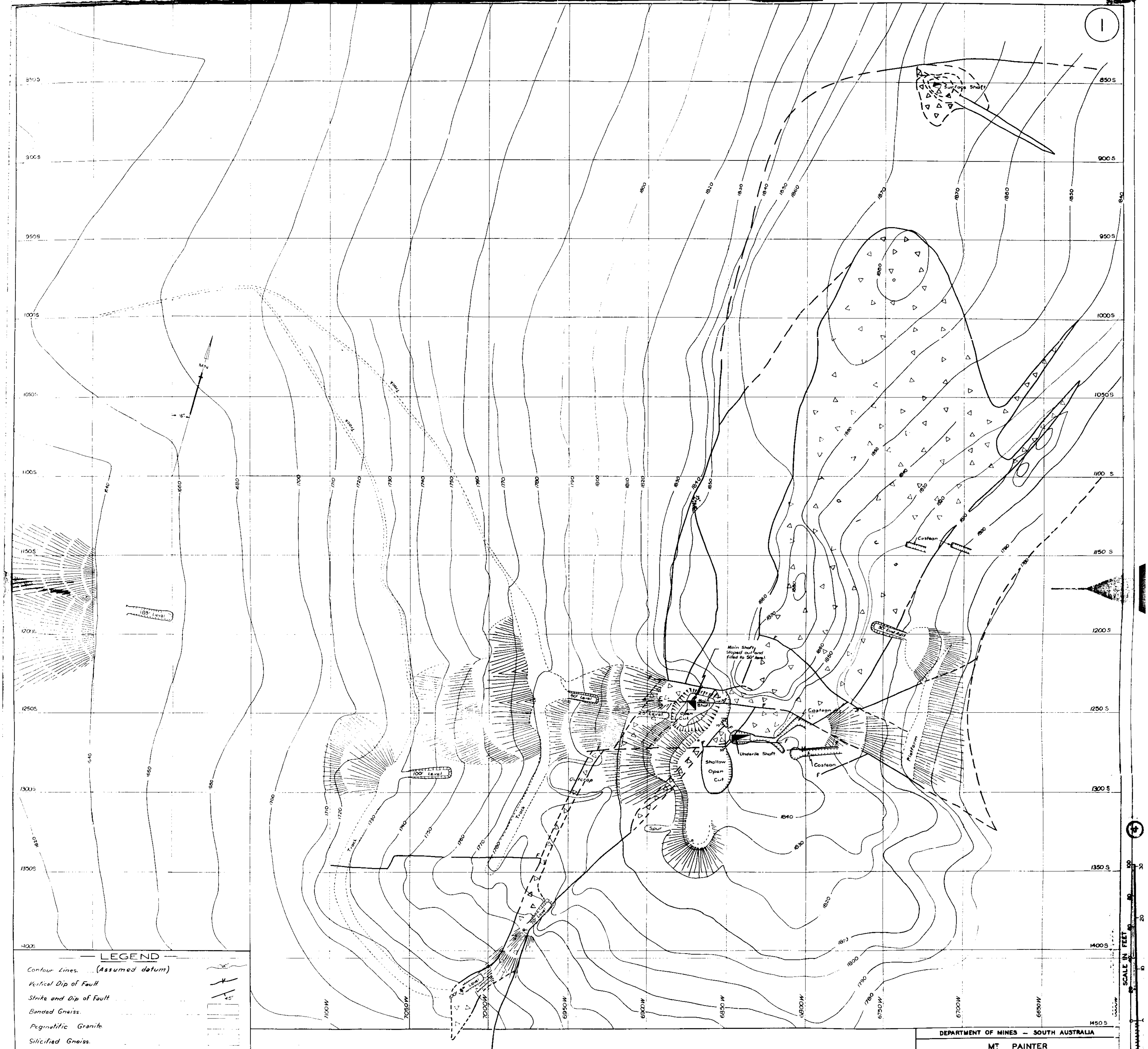
A	R-g 1020	17-9-48	Additional Occurrences.

Fluorescent Quartz localities determined by R. G. Thomas, S. C. Dickinson, Dr. A. B. Edwards, and D. R. Bowes.



A.H. Blissett Assistant Senior Geologist

S.A. Dept. of Mines



Signed E. Broadhurst.
ASSIS.T GOVERNMENT GEOLOGIST.

Signed S.B Dickinson.
GOVERNMENT GEOLOGIST.

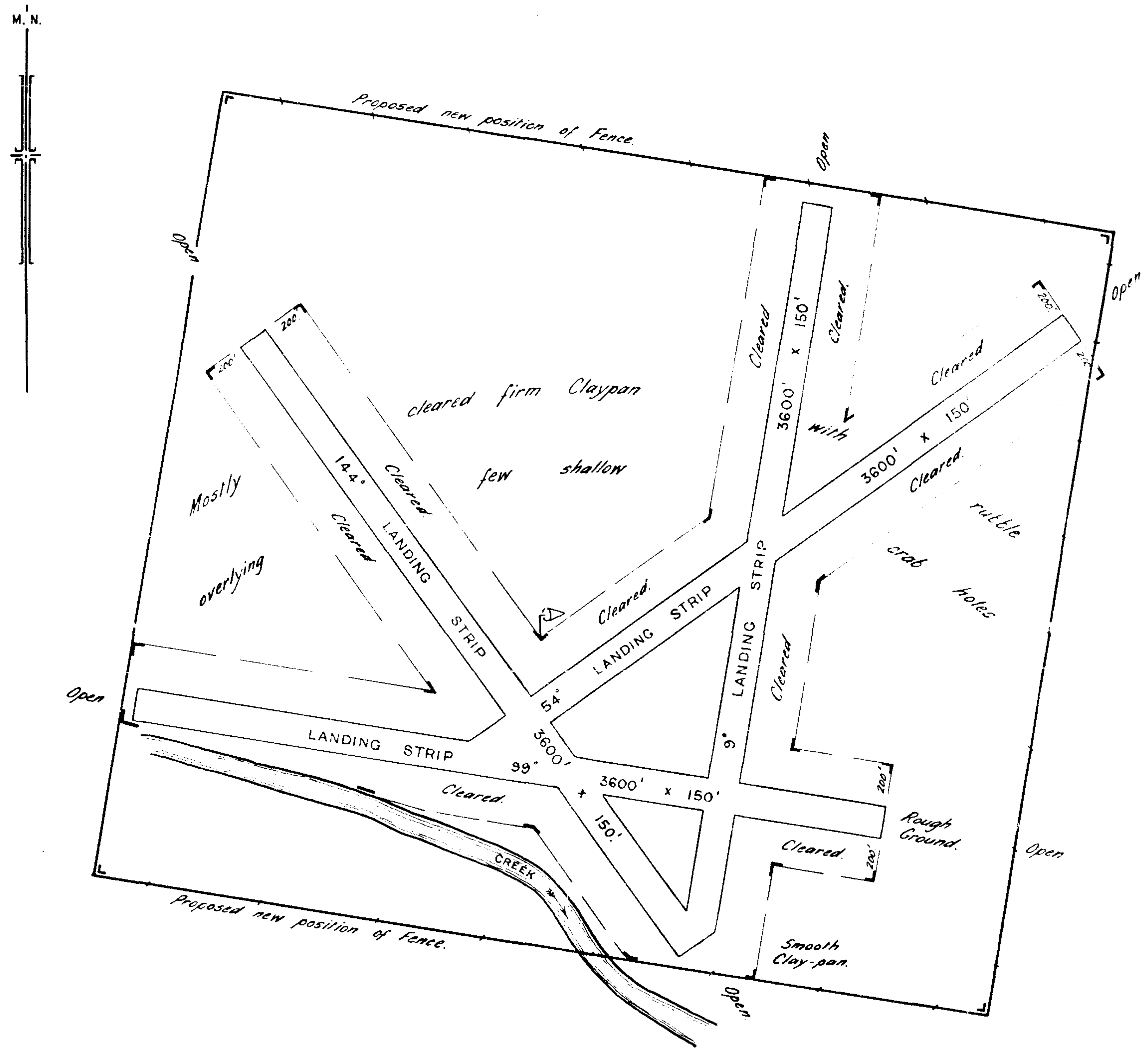
Drawing References	Approved
Composite level plan Nº 3185.	Signed E. Broadhurst.

Director of Mines

DEPARTMENT OF MINES — SOUTH AUSTRALIA

MT PAINTER
Nº 6 WORKINGS
SURFACE GEOLOGICAL PLAN

Drn. EB	SCALE: 1" = 20'
Tcd. NHF	Plan Nº LUP 104. Old Plan Nº 3184.
Chd.	
Exd.	DATE: 17·10·45.



Homestead 2½ miles West.
Magnetic Declination 6°
Altitude 500'±
Prevailing Winds:- Variable.
Latitude 30°31'30" S.
Longitude 139° 21' 30" E.
Galvanized Iron Markers at
Intersections & corners shown ┌

Drawing Reference.
Locality Plan. No. 3170.

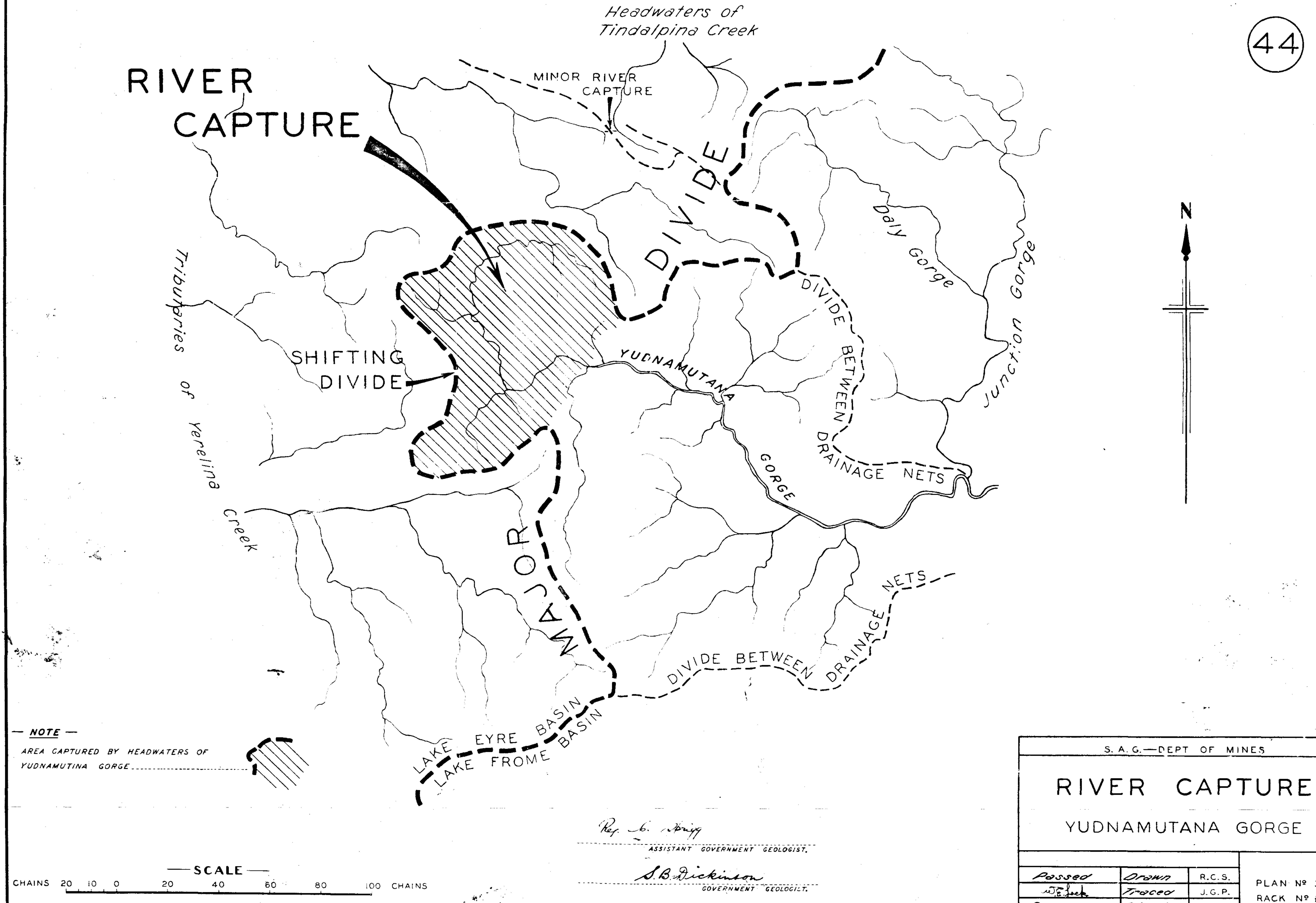
S. A. G. DEPT OF MINES

LANDING GROUND
BALCANOONA STATION S.A.

SCALE	ONE INCH = 500 FEET		
Passed		Reproduced from Civil Aviation Plan No Y-348	
		Traced	B.S.G.
Date	19-9-45	Checked	W.E.L.

PLAN Nº 3208
Cd
~~R. 21/15~~

(44)



Reg. C. Sprigg
ASSISTANT GOVERNMENT GEOLOGIST.

S.B. Dickinson
GOVERNMENT GEOLOGIST.

S.A.G.—DEPT OF MINES

RIVER CAPTURE

YUDNAMUTANA GORGE

Passed	Drawn	R.C.S.
W.E. Lock	Traced	J.G.P.
Date 3-12-45	Checked	W.E.L.

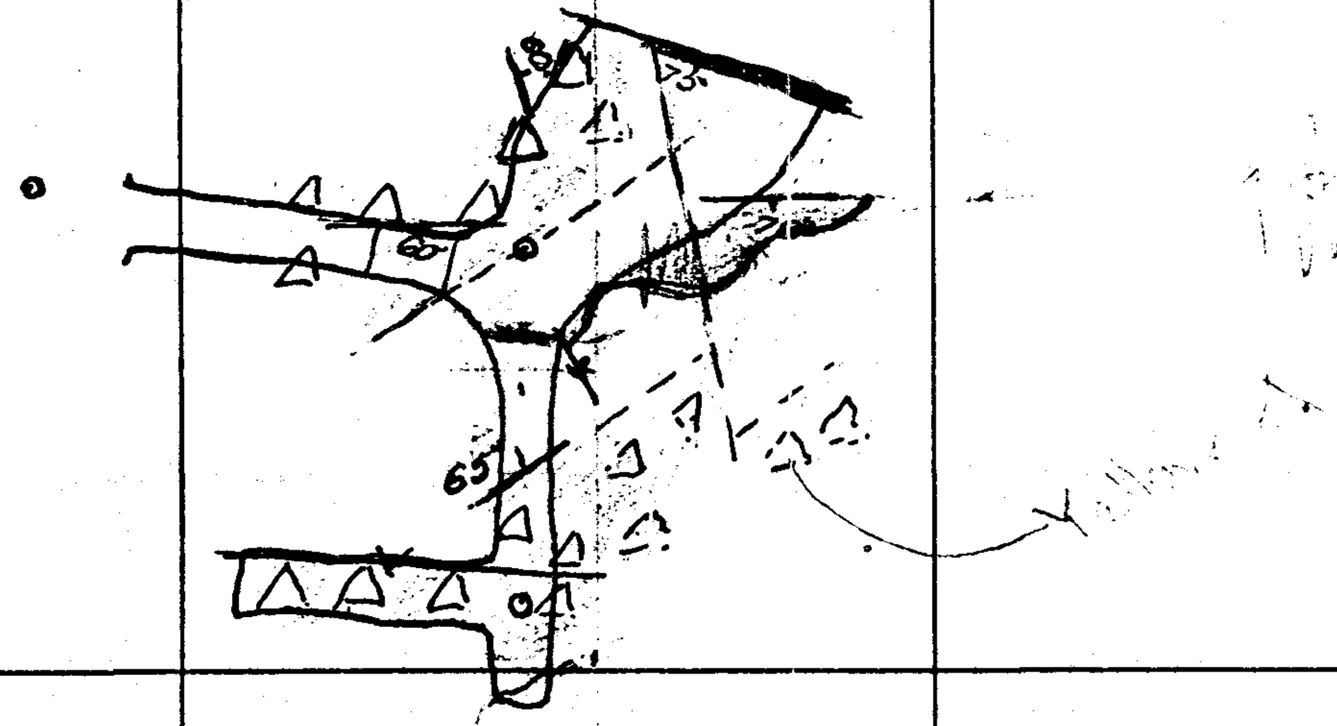
PLAN Nº 3287
RACK Nº ~~21~~ Cd/5

U.P.4

No 3

Open Cut. 25' level

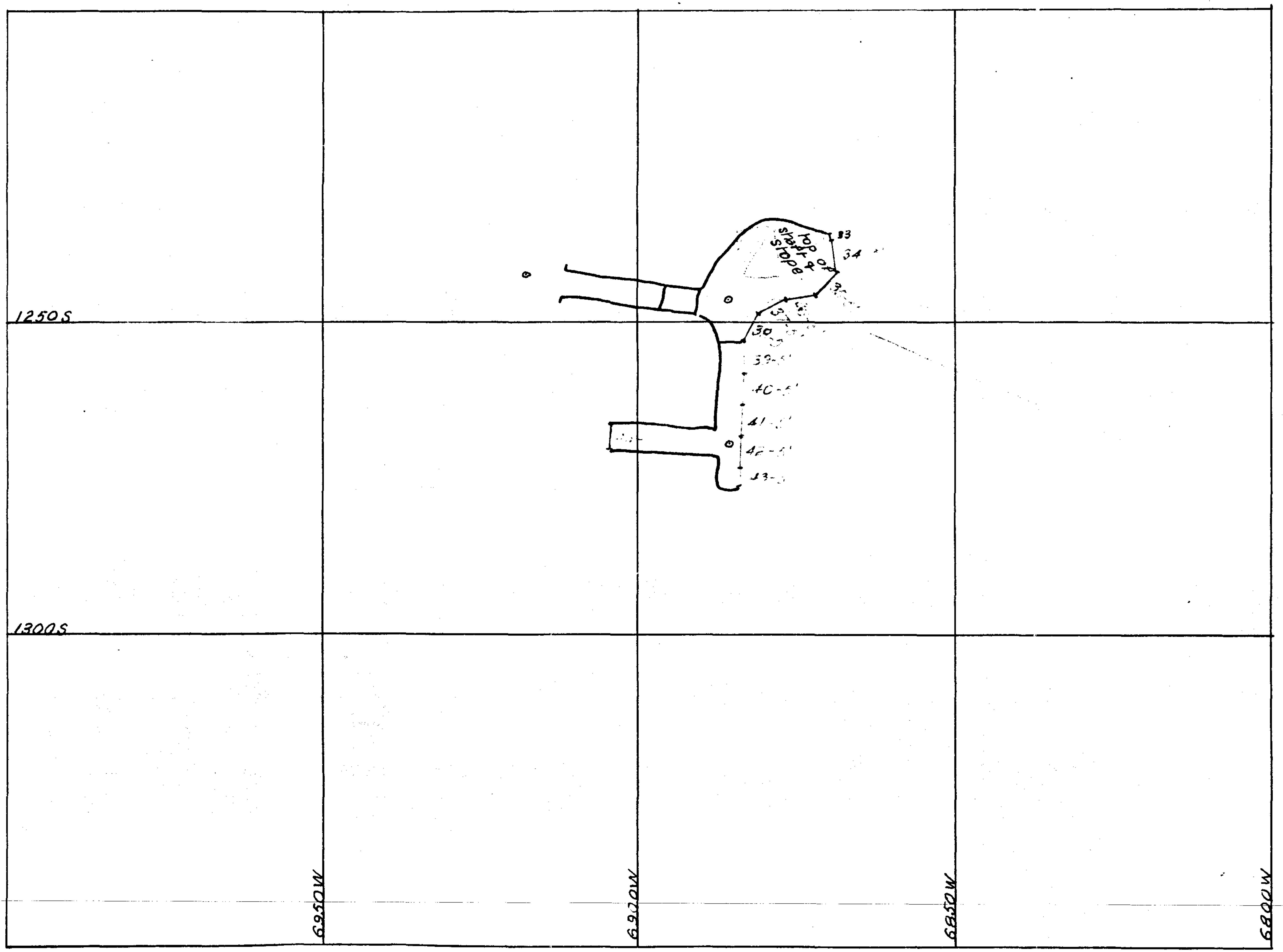
SUP 4 Cd.



Grid incorrect

No [illegible]
No 6 [illegible]
25' Level [illegible]
1" = 20' [illegible]?

UP2



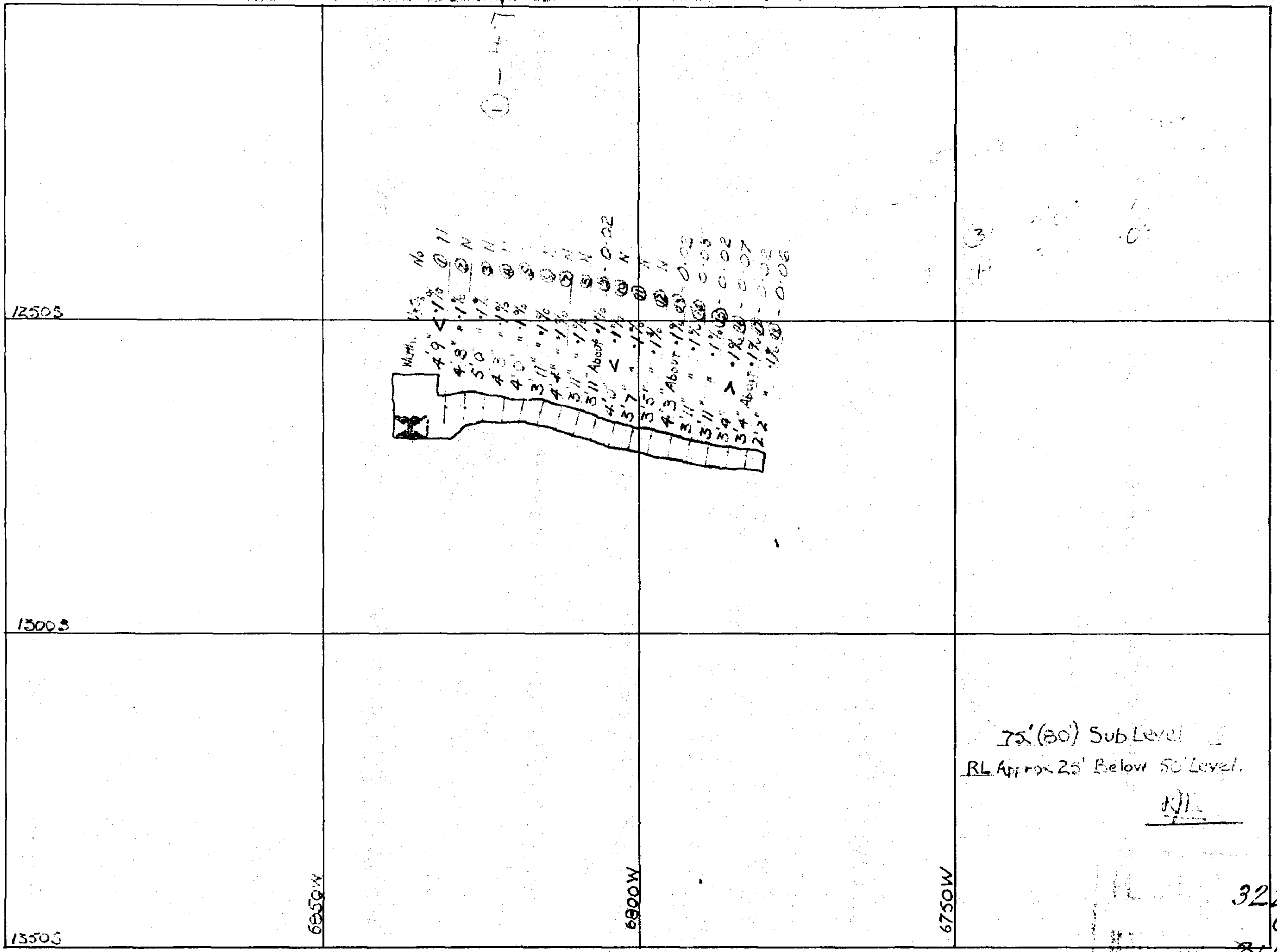
MT. PAINTER
Nº 6 WORKINGS
25' LEVEL

Nº 2813 S

~~[illegible]~~ SUP 2

UP3

SUP 3
cd.
⑧

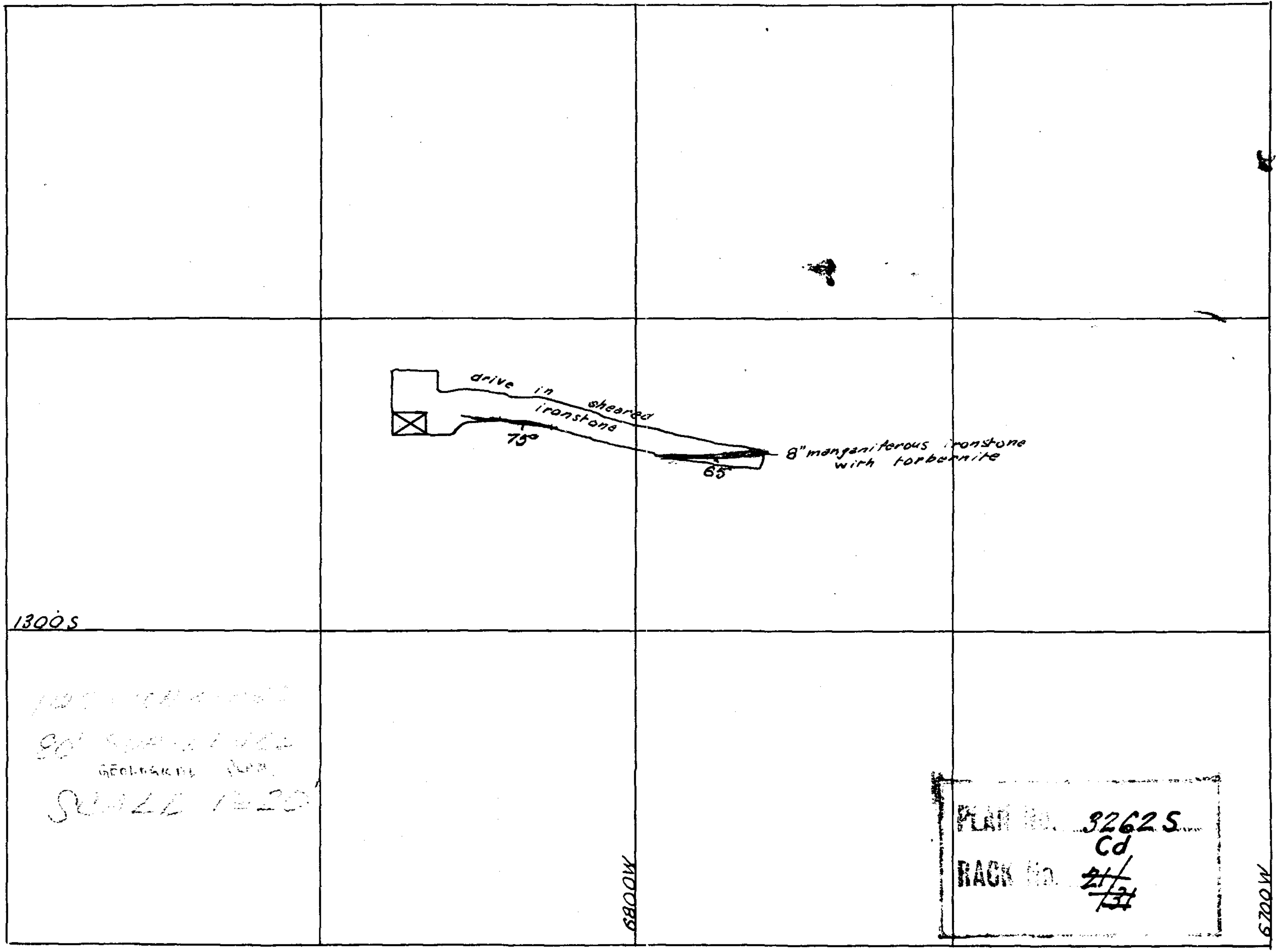


75'(80) Sub Level
RL Approx 25' Below 50' Level.

3222 S
Cd.

~~2/39~~

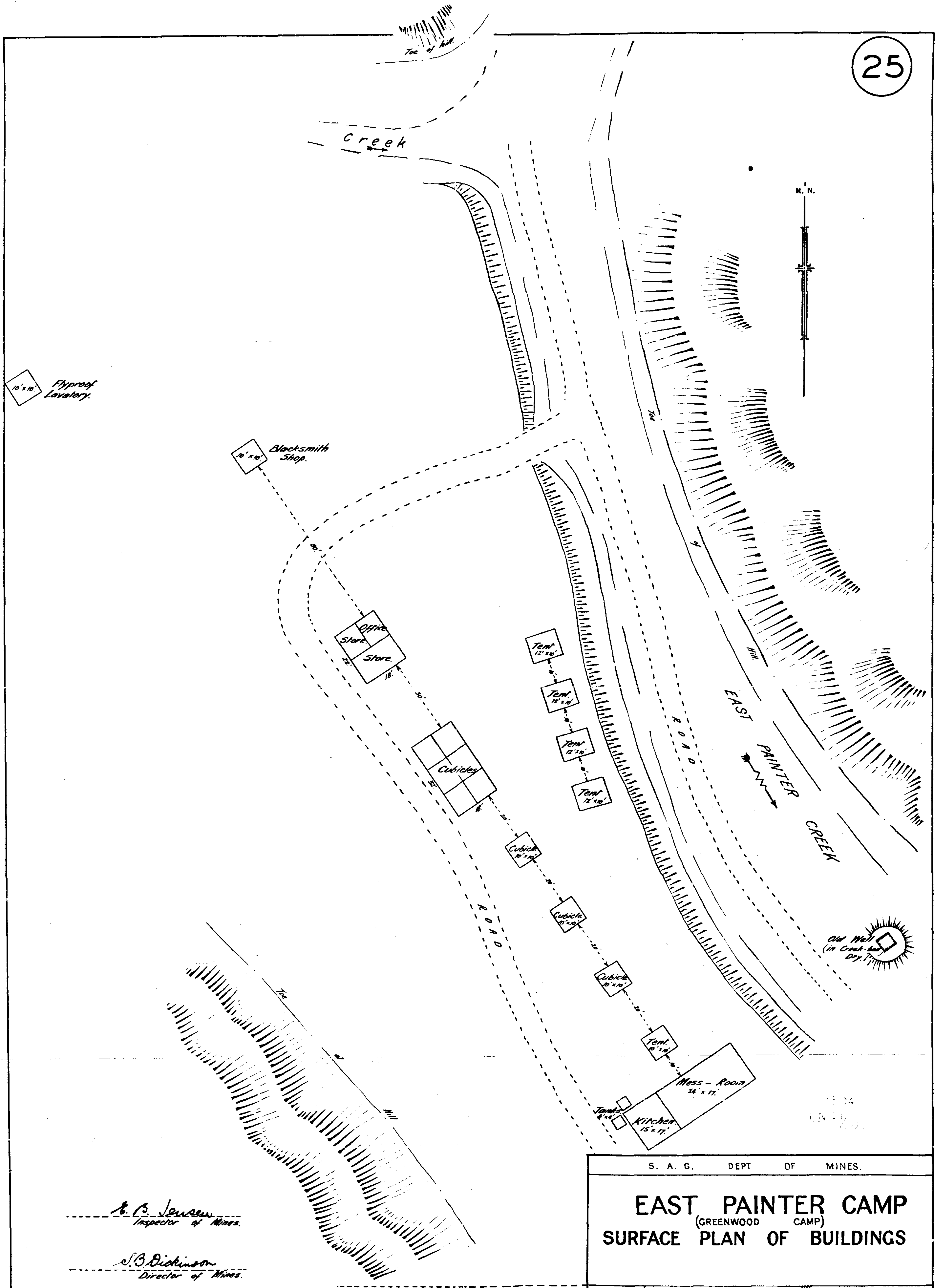
SUP 6
Cd



[illegible]
90' [illegible]
GEOLOGICAL PLAN
SCALE 1" = 20'

PLAN No. 3262 S
Cd
RACK No. ~~21~~/31

25



E. B. Jensen
Inspector of Mines.

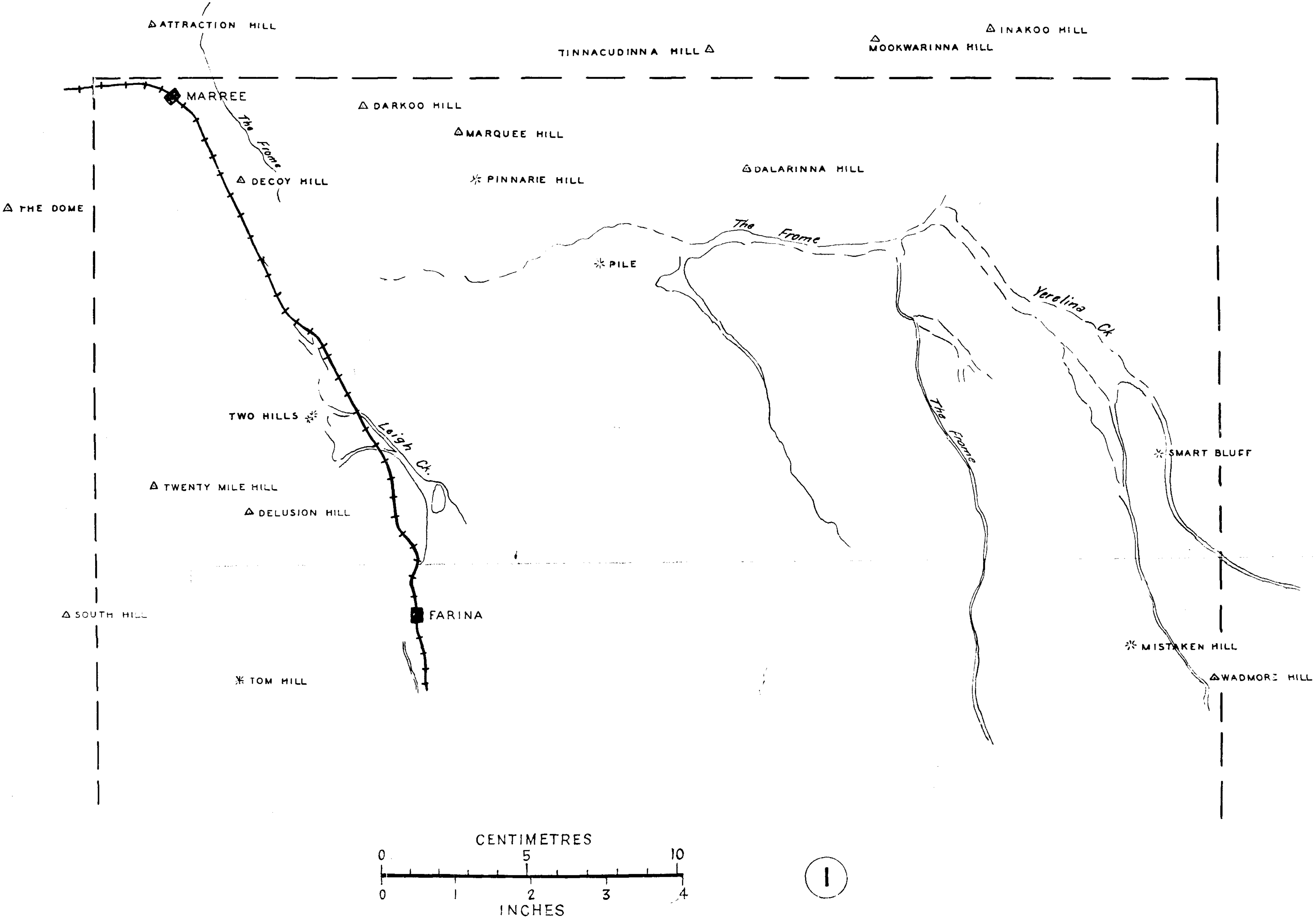
S. B. Dickinson
Director of Mines.

Drawing Reference	Surveyed
Plan Nº 2949	E. B. J.

S. A. G. DEPT OF MINES.

EAST PAINTER CAMP
(GREENWOOD CAMP)
SURFACE PLAN OF BUILDINGS

SCALE	ONE INCH = 20 FEET		PLAN
Passed	Drawn	B. S. G.	UEP. 23 Gd
	Traced	B. S. G	
Date. 4-9-45	Checked	G. T. J.	


ATTRACTION HILL
TINNACUDINNA HILL
MOOKWARINNA HILL
INAKOO HILL
MARREE
DARKOO HILL
MARQUEE HILL
The Frome
DECOY HILL
PINNARIE HILL
DALARINNA HILL
THE DOME
The Frome
PILE
Yerelina Ck
TWO HILLS
Leigh Ck.
The Frome
SMART BLUFF
TWENTY MILE HILL
DELUSION HILL
FARINA
SOUTH HILL
MISTAKEN HILL
WADMORE HILL
TOM HILL
CENTIMETRES
0
5
10
0
1
2
3
4
INCHES

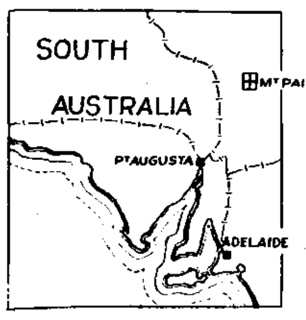
①

SHEET Nº 45b

LOCALITY PLAN

SHEET INDEX

45a	45b
45c	45d



SOUTH AUSTRALIA

DEPARTMENT OF MINES

Mt PAINTER AREA

GEOLOGICAL AND TOPOGRAPHICAL SURVEY

SCALE

Chains 20 15 10 5 0 20 40 60 80 100 120 140 160 Chains
0 ¼ ½ ¾ 1 2 Miles

Cartography by the Draughting and Reproducing Section. Dept. of Mines S.A. 1945.

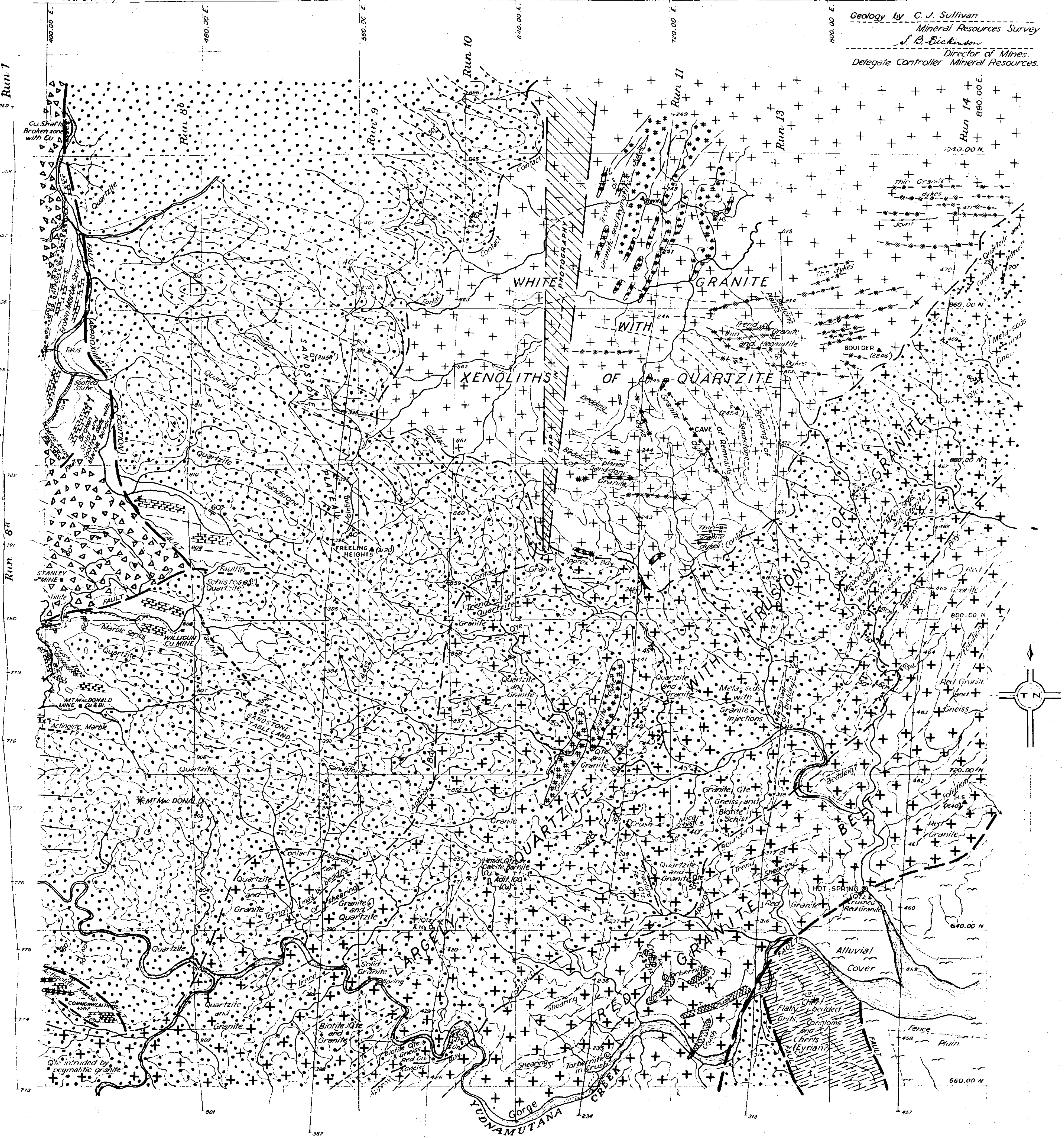
NOTES

Topography compiled from aerial photographs by the Department of Interior. Canberra. 1945.
Aerial photographs by the Royal Australian Air Force.
Geology by the Geological Survey South Australia and the Mineral Resources Survey Canberra.
Trigonometrical Survey by the Lands Department South Australia.

The grid shown is based on an arbitrary datum for Humanity Seat ▲ of 600.00E, 200.00N.
Co-ords. in Links & the grid is in 1 Mile Squares.
Form lines are approx. representations of contours at intervals of 250 Feet above sea level.

Geology by C. J. Sullivan
Mineral Resources Survey
S. B. Dickinson
Director of Mines.
Delegate Controller Mineral Resources.

JOINS MAP 45a



JOINS MAP 45d

Del. B.T. 1945

LEGEND

RECENT – ALLUVIUM .... ∼

TERTIARY – EYRIAN SERIES ....
CHERTS .... ▨

ADELAIDE SERIES (PRE-CAMBRIAN)
- CALCAREOUS SLATES & LIMESTONE ....
- STURTIAN TILLITE .... △▽
- ACTINOLITIC MARBLE ....
- QUARTZITE BED ....
- ARKOSIC QUARTZITE ....
- THICK QUARTZITE .... •

METAMORPHIC ROCKS –
- SCHIST & GNEISS ....
- GNEISS WITH GRANITE INTRUSION ....
- QUARTZITE WITH RED GRANITE INTRUSION ....
- WHITE GRANITE WITH QUARTZITE REMNANTS ....

IGNEOUS ROCKS –
- PEGMATITE .... #
- BASIC IGNEOUS ROCKS V
- WHITE GRANITE .... +
- RED GRANITE .... +

MINERALIZATION –
- HEMATITE .... ▨
- QUARTZ ....
- GEODIC QUARTZ ....

REFERENCE TO SIGNS

GEOLOGICAL BOUNDARY (KNOWN, INFERRED, APPROX) ?
SYNCLINE ....
ANTICLINE ....
FAULT ....
CRUSH ZONE XXXXXXXXXX
BEDDING (INCLINED, VERTICAL) .... 45°
STRIKE & DIP OF CLEAVAGE .... 70°
STRIKE & DIP OF FOLIATION IN GNEISSIC GRANITE .... 35°
WATER HOLE, BORE, WELL © ⊕
PITCH .... 45°
PROMINENT CLEAVAGE TRACES
ROAD, CAMEL PAD ....
FENCE ....
WATERCOURSE ....
FORM LINE ....
TREND LINE OR BEDDING PLANE ....
TRIG. STATION AND R.L. .... ▲ [illegible]
R.L. REFERRED TO SEA LEVEL (APPROX)
MINE, WORKINGS ....
TORBERNITE [illegible]

3289