NUREG/CR-3967
ORNL/TM-9956

An Analysis of Excursions at Selected In Situ Uranium Mines in Wyoming and Texas

Prepared by W. P. Staub, N. E. Hinkle/ORNL
R. E. Williams, F. Anastasi, J. Osiensky, D. Rogness/UI

Oak Ridge National Laboratory

University of Idaho

Prepared for
U.S. Nuclear Regulatory Commission

8607240286 860731
PDR NUREG
CR-3967 R PDR

NOTICE

This report was prepared as an account of work sponsored by an agency of the United States Government. Neither the United States Government nor any agency thereof, or any of their employees, makes any warranty, expressed or implied, or assumes any legal liability of responsibility for any third party's use, or the results of such use, of any information, apparatus, product or process disclosed in this report, or represents that its use by such third party would not infringe privately owned rights.

NOTICE

Availability of Reference Materials Cited in NRC Publications

Most documents cited in NRC publications will be available from one of the following sources:

1. The NRC Public Document Room, 1717 H Street, N.W.
   Washington, DC 20555
2. The Superintendent of Documents, U.S. Government Printing Office, Post Office Box 37082, Washington, DC 20013-7082
3. The National Technical Information Service, Springfield, VA 22161

Although the listing that follows represents the majority of documents cited in NRC publications, it is not intended to be exhaustive.

Referenced documents available for inspection and copying for a fee from the NRC Public Document Room include NRC correspondence and internal NRC memoranda; NRC Office of Inspection and Enforcement bulletins, circulars, information notices, inspection and investigation notices; Licensee Event Reports; vendor reports and correspondence; Commission papers; and applicant and licensee documents and correspondence.

The following documents in the NUREG series are available for purchase from the GPO Sales Program: formal NRC staff and contractor reports, NRC-sponsored conference proceedings, and NRC booklets and brochures. Also available are Regulatory Guides, NRC regulations in the *Code of Federal Regulations*, and *Nuclear Regulatory Commission Issuances*.

Documents available from the National Technical Information Service include NUREG series reports and technical reports prepared by other federal agencies and reports prepared by the Atomic Energy Commission, forerunner agency to the Nuclear Regulatory Commission.

Documents available from public and special technical libraries include all open literature items, such as books, journal and periodical articles, and transactions. *Federal Register* notices, federal and state legislation, and congressional reports can usually be obtained from these libraries.

Documents such as theses, dissertations, foreign reports and translations, and non-NRC conference proceedings are available for purchase from the organization sponsoring the publication cited.

Single copies of NRC draft reports are available free, to the extent of supply, upon written request to the Division of Technical Information and Document Control, U.S. Nuclear Regulatory Commission, Washington, DC 20555.

Copies of industry codes and standards used in a substantive manner in the NRC regulatory process are maintained at the NRC Library, 7920 Norfolk Avenue, Bethesda, Maryland, and are available there for reference use by the public. Codes and standards are usually copyrighted and may be purchased from the originating organization or, if they are American National Standards, from the American National Standards Institute, 1430 Broadway, New York, NY 10018.

NUREG/CR-3967
ORNL/TM-9956

An Analysis of Excursions at Selected In Situ Uranium Mines In Wyoming and Texas

Manuscript Completed: May 1986
Date Published: July 1986

Prepared by
W. P. Staub, N. E. Hinkle, Oak Ridge National Laboratory
R. E. Williams, F. Anastasi, J. Osiensky, D. Rogness, University of Idaho

Oak Ridge National Laboratory
Oak Ridge, TN 37831

Subcontractor:
University of Idaho
Moscow, ID 83843

Prepared for
Division of Waste Management
Office of Nuclear Material Safety and Safeguards
U.S. Nuclear Regulatory Commission
Washington, D.C. 20555
NRC FIN B0279

ABSTRACT

Intensive research and development of in situ uranium mining took place in the United States during the 1970s. By the end of the decade, nearly 10% of all uranium production came from in situ mines. Recent poor market conditions, however, forced the closure of all domestic in situ uranium mines.

When market conditions improve, domestic in situ mining is expected to make a modest recovery. Successful licensing of future in situ mines will depend to a large extent on the ability to contain the leach solutions (lixiviant) within the ore zone. Uncontrolled movement of lixiviant beyond the ore zone is called an excursion.

The industry has demonstrated the capability to reduce, monitor, and control lateral excursions in an ore zone aquifer. On the other hand, vertical excursions of lixiviant migrating into an overlying aquifer through poorly plugged exploration holes continue to plague the industry.

This paper summarizes the history of excursions at selected in situ mines in the United States and discusses methodologies for excursion identification and control. The incidence of vertical excursions can be significantly reduced by certifying well field units through field-wide testing of the ore zone aquifer prior to the injection of lixiviant. An overlying aquifer can be effectively monitored for excursions by periodically analyzing fluids in observation wells for nonreactive ionic species common to the lixiviant. These ions would necessarily have low probabilities of naturally exceeding a given concentration, referred to as an upper control limit. A mining unit would require recertification after each vertical excursion.

TABLE OF CONTENTS

	Page
ABSTRACT	iii
LIST OF FIGURES	ix
LIST OF TABLES	xiii
1. INTRODUCTION	1
1.1 STATEMENT OF THE PROBLEM	1
1.2 PURPOSE AND OBJECTIVES	2
1.3 METHOD OF STUDY	2
1.4 PREVIOUS INVESTIGATIONS	3
2. HYDROGEOLOGY OF IN SITU URANIUM MINE SITES	5
2.1 SEDIMENTARY ENVIRONMENTS OF IN SITU MINE SITES	10
2.1.1 Fluvial Plains	12
2.1.2 Alluvial Fans	12
2.1.3 Beach Barrier and Tidal Flats	16
2.2 HYDROGEOLOGIC TESTING OF IN SITU URANIUM MINE SITES	18
2.2.1 Borehole Logs	19
2.2.2 Aquifer Testing	20
2.3 SUMMARY OF LIMITATIONS IN CHARACTERIZING GROUND-WATER REGIMES	23
3. EXCURSIONS	27
3.1 HORIZONTAL EXCURSIONS	27
3.2 VERTICAL EXCURSIONS	29
3.3 SUMMARY OF EXCURSIONS, THEIR CONSEQUENCES, CAUSES AND CONTROL	31
3.4 AQUIFER TESTS	32
3.5 EXCURSION DETECTION	34
3.5.1 Present Methodologies and Their Shortcomings	34
3.5.2 Selection of Excursion Indicators	36
3.5.3 Selection of Upper Control Limits	42
4. REFERENCES	45
APPENDIX A. CASE HISTORIES	A-1
A.1 WYOMING MINERAL CORPORATION - IRIGARAY PROJECT	A-3
A.1.1 Description of the Site	A-3
A.1.2 History of Mining Operations	A-6
A.1.3 Regional Setting	A-10

		Page
	A.1.3.1 Regional Geology	A-10
	A.1.3.2 Regional Ground-water Flow Systems	A-12
A.1.4	Description and Analysis of Data Collected	A-12
	A.1.4.1 Exploration	A-12
	A.1.4.2 Aquifer Tests	A-13
A.1.5	Hydrogeology of the Mine Site	A-20
A.1.6	Monitoring Well System and Well Construction	A-22
A.1.7	Analysis of Excursions	A-25
A.1.8	Restoration	A-31
A.2	OGLE PETROLEUM, INC. - BISON BASIN PROJECT	A-36
A.2.1	Description of the Site	A-36
A.2.2	History of Mining Operations	A-36
A.2.3	Regional Setting	A-41
	A.2.3.1 Regional Geology	A-41
	A.2.3.2 Regional Ground-water Flow Systems	A-42
A.2.4	Description and Analysis of Data Collected	A-42
	A.2.4.1 Exploration	A-42
	A.2.4.2 Aquifer Tests	A-43
A.2.5	Hydrogeology of the Mine Site	A-49
A.2.6	Monitoring Well System and Well Construction	A-50
A.2.7	Analysis of Excursions	A-55
A.2.8	Restoration	A-59
A.3	ROCKY MOUNTAIN ENERGY COMPANY - NINE MILE LAKE PROJECT	A-62
A.3.1	Description of the Site	A-62
A.3.2	History of Mining Operations	A-62
A.3.3	Regional Setting	A-67
	A.3.3.1 Regional Geology	A-67
	A.3.3.2 Regional Ground-water Flow Systems	A-67
A.3.4	Description and Analysis of Data Collected	A-68
	A.3.4.1 Exploration	A-68
	A.3.4.2 Aquifer Tests	A-68
A.3.5	Hydrogeology of the Mine Site	A-74
A.3.6	Monitoring Well System and Well Construction	A-75
A.3.7	Analysis of Excursions	A-80
A.3.8	Restoration	A-83
A.4	EXXON MINERALS COMPANY - HIGHLAND PROJECT	A-89
A.4.1	Description of the Site	A-89
A.4.2	History of Mining Operations	A-89
A.4.3	Regional Setting	A-95
	A.4.3.1 Regional Geology	A-95
	A.4.3.2 Regional Ground-water Flow Systems	A-95
A.4.4	Description and Analysis of Data Collected	A-96
	A.4.4.1 Exploration	A-96
	A.4.4.2 Aquifer Tests	A-96
A.4.5	Hydrogeology of the Mine Site	A-104
A.4.6	Monitoring Well System and Well Construction	A-109
A.4.7	Analysis of Excursions	A-112
A.4.8	Restoration	A-117

		Page
A.5	TETON EXPLORATION DRILLING COMPANY - LEUENBERGER PROJECT	A-120
	A.5.1 Description of the Site	A-120
	A.5.2 History of Mining Operations	A-120
	A.5.3 Regional Setting	A-124
	A.5.3.1 Regional Geology	A-124
	A.5.3.2 Regional Ground-water Flow Systems	A-124
	A.5.4 Description and Analysis of Data Collected	A-124
	A.5.4.1 Exploration	A-124
	A.5.4.2 Aquifer Tests	A-125
	A.5.5 Hydrogeology of the Mine Site	A-130
	A.5.6 Monitoring Well System and Well Construction	A-134
	A.5.7 Analysis of Excursions	A-138
	A.5.8 Restoration	A-138
A.6	ROCKY MOUNTAIN ENERGY COMPANY - RENO CREEK PROJECT	A-146
	A.6.1 Description of the Site	A-146
	A.6.2 History of Mining Operations	A-146
	A.6.3 Regional Setting	A-150
	A.6.3.1 Regional Geology	A-150
	A.6.3.2 Regional Ground-water Flow Systems	A-150
	A.6.4 Description and Analysis of Data Collected	A-153
	A.6.4.1 Exploration	A-153
	A.6.4.2 Aquifer Tests	A-154
	A.6.5 Hydrogeology of the Mine Site	A-158
	A.6.6 Monitor Well System and Well Construction	A-159
	A.6.7 Analysis of Excursions	A-164
	A.6.8 Restoration	A-168
A.7	CLEVELAND CLIFFS IRON COMPANY - COLLINS DRAW PROJECT	A-175
	A.7.1 Description of the Site	A-175
	A.7.2 History of Mining Operations	A-175
	A.7.3 Regional Setting	A-178
	A.7.3.1 Regional Geology	A-178
	A.7.3.2 Regional Ground-water Flow Systems	A-178
	A.7.4 Description and Analysis of Data Collected	A-181
	A.7.4.1 Exploration	A-181
	A.7.4.2 Aquifer Tests	A-181
	A.7.5 Hydrogeology of the Mine Site	A-185
	A.7.6 Monitor Well System and Well Construction	A-185
	A.7.7 Analysis of Excursions	A-188
	A.7.8 Restoration	A-193
A.8	WYOMING MINERAL CORPORATION - BRUNI PROJECT	A-195
	A.8.1 Description of the Site	A-195
	A.8.2 History of Mining Operations	A-195
	A.8.3 Regional Setting	A-202
	A.8.3.1 Regional Geology	A-202
	A.8.3.2 Regional Ground-water Flow Systems	A-202
	A.8.4 Description and Analysis of Data Collected	A-203
	A.8.4.1 Exploration	A-203
	A.8.4.2 Aquifer Tests	A-203

Page

A.8.5 Hydrogeology of the Mine Site A-206
A.8.6 Monitor Well System and Well Construction A-206
A.8.7 Analysis of Excursions A-215
A.8.8 Restoration A-215
A.9 APPENDIX A REFERENCES A-223

LIST OF FIGURES

Figure		Page
2.1	Cross section of a typical uranium ore deposit and the solution mining wells	6
2.2	Typical uranium roll-front deposit	7
2.3	Simplified illustration of a sedimentary environment susceptible to excursions	9
2.4	Location map of the eight in situ uranium mine sites	11
2.5	Block diagram showing main features of a meandering river and its floodplain	14
2.6	Block diagram of a braided sandy system with low sinuosity channels	15
2.7	Distribution of sediment facies and morphological profiles of an ideal alluvial fan	17
2.8	Alternate interpretations of the same borehole data	24
3.1	Comparison of pathways for horizontal and vertical excursions	41
A.1	Location map of the eight in situ uranium mine sites	A-4
A.2	Location map of the Wyoming Mineral Corporation Irigaray mine	A-7
A.3	Irigaray uranium mine site and immediate vicinity	A-8
A.4	Research and development well fields at the Irigaray site	A-9
A.5	Regional geology of the Powder River Basin	A-11
A.6	Location map of pumping wells for the aquifer tests conducted at the Irigaray mine site	A-16
A.7	Stratigraphic section for the Irigaray project area	A-21
A.8	Simplified cross section through the Irigaray site	A-23
A.9	Location of trend wells and monitoring wells reported on excursion status at the Irigaray mine	A-27
A.10	Location map of the Ogle Petroleum, Inc. Bison Basin mine	A-37
A.11	Drainage patterns and their relationship to the Bison Basin project site	A-38

Figure		Page
A.12	Topographical map of the Bison Basin project site	A-39
A.13	Research and development well field at the Bison Basin site	A-40
A.14	Location map of pumping/injection wells for the aquifer tests conducted at the Bison Basin mine site	A-45
A.15	Stratigraphic section for the Bison Basin project area . . .	A-51
A.16	Surficial geology and structure map of the Bison project area	A-52
A.17	Geologic cross section through the Bison Basin mine site . .	A-53
A.18	Location of faults transecting the ore body at the Bison Basin mine	A-54
A.19	Location of monitoring wells reported on excursion status at the Bison Basin mine	A-56
A.20	Location map of the Rocky Mountain Energy Company Nine Mile Lake mine	A-63
A.21	Map of the Nine Mile Lake uranium project site and immediate vicinity	A-64
A.22	Research and development test patterns at the Nine Mile Lake site	A-65
A.23	Location map of pumping wells for the aquifer tests conducted at the Nine Mile Lake mine site	A-70
A.24	Surficial geology and structure map of the Nine Mile Lake project area	A-76
A.25	Geologic cross sections through the Nine Mile Lake mine site	A-77
A.26	Stratigraphic section for the Nine Mile Lake project area .	A-78
A.27	Location of monitoring wells reported on excursion status at the Nine Mile Lake research and development site	A-82
A.28	Location map of the Exxon Minerals Company Highland mine . .	A-90
A.29	Map of the Highland mine operations	A-91
A.30	Location map of wells at the original pilot solution mine field of the Highland site	A-92

Figure		Page
A.31	Location map of wells at the expanded solution mine well field of the Highland site	A-94
A.32	Water level contour map for September 1, 1978, at the Highland mine	A-98
A.33	Water level contour map for September 7, 1978, at the Highland mine	A-99
A.34	Water level elevations prior to the pump test in the Section 24 mine area	A-102
A.35	Generalized stratigraphic column of the Highland mine area	A-105
A.36	Highland ore deposit, showing relationship to local structures	A-106
A.37	Geologic cross section A-A' in the vicinity of the pilot mine well fields	A-107
A.38	Geologic cross section B-B' in the vicinity of the pilot mine well fields	A-108
A.39	Location map of the Teton Exploration Drilling Company Leuenberger mine	A-121
A.40	Leuenberger uranium project site and immediate vicinity	A-122
A.41	Research and development well field at the Leuenberger site	A-123
A.42	Location map of the four ore zone pumping tests and two Basal recovery tests at the Leuenberger site	A-127
A.43	Stratigraphic section for the Leuenberger project area	A-131
A.44	Simplified geologic cross section through the Leuenberger mine site	A-132
A.45	Surficial geology and structure map of the region surrounding the Leuenberger site	A-133
A.46	Map of the Leuenberger project area showing locations of mining unit I restoration baseline and excursion monitor wells	A-137
A.47	Schematic of a typical injection and recovery well completion at the Leuenberger project site	A-139
A.48	Location map of the Rocky Mountain Energy Company Reno Creek mine	A-147

Figure		Page
A.49	Reno Creek mine plan	A-148
A.50	Location map of wells in Pattern I at the Reno Creek mine	A-149
A.51	Location map of wells in Pattern II at the Reno Creek mine	A-151
A.52	Geologic cross section through Pattern II at the Reno Creek mine	A-160
A.53	Generalized stratigraphic column for the Reno Creek project area	A-161
A.54	Typical construction of monitor wells at the Reno Creek mine	A-163
A.55	Location map of the Cleveland Cliffs Iron Company Collins Draw mine	A-176
A.56	Location map of the facilities at the Collins Draw mine	A-177
A.57	Location map of monitor wells and production/injection wells at the Collins Draw mine	A-179
A.58	Cross section through the Collins Draw mine area	A-186
A.59	Generalized stratigraphic section for the Collins Draw project area	A-187
A.60	Typical well completion at the Collins Draw mine	A-191
A.61	Location map of the Wyoming Mineral Corporation Bruni mine	A-198
A.62	Location map of the Bruni mine facilities	A-200
A.63	Simplified geologic cross section through the Bruni mine site	A-207
A.64	Generalized stratigraphic section for the Bruni project area	A-208
A.65	Surficial geologic map of the Bruni project area	A-209
A.66	Location map of the monitor wells at the Bruni mine site	A-211
A.67	Typical well completion methods for the Bruni mine site	A-214
A.68	Location and dates of the shallow zone contamination at the Bruni mine	A-218

LIST OF TABLES

Table		Page
2.1	Sedimentary characteristics, fluvial depositional environments of uranium districts in the western United States . .	13
3.1	Typical monitor well spacings for ore zone aquifers and densities in upper and lower aquifers as related to project size	35
3.2	Generalized suitability of excursion indicators	39
A.1	Summary of major characteristics at seven in situ uranium mines in Wyoming and one mine in Texas	A-5
A.2	Summary of aquifer tests conducted at the Irigaray mine . .	A-15
A.3	Summary of aquifer and aquitard characteristics estimated by Wyoming Mineral Corporation and by the authors from pumping/injection test data for the Irigaray mine	A-17
A.4	Summary of excursions at the Irigaray mine site during mining operations	A-26
A.5	Water quality before and after restoration tests at the 517 well field of the Irigaray site	A-33
A.6	Water quality for the well field recovery solutions before, during, and after restoration at Well Field E of the Irigaray site	A-35
A.7	Summary of aquifer tests conducted at the Bison Basin mine .	A-44
A.8	Summary of aquifer and aquitard characteristics estimated by Ogle Petroleum, Inc., Hydro-Engineering, and the authors from aquifer test data for the Bison Basin mine	A-46
A.9	Summary of excursions at Bison Basin mine site during mining operations	A-58
A.10	Water quality in wells before, during, and after restoration at the research and development pilot site at Bison Basin .	A-61
A.11	Summary of aquifer tests conducted at the Nine Mile Lake mine	A-69
A.12	Summary of aquifer and aquitard characteristics estimated by D'Appolonia Consulting Engineers, Inc., In-Situ Consulting, and the authors from aquifer test data for the Nine Mile Lake mine	A-72

Table		Page
A.13	Summary of excursions at the Nine Mile Lake mine site during mining operations	A-81
A.14	Water quality in wells before and after restoration tests in Pattern 1 at the Nine Mile Lake mine	A-84
A.15	Water quality in wells before, during, and after restoration in Pattern 2 at the Nine Mile Lake mine	A-86
A.16	Water quality in wells before and after restoration in Pattern 3 at the Nine Mile Lake mine	A-88
A.17	Lixiviant compositons at the Highland mine	A-93
A.18	Results of core and injection tests conducted at the Highland mine site	A-97
A.19	Summary of aquifer and aquitard characteristics estimated by Exxon Minerals Company and the authors from aquifer test data for the Highland mine	A-103
A.20	Well completion information for the Highland mine	A-110
A.21	Baseline water quality at the Highland original pilot test site	A-113
A.22	Summary of excursion events at the Highland mine site	A-115
A.23	Post-restoration water quality data for the original pilot well field in 1983	A-119
A.24	Summary of aquifer tests conducted at the Leuenberger mine	A-124
A.25	Summary of aquifer and aquitard characteristics estimated by Teton Exploration Drilling Company and the authors from aquifer test data for the Leuenberger mine	A-128
A.26	Data used to establish preliminary mean baseline values for mining unit I at the Leuenberger mine site	A-135
A.27	Water quality in wells before, during, and after N-zone restoration at the Leuenberger mine site	A-140
A.28	Ground-water restoration-test water quality data in the N ore zone of the Leuenberger mine site	A-142
A.29	Water-quality in wells before, during, and after M-zone restoration at the Leuenberger mine site	A-143

Table		Page
A.30	Ground-water post-leaching water quality data in the M ore zone of the Leuenberger mine site	A-145
A.31	Characteristics of lixiviant used in leaching Pattern II at the Reno Creek mine	A-152
A.32	Summary of aquifer tests conducted at the Reno Creek mine	A-155
A.33	Test well information for the Reno Creek mine	A-156
A.34	Summary of aquifer characteristics estimated by In-Situ Consulting and by the authors from pumping/injection test data for the Reno Creek mine	A-157
A.35	Baseline water quality for Pattern I at the Reno Creek mine	A-165
A.36	Excursion control limits for monitor wells at the Reno Creek mine	A-166
A.37	Pattern II baseline water quality for monitor wells at the Reno Creek mine	A-167
A.38	Pattern I ground-water quality during leaching and restoration at the Reno Creek mine	A-170
A.39	Water quality data for Pattern I for first quarter, 1983, at Reno Creek mine	A-172
A.40	Water quality data for the Pattern II production zone at the Reno Creek mine	A-173
A.41	Comparison of uranium concentrations in Pattern II wells at the Reno Creek mine	A-174
A.42	Chemical analyses of lixiviant and produced solution in the B well field at the Collins Draw mine	A-180
A.43	Summary of aquifer tests conducted at the Collins Draw mine	A-182
A.44	Summary of aquifer characteristics estimated by In-Situ Consulting and by the authors from pumping/injection data for the Collins Draw mine	A-184
A.45	Average baseline concentrations for the AB-Sand, 1-Sand, and Stray Sand aquifers at the Collins Draw mine	A-189
A.46	Excursion indicators and upper control limits for the Collins Draw mine	A-190

Table		Page
A.47	Summary of excursions at the Collins Draw mine	A-192
A.48	Comparison of water quality before and after restoration of the A-1 well field at the Collins Draw mine	A-196
A.49	Comparison of water quality before and after restoration of the B well field at the Collins Draw mine	A-197
A.50	Typical composition of injected lixiviant at the Bruni mine	A-201
A.51	Summary of aquifer tests conducted at the Bruni mine	A-204
A.52	Summary of aquifer and aquitard characteristics estimated by Wyoming Mineral Corporation and by the authors from aquifer test data for the Bruni mine	A-205
A.53	Average baseline water quality for the Bruni mine	A-212
A.54	Excursion upper control limits for the Bruni mine	A-213
A.55	Summary of excursions at the Bruni mine during mining operations	A-216
A.56	Summary of the 1977 restoration test at the Bruni mine . . .	A-220
A.57	Water quality following the restoration demonstration project in Well Field AB at the Bruni mine	A-221

1. INTRODUCTION

1.1 STATEMENT OF THE PROBLEM

In situ uranium mining has been developed in preference to conventional open pit and underground mining for low grade ore deposits. Wyoming and southern Texas were the prominent areas for early field experiments and in subsequent commercial-scale production. In situ mining generally is environmentally and economically favorable to other methods of mining.

Stratigraphy of most in situ uranium mine sites consists of interbedded layers of sand, silt, and clay with sinuous deposits of sand and/or gravel, deposited in fluvial environments. Facies changes (both laterally and vertically), erosional channels, scour and fill deposits, and zones of high hydraulic conductivity often exist at in situ uranium mine sites. These natural factors and man-made factors, such as unplugged boreholes and poorly sealed wells, complicate in situ mining of uranium.

The in situ mining process involves injection of leaching solution (lixiviant) into an ore zone aquifer to dissolve uranium-bearing minerals. The lixiviant consists of water containing an oxidant and an ionic complexing agent. The oxidant converts uranium from the reduced (+4) to the oxidized (+6) valence state, so that dissolution and complexation can readily occur. Common oxidants are oxygen and hydrogen peroxide; common complexing agents are the sulfate ion in a sulfuric acid lixiviant and carbonate and bicarbonate ions (ammonium carbonate/bicarbonate or sodium carbonate/bicarbonate) in an alkaline lixiviant. The uranium-rich solution is pumped from the ore zone by recovery wells, stripped of its uranium by ion exchange, and reinjected into the ore zone through injection wells.

One of the major problems associated with the in situ mining method is the uncontrolled migration of lixiviant and dissolved constituents such as radionuclides, arsenic, selenium, chromium and lead outside of the production zone. These undesirable lixiviant migrations are known as excursions. Horizontal and/or vertical excursions have occurred at many of the in situ uranium mine sites in the western United States, and are recognized by mining companies as events that should be prevented.

Excursions are a result of both natural and man-made causes. However, the absence of critical data often prevents the complete understanding of the causes. In these cases, excursion clean-up can be very expensive and sometimes impossible. The development of a sound data base on the hydraulic properties of the aquifer and confining layers, hydrogeologic characteristics of faults, and the condition of abandoned drill holes prior to mining is essential to excursion prevention and control. This study is being conducted to obtain a better understanding of the hydrogeologic factors that control excursions and to develop proper methods of hydrogeologic evaluation of uranium in situ mine sites located in the United States.

1.2 PURPOSE AND OBJECTIVES

The purpose of this study is to obtain a better understanding of excursions at in situ uranium mines. The general objectives are to evaluate the hydrogeologic factors that control excursions, to develop hydrogeologic evaluation guidelines that should be followed in order to achieve a thorough understanding of the hydrogeologic characteristics at potential or existing in situ uranium mine sites. Proper understanding of the hydrogeology prior to mining a given area is essential to prevent excursions from the production zone during mining.

Specific objectives include:

1. Analyze existing hydrogeologic data and develop conceptual models of the hydrogeologic environment at seven in situ uranium mine sites in Wyoming and one site in Texas.

2. Analyze the excursions that have occurred at the mine sites, along with their causes.

3. Compare and critique the methods of hydrogeologic data collection at each of the sites to determine if they were adequate for their intended purpose(s).

4. Discuss the use of aquifer testing to assess the potential for excursions.

5. Develop and recommend hydrogeologic testing procedures and methods of analysis that can characterize site hydrogeology sufficiently to identify reliable barriers to excursions.

1.3 METHOD OF STUDY

A major portion of this investigation consisted of the review and analysis of existing hydrogeologic data for seven in situ uranium mines in Wyoming and one site in Texas. Most of the data for the Wyoming sites were collected by the mining companies and their consultants in response to requirements by the U.S. Nuclear Regulatory Commission (NRC) and the Wyoming Department of Environmental Quality for operating licenses. These data were supplied for this investigation by the NRC; a limited amount of information was supplied directly by the mining companies. Most of the data for the Texas site were collected by the mining company and its consultants in response to requirements by the State of Texas. These data were supplied for this investigation by the Texas Department of Water Resources. Site selection for the case histories (summarized in Appendix A) was based on data availability and not on the number of excursions that occurred at a given site.

The remaining information for this investigation was derived from the review and analysis of publications that pertain to uranium deposition, in situ uranium mining, and various aspects of hydrogeologic analysis.

1.4 PREVIOUS INVESTIGATIONS

Numerous geologic and hydrogeologic studies have been conducted for in situ uranium mine sites by mining companies and their consultants in Wyoming and Texas. The information ordinarily is supplied to the NRC and/or the Wyoming Department of Environmental Quality and the Texas Department of Water Resources, respectively. The NRC conducts environmental assessments for each mine site in Wyoming prior to the initiation of leaching activities. All regulatory authority in Texas pertaining to in situ uranium mining is vested in that state's Department of Water Resources.

Although many papers have been written about in situ uranium mining, very little has been reported regarding excursions, hydrogeologic controls on excursions, and hydrogeologic evaluation methods for characterizing in situ mine sites with respect to potential excursions. Most papers provide general environmental considerations of in situ mining with little discussion of excursions in case studies.

The U.S. Bureau of Mines has published many reports dealing with in situ uranium mining. Larson (1980) and Kasper et al (1979) discussed environmental controls of in situ mining. O'Rourke et al (1977) prepared "Field Permeability Test Methods with Applications to Solution Mining" for the U.S. Bureau of Mines. The NRC (1981, 1982) have presented "Groundwater Monitoring at Uranium In Situ Solution Mines" and "Hydrogeologic Characterization of Uranium Solution Mine and Mill Tailings Disposal Sites" as staff technical position papers. These two papers provide general guidance on hydrogeologic characterization and monitoring for in situ mine sites. Finally, Bishop (1980) outlined situations at in situ uranium mines that may contribute to excursions and suggested methods applicable to detecting the causes of non-production zone excursions.

This report uses the above sources of information whenever possible in the analysis of the eight case histories.

2. HYDROGEOLOGY OF IN SITU URANIUM MINE SITES

In situ uranium mining was developed primarily as a method for extracting uranium from sandstone uranium ores that are too deep, too small or too low grade to be mined economically by any other mining method. These uranium ore deposits consist predominantly of mineralized bodies of sandstone that contain uranium minerals which were precipitated out of solution or flocculated from colloidal suspensions. Uranium occurs as mineral coatings on sand grains or as precipitated deposits within interstitial pores (Bailey and Childers, 1977).

In situ uranium mining is used to extract uranium from permeable sandstone ore deposits approximately 30 to 180 m (100 to 600 ft) below ground surface. If in situ methods are to be applied, the uranium-bearing sandstone must be separated hydraulically from other aquifers by low hydraulic conductivity clay or shale units (Fig. 2.1). Uranium is present as an insoluble uranium oxide in roll-front type deposits (Fig. 2.2). Such roll-front deposits occur as a result of uranium deposition by ground water along an oxidation-reduction front.

The ore deposits in Wyoming and Texas occur in sandstone units in several configurations such as cresent-shaped along channel margins, tabular-shaped along hydraulic conductivity changes, and ellipitical- or dish-shaped within scour pockets. Uranium was carried in solution by ground water from source areas, through the sandstone aquifer, to the roll-front where uranium and associated minerals were precipitated. Distant weathered, granitic intrusives or overlying sedimentary rocks containing tuffs are considered probable sources of uranium in the Wyoming and Texas roll-front deposits. The zone of deposition occurred at the roll-front as a result of a reduction in oxidation potential (Eh) and pH in the solution, generally due to the presence of carbonaceous material, native sulfur and pyrite at that location in the aquifer. Nearly all roll-front deposits are below the water table, thereby being unaffected by recent oxidation and redistribution; uraninite and coffinite are the principal ore minerals.

Uranium ore bodies are typically associated with the coarser grained clastic sediment within a given mining district. Those of the Gas Hills District (Wyoming) accumulated in the East Canyon, Dry Coyote, and Muskrat boulder conglomerates, members of the Eocene Wind River Formation (Soister, 1968). Uranium ore in the Powder River Basin, Wyoming, is found in the coarser sandstone members of the Eocene Wasatch Formation. Sandstones in the Wasatch are generally barren where they contain appreciable amounts of silt and clay-sized material. On the other hand, concretionary sandstones are generally free of fine grained materials and they often contain uranium (Sharp and Gibbons, 1964). Fluid flow in and around Wasatch ore bodies is complicated by wide variations in permeability resulting from the presence of carbonate concretions and highly variable silt and clay content within the sandstone units. In situ mining of Wasatch ore bodies present challenging but not insurmountable problems in lixiviant control, whereas lixiviant excursion could never be controlled in the boulder conglomerates

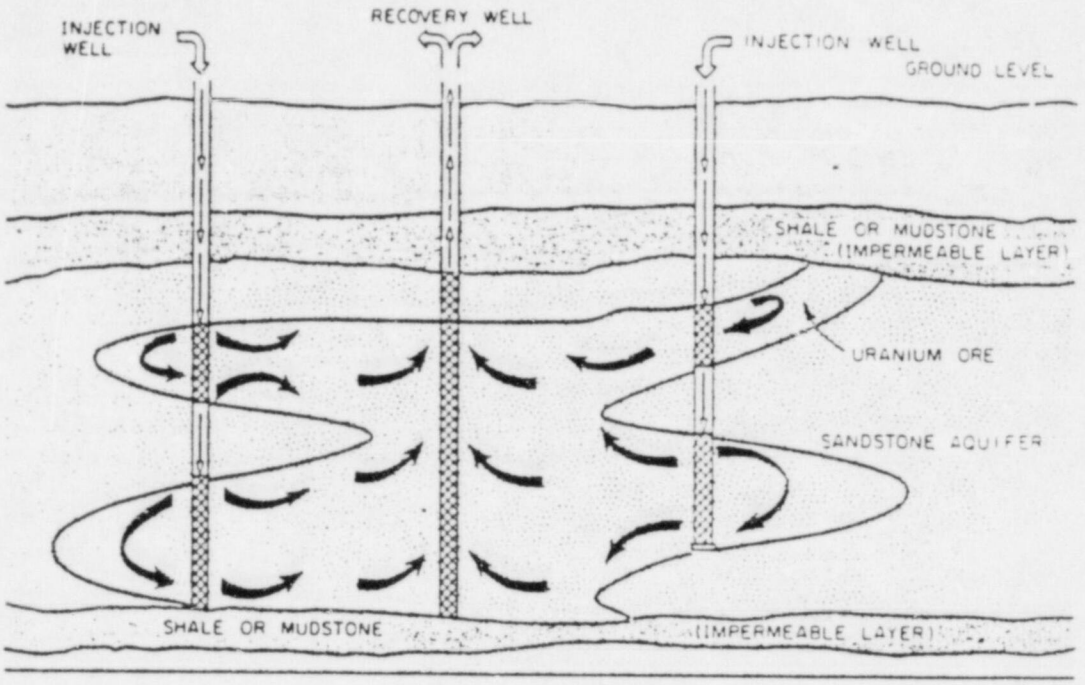


Fig. 2.1. Cross section of a typical uranium ore deposit and the solution mining wells. *Source.* NRC, 1978.

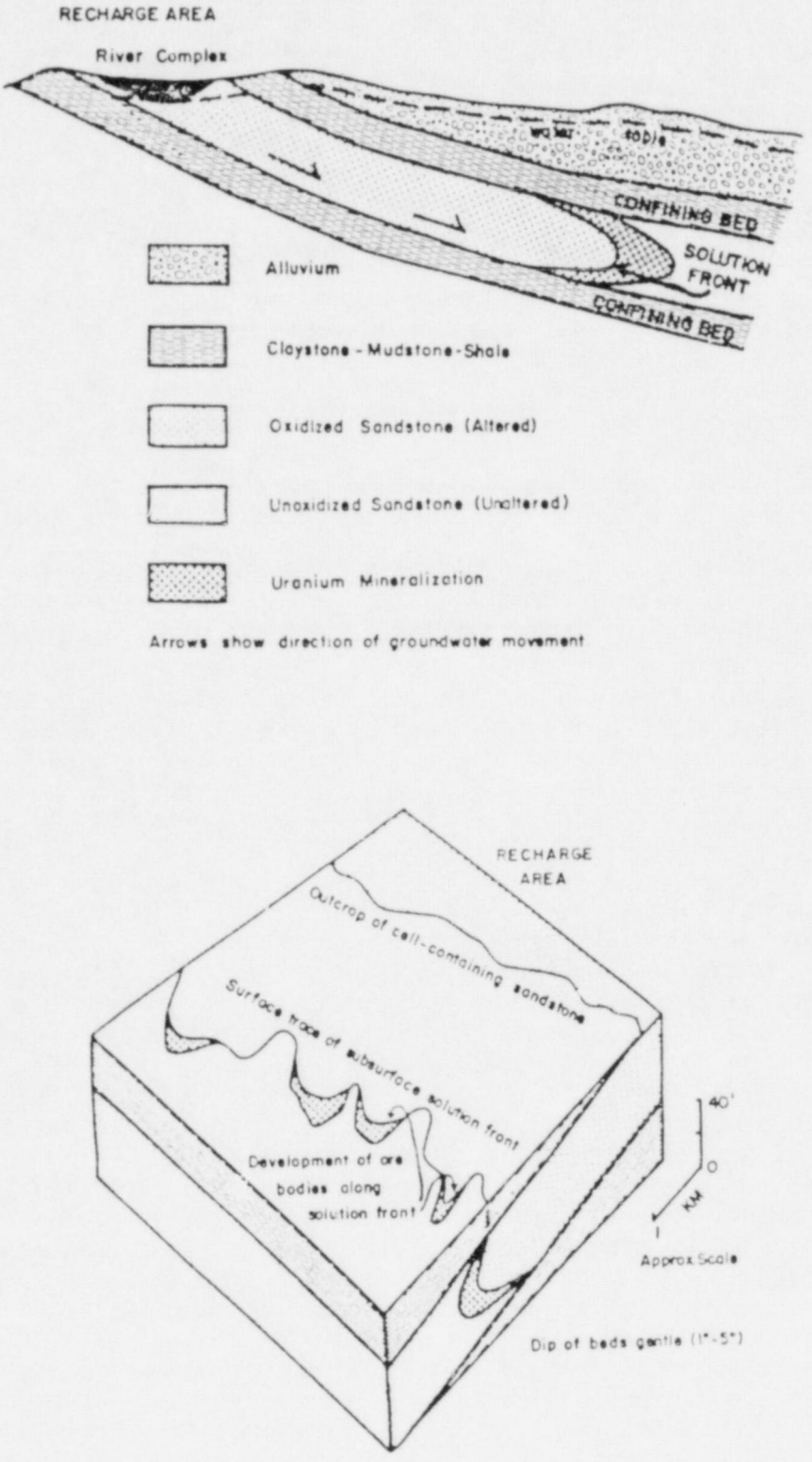


Fig. 2.2. Typical uranium roll-front deposit. *Source:* Campbell and Biddle, 1977.

of the Wind River Formation. To date, no in situ mines have ever been developed in the latter, whereas several ore bodies have been mined by in situ methods in the Wasatch.

A detailed understanding of the site-specific sedimentary environment and associated stratigraphy at in situ uranium mine sites is important to the evaluation and interpretation of excursions. The extent to which sedimentary deposits at in situ mine sites are heterogeneous and anisotropic (directional non-uniformity of specific physical properties) may influence the degree to which leaching solutions can be controlled within the production zone. Post-depositional changes due to structural or erosional instability may have significantly influenced the character of the stratigraphy. The stratigraphy of an existing or proposed in situ uranium mine site should be well defined both laterally and vertically in order to analyze the suitability of that site for leaching of uranium.

Depositional mode determines the geometry of a sedimentary unit; the geometry establishes the degree of lateral and vertical homogeneity and the hydraulic bounding conditions. Fluvial sands are neither uniform nor do they vary in a uniform manner. Alluvial fan deposits also are not uniform in character; they contain numerous abrupt lateral and vertical facies changes. Lacustrine sediments deposited in the central portion of a lake generally are uniform and laterally consistent; sediments at the lake margins are less uniform in character. Delta sands are deposited in a radiating network of channels bounded by levee silts and clays. Clastic shoreline deposits differ in character between transgressions and regressions of the sea, and the deposits vary considerably from the shoreline out to the marine shelf. The reader is referred to Selley (1978) for further discussions of sedimentary environments.

The above descriptions illustrate that some sedimentary environments may be more susceptible to excursions than others. Fig. 2.3 is a simplified illustration of potential excursion pathways. An ideal stratigraphic sequence for in situ mining is a homogeneous and isotropic uranium-bearing sandstone bounded by uniformly thick clay or shale units. Such a sequence is not found in nature. Stratigraphic units deposited in braided river systems, alluvial fans, and deltas probably are the least desirable units in which to mine uranium in situ. These units tend to be heterogeneous, thereby facilitating excursions. Lacustrine and deep marine sedimentary environments probably are the most desirable for in situ mining of uranium. Deep marine environments are not known to have existed at any in situ uranium sites in Wyoming or Texas. All of the sites examined in this study are fluvial systems except for the Nine Mile Lake mine. The ore zones in the latter case were developed in sandstones of shallow marine origin.

Post-depositional changes in the sedimentary deposits may significantly affect stratigraphic continuity. Erosional channels (scour and fill deposits) in confining units may provide preferential pathways of higher hydraulic conductivity for vertical and lateral lixiviant migration (Fig. 2.3). Scour and fill deposits may occur between boreholes and remain undetected during exploration; these discontinuities may exist in apparently continuous strata. Faulted and fractured areas are especially

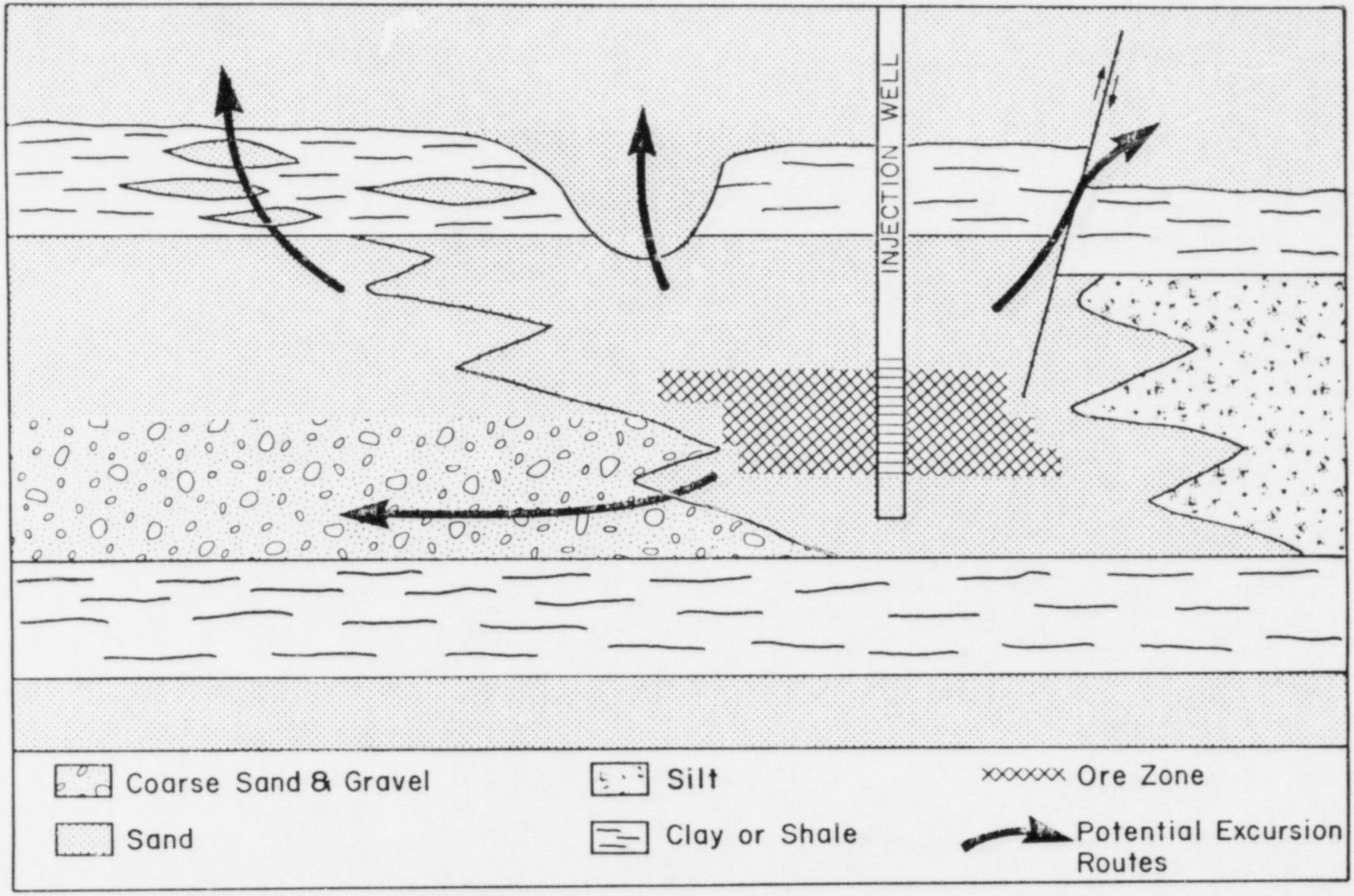


Fig. 2.3. Simplified illustration of a sedimentary environment susceptible to excursions.

vulnerable to excursions because zones of potentially high vertical hydraulic conductivity may exist. The exploration and testing programs at existing and potential in situ uranium mine sites must investigate the stratigraphy in detail so that preferential pathways for fluid movement in the system can be detected.

Six of the seven in situ uranium mines in Wyoming described in Appendix A are located in the southern portion of the Powder River Basin; the other site (Ogle Petroleum, Inc.) is located in the Bison Basin (Fig. 2.4). The remaining site in this study is located in the Texas Coastal Plain.

Periods of marine transgression and regression occurred during Cretaceous time in much of Wyoming; deposition of thin layers of marine sands and carbonates interbedded with non-marine clastic sediments resulted. The Laramide orogeny began during late Cretaceous time along the western continental margin and controlled structural development in Wyoming. After withdrawal of the Cretaceous sea, mountains surrounding the basins were uplifted repeatedly and were eroded continuously; subsidence occurred in the basins at the same time. Large volumes of river and lake deposits filled the intermountain basins throughout Paleocene and Eocene time. Sediments also were deposited on the subsiding margin of the Gulf coast sedimentary basin in Texas (Galloway et al, 1977). Minor periods of degradation occurred during this time also.

The early Tertiary also was a period of vigorous volcanism in the western United States. Thin units of volcanic ash containing disseminated uranium are interspersed with normal Eocene sedimentary units.

None of the depositional environments of the eight in situ mine sites described in this report were identical; characteristically, the boundaries between different adjacent environments are transitional. The key to subsurface designation, interpretation, and evaluation of sedimentary environments lies in the study of vertical sequences of lithology, and the study of sedimentary structures from cores, grab samples, and geophysical logs. The sedimentary environments of the eight sites are discussed in Appendix A.

2.1 SEDIMENTARY ENVIRONMENTS OF IN SITU MINE SITES

Most sandstone uranium deposits occur in hydrogeologic environments that consist of sediments that were deposited in fluvial or coastal environments. The stratigraphy of these sediments typically is very complex due to the nature of the depositional environments in which the sediments were deposited. These depositional environments include: 1) fluvial plains associated with meandering and/or braided streams, 2) alluvial fans and bajadas, and 3) beach barrier and tidal flats along coast lines.

Klein (1980) suggests that sandstone uranium deposits occur most commonly in fluvial channel systems; the most favorable zones for uranium accumulation appear to be the bottoms and margins of stream channels. According to Klein (1980) mineralization of uranium in the Texas Coastal

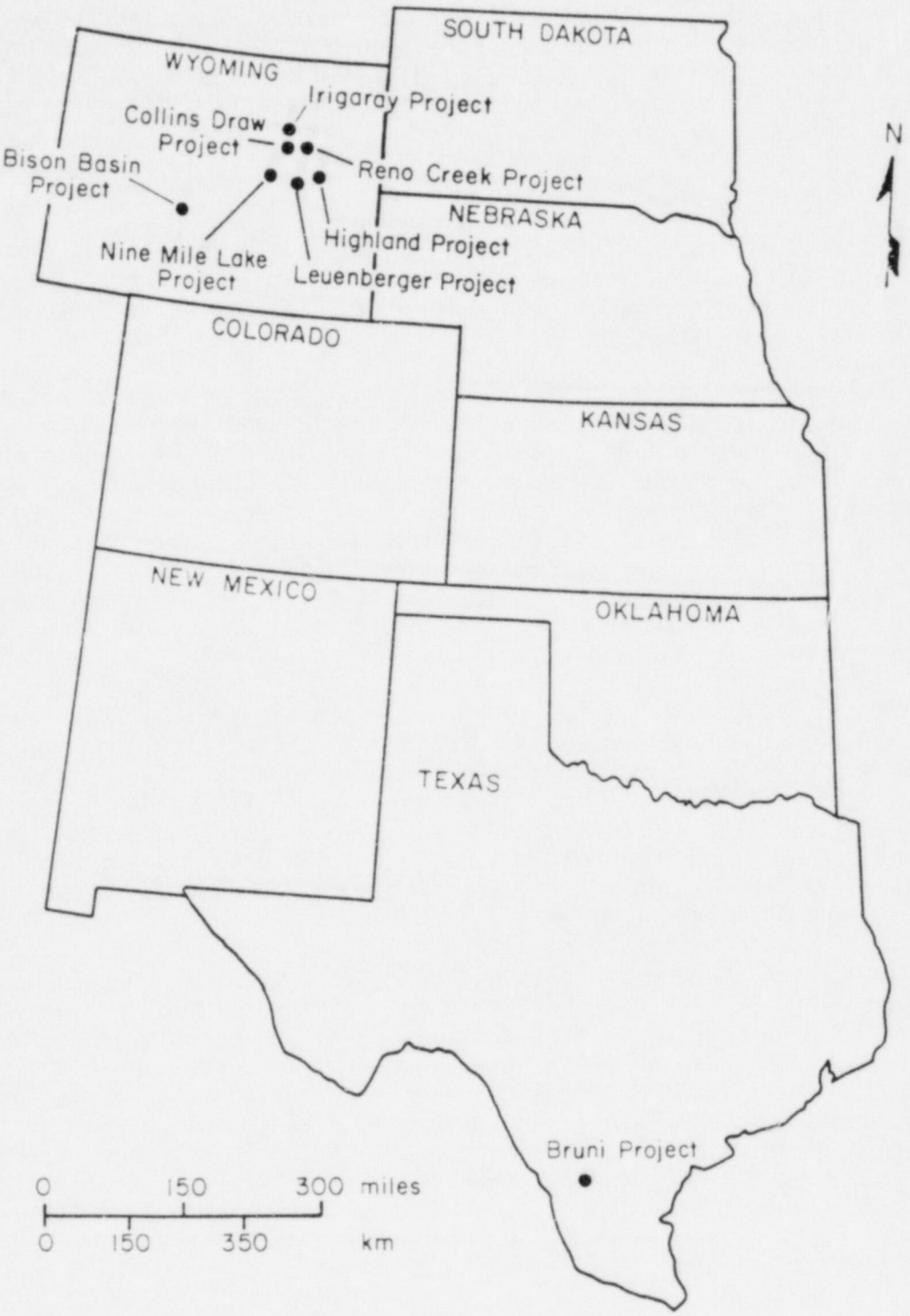


Fig. 2.4. Location map of the eight in situ uranium mine sites.

Plain is confined largely to the channels. In other areas, mineralization is favored along channel walls in braided alluvial fans, as has been demonstrated to occur in the Morrison Formation by Fisher (1970, 1974) and Rackley (1975), and in the Gas Hills district of Wyoming by Rackley (1972). Table 2.1 lists the depositional environments for some of the major uranium districts in the western United States.

2.1.1 Fluvial Plains

Fluvial plains consist of deposits of broad aggraditional floodplains that were formed by meandering streams and/or braided streams (Figs. 2.5 and 2.6); there also may be a type of fluvial plain that is intermediate between alluvial fans, and floodplains in humid areas (Blatt et al, 1980).

The geometry of sandstone bodies formed in meandering stream environments usually are elongate (shoestring), and bounded below and on both sides by flood-basin fines. The shoestrings also may be covered by fines due to migration of the stream to a new position during deposition (Walker and Cant, 1980). Models of high sinuosity meandering streams predict that a series of sand lenticles interbedded with shales should be developed if there is basin subsidence and a continual supply of sediment. The thickness of a single-sequence sand body should reflect the approximate depth of the depositing stream; however, stacking of sand bodies can occur with younger channel deposits cutting into older ones (Walker and Cant, 1980).

According to Klein (1980), braided streams show a bifurcating and rejoining channel pattern. They occur commonly in terrains of moderate slope where fluctuations in streamflow velocity result in the deposition of sediment as a series of longitudinal braid bars within the channel system. Fine-grained materials (silts and clays) tend to be transported through the system without accumulation; vertical accretion deposits are rarely preserved, and flood basin deposition is not as important a process as it is in meandering systems (Walker and Cant, 1980).

Sandstones formed by braided stream systems consist mostly of sand and gravel; muds usually are subordinate or absent (Friedman and Sanders, 1978). The absence of mudstones or shales in the stratigraphic record, distinguishes sandstone bodies that were formed by braided streams from those that were formed by meandering streams; also a progressive upward decrease in particle size is not nearly as prevalent in braided stream deposits as in meandering stream deposits (Friedman and Sanders, 1978). Walker and Cant (1980) suggest that coalescing bars and sandflats of braided streams typically result in laterally continuous and extensive sand sheets that are unconfined by shales; any shales preserved in the section will tend to be patchy and laterally discontinuous.

2.1.2 Alluvial Fans

Alluvial fans form when sediment-bearing water flowing out of uplands undergoes a sudden reduction in open channel flow velocity due to an abrupt reduction in slope angle. Stream morphology and sediment distribution

Table 2.1. Sedimentary characteristics of fluvial depositional environments in the uranium districts in the western United States.

District	Characteristics
Crooks Gap, Wyoming	Alluvial fans: unsorted, coarse-grained, poorly stratified, rare mudstones
Gas Hills, Wyoming and Ambrosia Lake, New Mexico	Braided river system: sandstones dominant; shallow, broad channels, few mudstones
Lisbon, Utah Laguna, New Mexico	Coarse-grained meander belt: mudstones more common, lenticular sandstones, deeper channels, point bars
Powder River Basin, Wyoming and Uravan, Colorado-Utah	Fine-grained meander belt: mudstones dominant, few lenticular sandstones, point bars
Bison Basin, Wyoming	Large system of channel sandstone deposits, siltstones and mudstones
Ray Point, Texas and George West, Texas	Bed load fluvial system: mixed load fluvial system, deltaic and barrier systems
Webb-Duval, Texas	Crevasse splay marginal to main fluvial channel complex

Source: DeVoto, 1978 (modified).

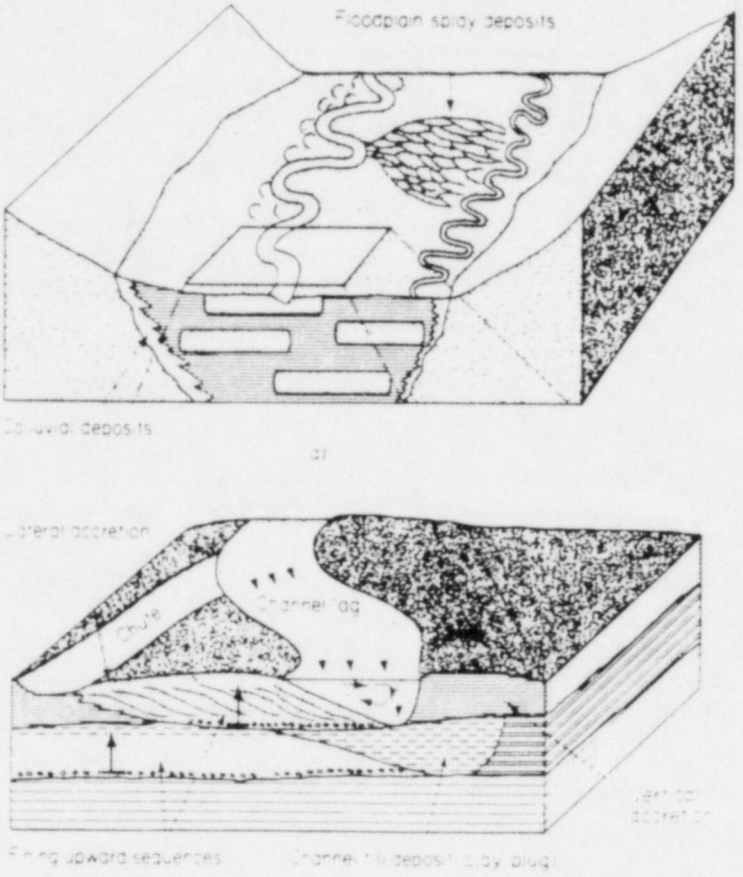


Fig. 2.5. Block diagrams showing main features of a meandering river and its floodplain. *Source:* Blatt et al., 1980.

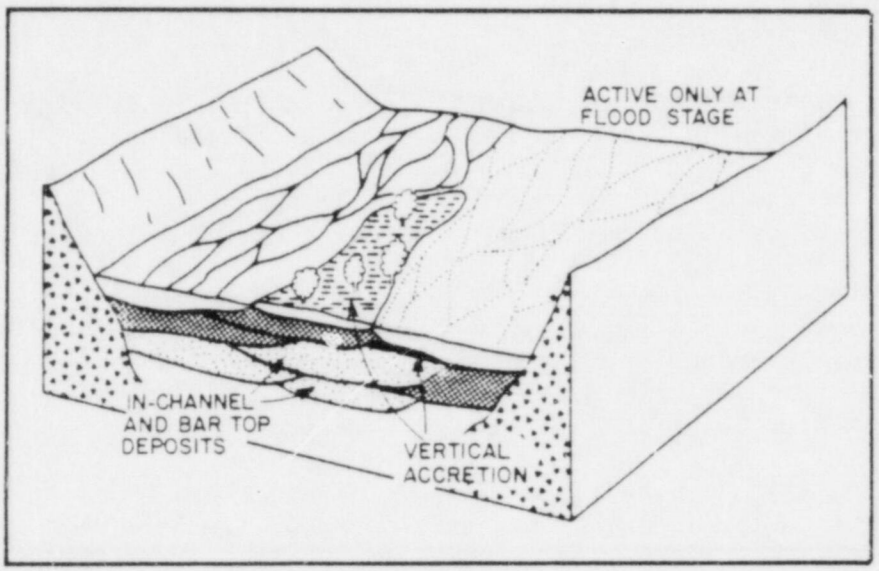


Fig. 2.6. Block diagram of a braided sandy system with low sinuosity channels. *Source:* Modified from Allen, 1965.

(Fig. 2.7) are controlled by the slope changes, depositional processes and flow velocity (Klein, 1980).

Sediment deposited on alluvial fans typically is coarse; poorly sorted debris flow deposits or poorly stratified fluvial gravels typically occur near the apex (Blatt et al, 1980). Normally, the maximum clast size is found near the apex, even in fans deposited mainly from debris flows. Fans deposited by stream floods and flows generally contain sandy interbeds with plane stratification and planar and trough cross-stratification (formed by transverse bars, sand waves, and dunes) (Blatt et al, 1980).

According to Friedman and Sanders (1978) the following diagnostic features should aid in the recognition of alluvial fan deposits in the rock record:

1. Fan deposits generally are lenticular or wedge shaped.

2. Water-laid deposits consist typically of moderate to well sorted, coarse-grained sandstones and conglomerates. Debris-flow deposits consist generally of sand particles, cobbles and boulders embedded in a poorly sorted shaley and silty matrix. Shale is dominant in mud-flow deposits and large particles are minor.

3. Stratification ranges from good in water-laid deposits to poor in debris-flow deposits to nearly nonrecognizable in mud-flow deposits. Water-laid deposits may be cross stratified or plane bedded.

4. Water-laid deposits may occur in channels as channel fills having truncations at their base, or as extensive sheetlike beds.

5. Organic material is usually absent.

6. The colors of the rocks are yellow, brown or red, indicating that the iron is in an oxidized state.

7. Typically fan deposits interfinger with other products of desert environments, such as those of playas; in some cases, however, fans build at the edge of the sea.

2.1.3 Beach Barrier and Tidal Flats

Sandstone bodies associated with three main environments of barrier-island depositional systems are of significance with respect to potential uranium deposits. These environments are the barrier beach, lagoon, and tidal channel-delta complex. Reinson, (1980) describes these environments as follows. Sediments deposited in barrier-beach and channel-delta environments consist predominantly of sand and gravel, whereas the lagoonal (back-barrier) deposits can consist of both mud and sand. Elongate barrier-beach deposits lie parallel to the strandline and enclose finer-grained deposits of the lagoon. Tidal-channel and delta sand deposits, generally are oriented perpendicular or oblique to the barrier complex;

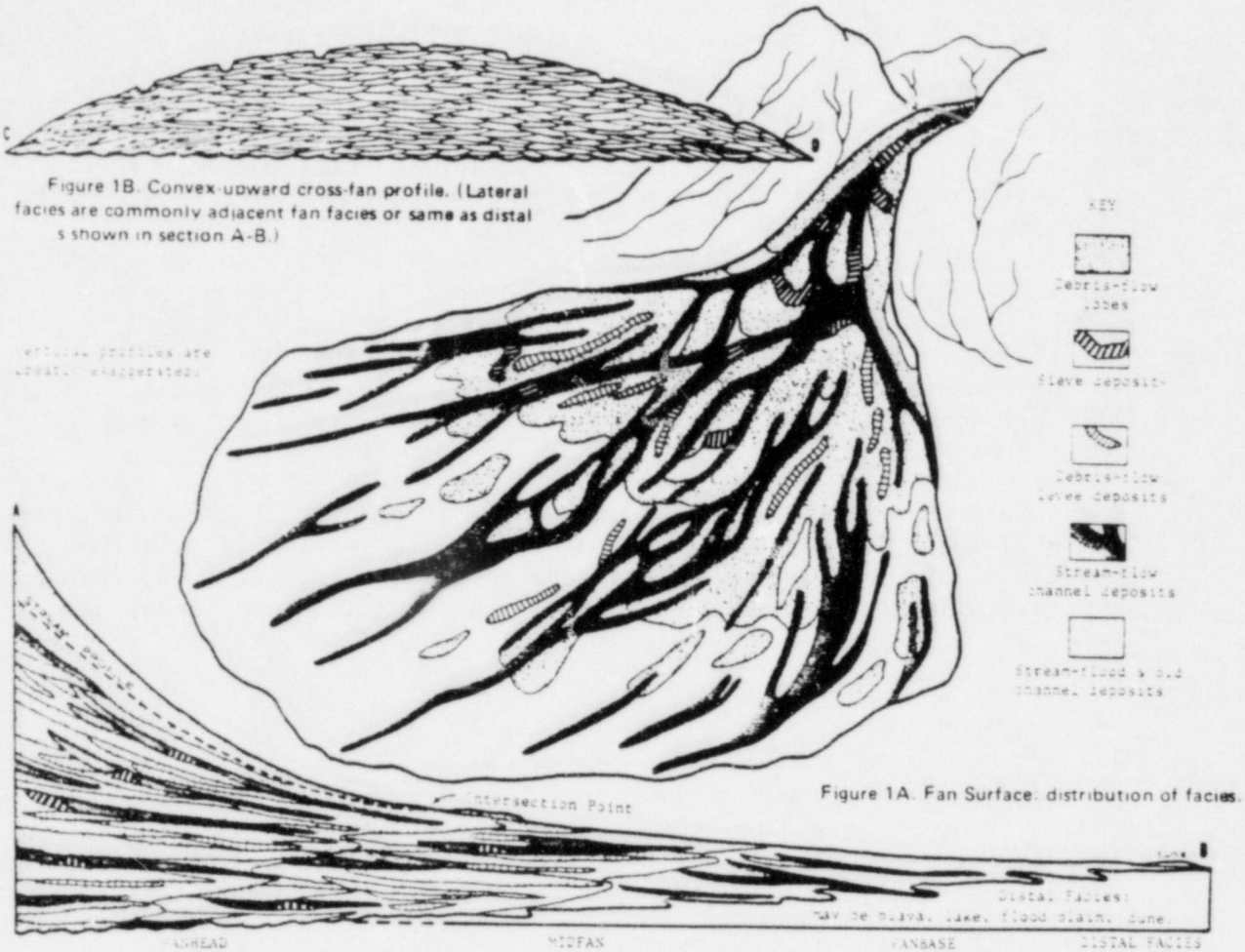


Figure 1C. Concave-upward radial profile.

Fig. 2.7. Distribution of sediment facies and morphological profiles of an ideal alluvial fan. *Source:* Spearing, 1975.

these deposits can extend into the lagoon and seaward into the nearshore zone. Transition between deposits of the lagoon and barrier and channel-delta deposits occurs in the overlapping subenvironments of the back-barrier tidal flats, marsh, washover fans and flood-tidal deltas.

According to Klein (1980), uranium occurrences in sandstones usually are limited to sandstones that are interbedded with mudstones; precipitation of uranium from solution is believed to have been accelerated by impedence to ground-water flow at sandstone-mudstone intertongues, or by interbedded organic rich layers. Good examples of the barrier sandstone-type uranium deposits occur in the Texas Coastal Plain in the Jackson Group (Dickinson, 1976; Dickinson and Sullivan, 1976; Fisher et al, 1970). Uranium most commonly occurs in sandstones where they are in contact with mudstones and organic rich beds; uranium concentrations typically are highest on the updip side of barrier sands (Fisher et al, 1970), presumably because the updip sides of such systems tongue into lagoonal muds and marshes.

Interbedded sandstones and mudstones are common to tidal flats, particularly where intertidal channels are filled with sand (Klein, 1980). Supratidal environments potentially are conducive to uranium precipitation, if a source is present; organic matter in salt marshes may serve potentially as a precipitant for uranium in solution (Klein, 1980).

2.2 HYDROGEOLOGIC TESTING OF IN SITU URANIUM MINE SITES

Delineation of stratigraphic units and evaluation of hydrogeologic properties at in situ uranium mine sites are essential aspects of site characterization. Variations in subsurface hydraulic conductivity can significantly affect the movement of lixiviant by providing preferential pathways for the movement of lixiviant (excursions) beyond the production area of the in situ mine. Failure to collect comprehensive data during exploration and testing programs will result in an inadequate understanding of the hydrogeology and an inadequate monitoring system (Williams and Osiensky, 1983, Osiensky et al, 1984).

Questions that must be asked about proposed in situ mine sites include:

1. Can the hydrogeology be defined adequately to ascertain whether or not an excursion will develop?

2. Where will the lixiviant migrate?

3. What measures may be taken to prevent excursions?

4. Can excursions be controlled?

Upon completion of exploration and testing programs, the mining company and regulatory agency must decide what risks are acceptable.

Standard methods for delineating hydrostratigraphy include analysis of borehole logs such as drill cuttings and core samples. Hydraulic properties are estimated by a variety of field and/or laboratory methods. Laboratory methods and some field methods provide point values of the basic hydrogeologic characteristics; laboratory tests are performed on small samples that may not be representative of the entire area of interest or may have been disturbed during handling. Field methods of aquifer testing are more acceptable because they provide estimates of hydrogeologic parameters that are measured in situ. These estimates are representative of a volume of porous media surrounding pumping and injection wells. Single-well tests provide values representative of a small volume of porous media in the immediate vicinity of the well. Multiple-well field tests provide measurements that are averaged over a larger aquifer volume.

2.2.1 Borehole Logs

Exploration boreholes usually are drilled throughout the mine areas to evaluate the extent and quality of the ore zones. Some of these wells are logged throughout while many are logged only in specific ore zones. Once a uranium ore zone has been detected, boreholes typically are drilled at spacings from 6 to 15 m (20 to 50 ft). This grid of boreholes extends across the plan area of the ore zone. Natural gamma geophysical logs generally are recorded in closely spaced exploration boreholes primarily to locate the uranium-bearing zone(s). Additional geophysical logs (such as single point resistance, spontaneous potential, neutron epithermal neutron, sonic and caliper logs) are rarely recorded.

Data from boreholes drilled at in situ uranium mine sites are typically not obtained or fully utilized for the purpose of delineating stratigraphic units and sedimentary facies. Continuous cores or extracted samples and geophysical logs throughout the length of each borehole would enhance the ability to interpret stratigraphy and its associated ground-water regime. It is difficult to predict the nature of the ground-water regime without the ability to map sedimentary facies in the subsurface. However, collection and interpretation of these data are arduous tasks and the results of such efforts are not always satisfactory because of the stratigraphic and structural complexities involved. Even subtle changes in lithology can produce large changes in permeability as exemplified by the concretionary and silty sandstone facies of the Wasatch Formation.

Geophysical borehole logging requires considerable care. It is best to record all geophysical logs in open boreholes. Electric logs cannot be recorded properly in cased holes. However, uncased holes have a tendency to collapse below the water table. Service companies are reluctant to log uncased holes in unconsolidated sediments for fear of inability to recover instruments from a collapsed hole. Radiation logs can be recorded in uncased or cased boreholes. They generally require a maximum logging speed of 4.6 m (15 ft) per minute in order to give an adequate degree of detail. A constant hole diameter should be maintained to keep the radius of investigation constant and to eliminate the difficulties of interpreting logs from multidiameter holes. Caliper logs are necessary in all open holes when radiation logs are to be interpreted because the response of radiation

logs is sensitive to borehole diameter. The reader is referred to Keys and MacCary (1971), Pirson (1970), and Visher (1965) for detailed discussions of geophysical logs and their interpretation.

2.2.2 Aquifer Testing

Aquifer tests are used to delineate and characterize the stratigraphic units at in situ uranium mine sites. Carefully designed, conducted, and analyzed aquifer tests can be used to evaluate many hydrogeologic characteristics of importance. The characteristics include:

1. degree of hydraulic separation between the ore zone and adjacent overlying and underlying aquifers,

2. hydraulic continuity in the ore-bearing unit,

3. identification of boundaries and concomitant variations in hydraulic conductivity,

4. hydraulic conductivity of the ore zone and vertical hydraulic conductivity of the aquitards overlying and underlying the ore zone, and

5. velocity of ground-water flow through individual stratigraphic horizons.

Multiple-well and single-well pumping/injection tests are the most common aquifer testing methods used at in situ uranium mine sites. Values of hydraulic conductivity, transmissivity, and storativity should be estimated for the ore zone aquifer from data for each well completed in the aquifer. Vertical hydraulic conductivity of the confining bed(s) should be estimated by an appropriate methodology. Aquifer test data also should be analyzed for boundary and leakage conditions.

Water levels often are measured in observation wells completed in aquifers overlying and underlying the pumped aquifer to evaluate possible leaky conditions. Such observations provide the most direct evidence of hydraulic communication between aquifers. In the most serious cases (broken casings, improperly plugged and abandoned holes, a discontinuous aquitard) of potential excursion pathways, observation wells may respond within a few hours of the initiation of pumping. For sluggish pathways of excursion, the response time may be excessive. In sluggish ground-water regimes, drawdown in observation wells may also be masked by other factors, such as antecedent water level trends, barometric pressure changes, interference pumping, and consolidation (the Noordbergum effect) (Verruijt, 1969). Antecedent water level changes and barometric pressure changes have not been measured during most aquifer tests conducted at in situ uranium mine sites; therefore, it is occasionally difficult to interpret the data collected from wells completed in the overlying and underlying aquifers. Where the results of the above observations are inconclusive, other methods of analysis to detect leakage and differentiate between causes of leakage will be required.

The Theis (1935) nonequilibrium method and the Cooper and Jacob (1946) straight line method are used extensively to estimate the transmissivity and storativity of the pumped aquifer from an analysis of pumping test data. These methods of analysis give valid results when a sufficient amount of drawdown data is available and when the following assumptions are approximated closely:

1. The aquifer is homogeneous and isotropic.

2. The aquifer has infinite areal extent (its boundaries are well beyond the effects of pumpage).

3. The pumping well fully penetrates the aquifer.

4. The well has a small diameter (of no practical significance for periods of pumping more than a few minutes).

5. Water is released from storage in the aquifer instantaneously with decline in head.

Modifications to these methods exist and must be applied if geologic and hydrologic conditions do not approximate the assumptions. Such modifications must be applied for changing water table conditions, leakage, partial aquifer penetration, boundary conditions, variable pumping rates, barometric pressure changes, tidal effects, or multiple confined aquifers.

One of the most common methods of analysis used for leaky aquifer conditions is the Hantush and Jacob (1955) leaky artesian method. This method allows the estimation of values of hydraulic conductivity and storativity of the pumped aquifer and the vertical hydraulic conductivity of the overlying confining bed. The calculated values of vertical hydraulic conductivity constitute highest estimates when water is leaking through both confining beds. The Hantush and Jacob (1955) method has limited applicability because it does not account for head changes in overlying and/or underlying aquifers, or water released from storage in the confining layers. The method of Hantush and Jacob (1955) is best suited for long pumping periods and where the leaky confining bed is noncompressible or thin so that water released from storage in the confining bed is significant.

The Hantush (1960) modified leaky artesian method, which allows estimation of the transmissivity and storativity of the pumped aquifer, takes into account water released from storage in the confining beds. Vertical hydraulic conductivity of a single confining bed can be estimated if the specific storage is known, and if the vertical hydraulic conductivity and specific storage of the other confining bed are known. The Hantush (1960) method is best suited for situations where the leaky confining units are thick and compressible so that water derived from storage in the confining beds is predominant.

Values of vertical hydraulic conductivity estimated in Appendix A using the Hantush (1960) method constitute highest (most conservative)

estimates for a single (overlying) confining bed because the vertical hydraulic conductivity (K") of the underlying aquitard is assumed (for the purpose of the calculations only) to be zero. Lowest (least conservative) estimates of vertical hydraulic conductivity occur when the vertical hydraulic conductivities of the two confining beds are equal. The resulting lowest estimates are less than an order of magnitude smaller than the highest estimates. It is assumed also that the specific storage of the upper and lower confining beds (S_s' and S_s'', respectively) is two orders of magnitude larger than that calculated for the corresponding aquifer; these values of specific storage correspond reasonably well with representative values calculated using the values of compressibility for clay cited in Freeze and Cherry (1979). The values of vertical hydraulic conductivity estimated using the Hantush (1960) method must be considered as approximations because of the assumptions described above. Leaky aquifers will receive water from storage in the confining bed(s) during early periods of pumping whereas water derived from overlying and/or underlying aquifers will reach the pumped aquifer by movement through the confining layers during later periods of time (assuming there is no "short circuit" such as a broken well casing, improperly abandoned borehole, or discontinuous confining layer in the vicinity). These time periods are a function of aquitard thickness and vertical hydraulic conductivity.

The "ratio method" developed by Neuman and Witherspoon (1972) is the most sophisticated pumping test method currently available for estimating the aquifer characteristics of multiple aquifer-aquitard systems. The vertical hydraulic conductivity of the individual confining layers can be estimated as well as the hydraulic properties of the aquifer being pumped. Because the ratio method does not provide an estimate of specific storage for the confining layers, these values must be measured independently. The method relies on early drawdown data, so the pumping test can be of relatively short duration. Prior knowledge of the aquitard thickness is not required. In addition, the ratio method is simple to use, and does not involve graphical curve-matching procedures.

Injection tests of zones isolated by hydraulic packers often are used to estimate the saturated horizontal hydraulic conductivity of selected intervals of boreholes. Saturated horizontal hydraulic conductivity using a packer assembly can be calculated from the following relationship (DOI, 1974):

$$K = \frac{Q}{2\pi LH} \ln \frac{L}{r}$$

where:

K = hydraulic conductivity,

Q = constant rate of flow into the hole,

L = length of the portion of the hole tested,

H = differential head of water, and

r = radius of the hole tested.

This equation is valid for L>10r. Relative rather than absolute values of hydraulic conductivity are best described when conducting injection tests using packers. Causes for common errors include (Cedergren, 1977):

1. leakage along casing and around packers,

2. clogging due to sloughing of sediment in the test water,

3. air locking due to gas bubbles in soil or water, and

4. flow of water into cracks that are opened by excessive head in test holes in soft rocks.

The hydrogeologic characterization of existing or potential in situ mine sites is a difficult task given the sedimentary environments at these locations. The NRC (1982) lists the following factors of primary importance in the measurement of hydraulic conductivity:

1. selection of a method compatible with the hydrogeologic conditions,

2. recognition of the limitations and assumptions of the method used,

3. testing only the proper hydrostratigraphic unit as defined before the test, and

4. have the test conducted and analyzed by someone thoroughly familiar with the test, the hydrogeology, and the applicability of the test to the hydrogeologic environment in question.

2.3 SUMMARY OF LIMITATIONS IN CHARACTERIZING GROUND-WATER REGIMES

Conventional exploration methods as described above seldom provide enough detail to determine whether an ore zone aquifer is sufficiently isolated from other aquifers. The complex stratigraphy of alluvial sediments is a serious obstacle to projection of lithologic units between boreholes even at short distances. Figure 2.8 presents several interpretations of the same data. In the absence of proof of the contrary, stratigraphic units are often projected as continuous layers between boreholes (refer to case histories in Appendix A) which may lead to a false sense of security with respect to aquifer isolation. Furthermore, two-dimensional cross-sections do not necessarily portray accurate relationships between aquifers in three dimensions. Thus, an aquitard may be continuous in one direction and discontinuous in another.

ORNL-DWG 85-17985

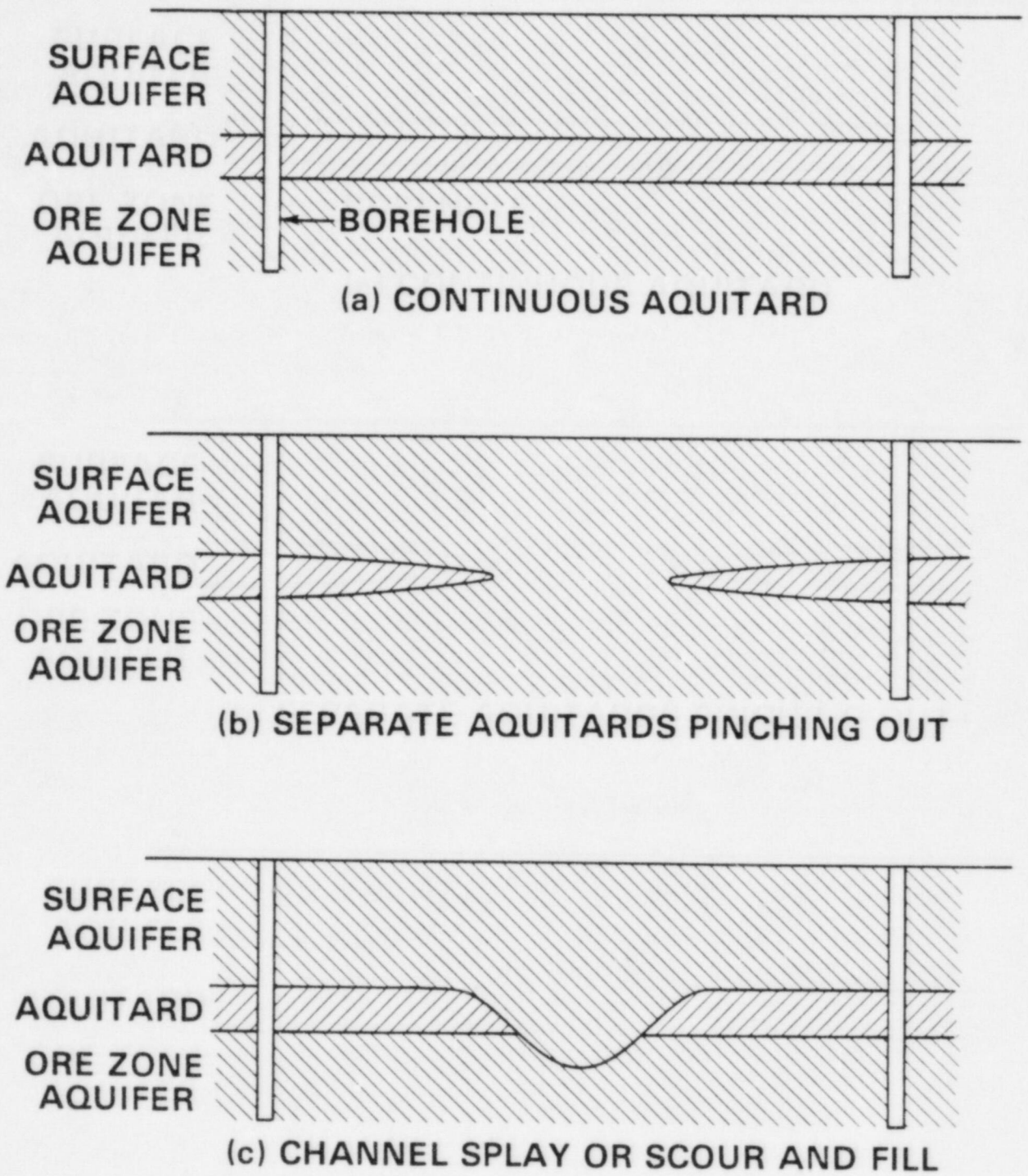


Fig. 2.8. Alternate interpretations of the same borehole data.

It is recognized that there are limits to the amount of data that can be collected and analyzed. Furthermore, subtle changes in lithology may remain undetected despite rigorous data collection and analysis. Subtle changes in lithology (slight variations in silt, clay or carbonate cementing agents) have the potential to profoundly influence hydraulic conductivity.

Potential in situ mine sites also have an uncertain prior drilling history. Older exploration holes in search of fossil fuels and uranium are difficult, if not impossible, to locate and many of them were improperly plugged and abandoned.

To counter the above uncertainties, aquifer tests are used to provide proof of aquifer isolation. To date, aquifer tests have not provided consistently definitive results. There are several reasons for this:

1. Ore zone aquifers yield only a few gallons of water per minute compared to the larger and more robust systems for which aquifer tests were originally designed.

2. The duration of aquifer tests is often too short to determine the aquifer properties that they were designed to measure.

3. Proper choice of testing methodologies and analyses depends on the level of understanding of the ground-water regime.

It is important to keep the above limitations in perspective in examining the case histories of Appendix A. These case histories were compiled from NRC docket files. A licensee's interpretation of stratigraphic information in these files is often oversimplified and fails to address the uncertainties involved. The reader is advised to consider the uncertainties. For example, the continuity of a thin aquitard in three dimensions is highly uncertain relative to that of a thick aquitard. On the other hand, even a thick aquitard may be truncated by an unmapped fault or by channel scour and fill. Finally, failure to identify the presence of poorly plugged or abandoned holes does not imply their absence.

3. EXCURSIONS

An excursion is an uncontrolled migration of lixiviant and its associated dissolution products and by-products away from controlled areas (production well fields) of in situ mining ore zones. Early detection of an excursion is a necessary prelude to the implementation of timely corrective action to return escaping fluid to the well field. Without timely corrective action, a large quantity of valuable production fluid may be irretrievably lost and serious contamination of adjacent ground-water resources may result.

Analysis of excursions at the 8 in situ mines in this study is complicated by the diverse methodologies used to identify excursions. Methodologies differed mainly in the selection of excursion indicators, the determination of their upper control limits (UCLs), and in the definition of a legal excursion.

Some reported excursions were probably false alarms. There are a variety of reasons for these false alarms. Most false alarms were caused by setting unrealistically low UCLs for trace elements or poor selection of indicators. Others were caused by placing monitor wells too close to the ore zone perimeter. At most mine sites close-in observation wells would be considered as trend wells rather than monitor wells.

Despite inconsistencies in identifying excursions it is evident that many excursions did occur. Most horizontal excursions were brought under control quickly. However, wells used to monitor for vertical excursions were on excursion status repeatedly and for excessively long periods of time. In many cases restoration procedures were eventually required. It is particularly important to recognize vertical excursions at an early stage in order to avoid costly and time consuming restoration.

Sections 3.1 through 3.3 summarize the causes, consequences, and control methodologies for horizontal and vertical excursions. Sections 3.4 and 3.5 summarize methods for recognizing the potential for excursions at in situ mining prospects and methods for recognizing excursions when they occur, respectively.

3.1 HORIZONTAL EXCURSIONS

The only serious horizontal excursions at sites in this study occurred during the early history of in situ uranium mining. At the Highland site several intense and long-term excursions took place during experimental pilot tests in the early- to mid-1970's (predating the Nuclear Regulatory Commission). These excursions were attributed to numerous breakdowns in pumping equipment and failure to adjust injection-production rates. Between March, 1972 and October, 1974 the total volume of fluid injected exceeded the amount produced at Highland, a practice which is now recognized by the industry as a fundamental cause of horizontal excursions. At Bruni, Texas 15 horizontal excursions were reported between Sept., 1976 and

Dec., 1977 in a commercial operation containing over 1700 production and injection wells. However, during the next four years of mining no more horizontal excursions were reported. The same company (Wyoming Mineral Company) performed commercial scale mining (over 250 production and injection wells) at Irigaray, Wyoming from Nov., 1978 to Dec., 1981 with only one brief (3 months) and unequivocal horizontal excursion. Evidently, WMC has developed an effective methodology for controlling the horizontal movement of lixiviant within the ore zone and its immediate surroundings.

There is direct evidence that poor control of injection and production rates is the fundamental cause of horizontal excursions. Early excursions at Highland occurred when malfunctioning production wells were shutdown for repairs without taking any compensating measures. At Bison Basin, excursions occurred in several monitor wells adjacent to a row of injection wells without any production wells between them. At Nine Mile Lake, lixiviant was simultaneously injected into two ore zones without control over the rates of injection into each zone. All of the horizontal excursions at Nine Mile Lake occurred in the upper ore zone, suggesting that the majority of lixiviant was discharged through the upper, rather than the lower, screened interval of each injection well. A number of horizontal excursions occurred at Collins Draw where no storage space was available for excess production fluid. Therefore, injection rates were (out of necessity) equal to or greater than production rates.

The natural ground-water gradient and external influences on the gradient contribute to the incidence of horizontal excursions. Well-field design should take these factors into consideration. This was not done at Bison Basin where the natural gradient was parallel to single rows of production and injection wells without any production wells on the downgradient side of the field. A monitor well on the southeast (downgradient) side of the well-field remained on excursion status for over a year beyond the end of the pilot test, presumably because the escaping lixiviant could not be drawn back into the field. Underground conventional mine dewatering contributed to horizontal excursions during expanded pilot tests at Highland. Nearly all of these excursions took place on the downgradient side of the well-field and toward the underground mine. However, production wells were located downgradient from the nearest injection well and these excursions were all brought under control within 2 to 5 months by adjusting production and injection rates.

Excursion control remedial actions provide indirect evidence that poor control of injection and production rates was responsible for initiating an excursion. Most excursions were brought under control within a few weeks or months simply by adjusting injection rates downward and production rates upward in wells adjacent to the effected areas. The implication is that excursions can be turned on and off, simply by manipulating local or field-wide injection and production rates. The most persistent horizontal excursions occurred at Collins Draw because excess production fluid could not be stored. Hence, unbalanced injection and production rates could not be corrected in one part of the field at Collins Draw without throwing them out-of-balance elsewhere.

A few of the relatively persistent horizontal excursions required more intense remedial action measures for their control. All of these measures, however, are variations involving the adjustment of injection and production rates. In some cases natural ground water was injected without refortifying it with lixiviant, while increasing the production rate. In other cases injection wells were shutdown completely while production continued. At Nine Mile Lake (where lixiviant had been simulatenously injected into upper and lower ore zones) the upper zone was sealed off while production continued from both ore zones. All of the above methodologies were effective in bringing horizontal excursions under control and all, in one way or another, involved the adjustment of injection and production rates.

In summary, the industry has demonstrated both the ability to reduce the frequency of horizontal excursions and to control them when they do occur. Excursions are reduced by controlling the field-wide production rate at a few percent greater than the injection rate (known as "bleeding" the ore zone aquifer) and storing the excess fluid in a secure surface impoundment for eventual evaporation. The size of the bleed will necessarily be limited by the size of the impoundment and evaporation rate. In no case should in situ mining of uranium take place without storage space for excess production fluid. In any event, infrequent horizontal excursions can be expected because operating systems are occasionally upset, the ore zone aquifer is nonhomogeneous, and the ground-water system is subject to transient external influences. Thus, local imbalances in injection and production rates will occur despite the implementation of a field-wide bleed. The frequency and intensity of localized horizontal excursions are inverse functions of the bleed rate. Nevertheless, horizontal excursions can be controlled by either local or field-wide manipulations of injection and production rates ranging from slightly reduced injection to complete injection well shut-down while continuing to produce from the ore zone aquifer. The options available will depend on the size of the impoundment for storage of excess production fluid.

3.2 VERTICAL EXCURSIONS

Serious vertical excursions persisted through the end of 1981 when commercial scale in situ mining was suspended in response to the declining uranium market. At Bruni, 12 shallow aquifer monitor wells were on excursion status one or more times during the same time interval (1978-81) that no horizontal excursions were reported. The duration and current status of these excursions is unknown. About half of the wells on excursion status at Bruni may have received contaminated water from a leaking evaporation pond, but other monitor wells appear to be beyond the influence of the evaporation pond. At Irigaray 8 shallow monitor wells were on excursion status one or more times for durations ranging from 2 months to 3.5 years. Although the causes of most vertical excursions are well known, attempts to control them have not been very successful.

The preponderance of evidence suggests that most vertical excursions are caused in the upper aquifer by improperly plugged and abandoned exploration holes and by poorly completed field wells. Among in situ mining

sites where both upper and lower aquifer monitor wells were present, all but one of the reported excursions took place in the upper aquifer. Thirteen of 30 monitor wells completed in the upper aquifer were reported on excursion on one or more occasions (some for extended periods of time) whereas only 1 of 14 wells completed in the lower aquifer was placed on excursion status (twice, for 1 and 6 weeks). It is likely, however, that this lone well experienced false alarms. It was placed on excursion status when a single trace element (uranium) exceeded its unrealistically low UCL (12 ppb) by only 5 ppb. No common anions were ever reported above their UCLs for any lower aquifer monitor well at the 8 in situ mine sites in this study. The good statistical correlation between the number of holes that penetrate the upper and lower aquifers and the number of excursions suggests the obvious: that somehow vertical excursions are directly related to the intensity of drilling activity. Exploration holes and wells that penetrate the upper aquifer outnumber those that penetrate the lower aquifer by perhaps two orders of magnitude. Furthermore, wells that penetrate the lower aquifer are seldom (if ever) used to transport lixiviant.

When a vertical excursion occurs standard practice is to locate abandoned open boreholes and pressure test completed wells in a search for defective or broken casings. Improperly plugged exploration holes are sealed and damaged wells are either recompleted or abandoned and sealed. The effectiveness of these procedures in controlling vertical excursions depends on the ability to locate all the abandoned holes. Older holes are most likely to be unplugged. However, they are difficult to locate because of the lack of records, the scattering and covering of well cuttings by erosion and vegetation, respectively, and the collapse of exposed surface casing, if indeed permanent casings were ever installed.

Approximately 15 monitor wells were placed in the upper aquifer at Irigaray during pilot tests and commercial scale mining. Eight of these wells were placed on excursion status for periods ranging from one month to more than 3 years. Three excursions were confirmed casing failure incidents. A total of 62 field wells were repaired or plugged and abandoned to shutoff the flow of lixiviant to the upper aquifer. Field well repairs in combination with increased production from nearby ore zone wells and direct pumping from monitor wells brought two of these excursions temporarily under control within 5 months. Although these corrective actions were beneficial, they were not completely effective. One well remained on excursion status continuously for over 3 years and the other 2 wells experienced later excursions that lasted 2 and 5 months. Three other excursions were evidently caused by open exploration holes. As in the casing failure excursions previously described, open bore holes were sealed, monitor wells were pumped, and production rates were increased in the vicinity of the excursions. Again the corrective actions were partially effective.

Possible causes of renewed excursions at the above sites are speculative. It is reasonable to assume that some open exploratory holes were never found. On the other hand, the confining bed overlying the ore zone aquifer is relatively thin (1.5 to 9 m) at Irigaray and may even be absent

in some places. Most upper aquifer monitor wells were screened in a thin coal unit to provide for an early warning of an excursion. Thus, the possibility exists that vertical excursions are taking place through a leaky confining layer as well as through undiscovered open boreholes.

Nevertheless a leaky aquifer does not appear to be the main cause of vertical excursions at Irigaray. Otherwise, adjusting injection and production rates should have been more effective in controlling these excursions. Furthermore, there should have been some leakage through the lower confining layer as well. In contrast to the upper aquifer where a number of prolonged and difficult to control excursions were reported, none of the 6 monitor wells in the lower aquifer was ever placed on excursion status. The lower confining bed is about 4.5 to 18 m thick, about twice as thick as the upper confining bed. Therefore, the lower bed should be only fractionally less leaky than the upper bed and at least a few excursions of brief duration might have been expected.

At Bruni, 45 of approximately 120 upper aquifer monitor wells were placed on excursion status; many for extended periods of time. As at Irigaray, broken casings and open boreholes are believed to be the main pathways of excursion. At Bruni, however, a number of vertical excursions were also caused by the rupture of an evaporation pond liner. Leakage is unlikely through confining strata because they are thick (15 to 18 m), dense claystones.

Corrective actions at Bruni were apparently ineffective. The licensee attempted to halt the spread of contaminants in the shallow aquifer by injecting fresh water around the perimeter of the affected areas while withdrawing contaminated water from the interior. A number of monitor wells remained on excursion status for several years.

Three other mine sites reported a total of 5 minor vertical excursions. Only one of these excursions was of any consequence. It was caused by a ruptured casing in an injection well. A new well was drilled within 3 meters of the damaged well to clean-up and monitor the affected aquifer. The clean-up well remained on excursion status for approximately 4 months.

3.3 SUMMARY OF EXCURSIONS, THEIR CONSEQUENCES, CAUSES, AND CONTROL

Horizontal excursions are not of serious concern in in situ uranium mining. First, the manipulation of injection and production rates is adequate for controlling horizontal excursions. Second, eventual restoration of an ore zone aquifer is an extension of horizontal excursion control. During aquifer restoration the bleed rate would be increased and natural or treated (rather than lixiviant rich) ground water would be injected. Natural ground water would also be swept through the ring of monitor wells and toward producing field wells. Thus aquifer restoration methodology is similar to, but more intensive than, horizontal excursion control. Any areas remaining on excursion status at the end of mining would automatically be brought under control during the restoration process.

On the other hand, the in situ mining industry has yet to demonstrate an effective method for controlling vertical excursions. Vertical excursions in this study generally lasted from several months to several years. A number of excursions that appeared to have been brought under control eventually recurred. Although notable improvements have been made in pressure testing field wells after their completion, the ability to locate and seal all open exploration holes is still questionable.

There is serious concern for vertical excursions for several reasons. First, shallow aquifers are likely to be suitable for livestock and domestic use. Second, existing technology does not appear to be adequate for vertical excursion control. Finally, eventual ore zone aquifer restoration has no effect upon an overlying aquifer after excursion pathways have been sealed-off. Therefore, excursion control of an overlying aquifer may require the installation of a separate set of field wells and a level of effort similar to that used for the restoration of the ore zone aquifer, itself.

The following sections discuss: 1) procedures for identifying excursion prone areas before mining begins, and 2) methodologies for early detection of excursions in an active mining area. The prevention and control of vertical excursions is emphasized.

3.4 AQUIFER TESTS

Aquifer testing is an accepted methodology for developing a well-field design at an in situ mine. Hydraulic properties of the ore zone aquifer are determined from these tests and they are used to determine distances between injection and production wells and their design flow rates.

Ore zone aquifers having suitable hydraulic properties for in situ mining also have the potential for horizontal excursions. Areas having low hydraulic conductivities and transmissivities would be unlikely candidates for either horizontal excursions or in situ mining. On the other hand, some areas might have such high natural ground-water gradients and transmissivities that the risk of horizontal excursions is unacceptably high. However, none of the 8 mine sites in this study experienced horizontal excursions that could not have been controlled through good management practices such as those described in previous sections.

Present aquifer testing procedures are adequate for measuring hydraulic properties in areas where the risk of horizontal excursions might be unacceptable. However, neither the upper limit for transmissivity (this limit being functionaly related to the ground-water gradient) nor the unacceptable risk has ever been defined. These limits will only become known when state-of-the-art technology fails to control a horizontal excursion.

In contrast to their use in appraising the potential for horizontal excursions, present aquifer testing procedures have had more limited success in determining the potential for vertical excursions. The limited

duration of these tests and the limited number of wells involved reduces the possibility of measuring observable responses in upper or lower aquifer monitor wells. Furthermore, such tests reflect only localized rather than field-wide conditions.

Short duration aquifer tests have limited value in appraising the potential for vertical excursions. Therefore, the decision to proceed with field development should be based on the probability of encountering open exploration holes and thin confining strata as well as aquifer tests. These would be judgment decisions based on exploration history, examination of available drilling logs, geophysical logs, and other geological data.

A pilot test or commercial mining unit should undergo pressure tests before lixiviant is introduced to an ore zone. Pressure testing of individual injection and production wells to certify casing integrity is already standard practice. The mining unit would be pressurized by activating all the injection wells in the unit while all the production wells are in a shutdown mode. Mine unit (e.g., all the field wells inside a 5-spot monitor well pattern) pressurization would be used to determine whether there are any open exploration holes within the unit. Perhaps dye tracers could also be injected into the ore zone aquifer while the mining unit is being pressurized. Ore zone, upper and lower aquifer monitor wells could be observed for the appearance of dye and increase in water level. If upper and lower aquifers do not respond within a week, monitor wells could be pumped during a second week in an attempt to recover the dye tracer. Alternatively, dye could be introduced into upper or lower aquifer monitor wells and production wells could be pumped vigorously after pressurizing by injection is halted. Water levels would be observed in monitor wells while the production fluid would be observed for the appearance of dye. Evidence of hydraulic communication with an upper and lower aquifer would result in denial of certification. Certification tests would be repeated until all vertical excursion pathways are sealed or the field unit is abandoned. Ore zone monitor wells would be observed to determine whether they are close enough to measure a response to the certification test. Pressure tests are limited by maximum safe pressurization without hydrofracturing and by the availability of a suitable water supply. The practicality of pressure testing a well-field unit and use of tracers have not been demonstrated. It is recommended that these procedures be attempted on a pilot scale before adopting them for industry-wide use.

Rupture tests should be performed on the aquifer outside the area to be mined. These tests are used to determine the pressure required to fracture the aquifer at various depths of injection. Injection pressure during mine unit tests and operations should be restricted to a maximum safe limit below the fracture pressure at a given injection depth.

A mining unit can also be tested by commencing production and injection without introducing lixiviant or extracting uranium. Excess production fluid would be held in surge tanks or an evaporation pond until the test is completed. Monitor wells would be observed for changes in water level and the appearance of tracers as in pressure testing. Operational mode testing without lixiviant is less likely to hydrofracture the aquifer

and requires less water than does pressure testing, but it takes more time for observable monitor well responses.

An appropriate duration for certification tests is uncertain at this time. Two weeks might be a reasonable time frame, assuming that the ore zone aquifer undergoes a ten-fold stress increase during the test and that a typical vertical excursion occurs within 6 months of the initiation of mining.

3.5 EXCURSION DETECTION

Early detection of an excursion (uncontrolled migration of lixiviant and its associated dissolution products away from a well field) leads to timely corrective action to prevent loss of valuable fluids and serious contamination of ground-water resources. This section reviews present detection methodologies and their shortcomings, and suggests a rationale for selecting excursion indicators and for setting upper control limits (UCLs).

3.5.1 Present Methodologies and Their Shortcomings

The chemical indicators used to monitor for water quality deterioration, methods for establishing upper control limits (UCLs) for these indicators, and definitions of legal excursions, varied considerably among the 8 mine sites in this study. A total of eleven different indicators were used to monitor for excursions of alkaline lixiviants and the number of indicators used at each mine site ranged from 3 to 6. A single set of excursion indicators was used for all monitor wells at a given site whereas their UCLs were generally determined from baseline water quality data for individual wells. A few adjustments in indicators and their UCLs were made as mine operators gained experience. The most significant change was the increase in the UCL for uranium from a given percentage above maximum baseline (MBL) to 1 mg/L above MBL. At some sites legal excursions were identified on the basis of a) a single indicator exceeding its UCL, or b) by either a single indicator exceeding its UCL by 20% or by two indicators exceeding their UCLs. Some indicator UCLs were set between 10 and 25% above MBL concentrations; others were set between 5 and 20 mg/L (in one case, 150 mg/L) above MBL; still others were set at 10% greater than a concentration of two standard deviations above the mean or by taking the product of the standard deviation and the 90% t-distribution factor and adding it to the baseline mean value. It is unclear whether the statistical procedures were more conservative than setting UCLs at a given percentage above MBL values.

Monitor well spacings and sampling intervals were more consistent among the 8 in situ mine sites. Table 3.1 lists the most common well spacings used for pilot tests, expanded pilot tests, and commercial scale mining. Ore zone aquifer monitor wells were spaced closer together and closer to the ore zone perimeter for pilot tests in comparison to commercial scale mining. The density of upper aquifer (where present) monitor wells ranged from one per pilot test to about 2.5 per hectare for

commercial scale mining. Lower aquifers had about a fifth to a half of the number of monitor wells in the upper aquifer. Monitor wells were sampled at most facilities on a biweekly basis; when analysis indicated that an excursion had taken place the suspected well was resampled within 48 hours for verification. Monitor well placement, construction, and sampling methods are the subjects of a number of recent reports (Bishop, 1980; Durler and Bishop, 1980; and Humenick et al., 1980).

Although well placement was of some concern at the 8 mine sites in this study, the selection of excursion indicators and their UCLs created much more controversy. In some cases, the poor selection of an indicator may lead to placing a well on excursion status when no excursion actually took place. In other cases, an excursion indicator may be absorbed or precipitate before the lixiviant reaches a monitor well (Deutsch et al., 1985). Thus, the consequences of poor indicator selection range from false alarms to false senses of security.

Table 3.1. Typical monitor well spacings for ore zone aquifers[a] and densities in upper and lower aquifers[b] as related to project size.

	Horizontal Excursions		Vertical Excursions	
	Distance from Perimeter of Ore Zone (m)	Distance Between Wells (m)	Monitor Wells in Upper Aquifer #/ha	Wells in Lower Aquifer #/ha
Pilot Test[c]	30	45	1	0
Expanded Pilot Test[d]	60	90	2	1
Commercial Scale[e]	90 to 120	135 to 180	2.5	0.5

[a]For detection of horizontal excursions.

[b]For detection of vertical excursions.

[c]1 to 3 five- or seven-spot well patterns.

[d]4 to 10 five- or seven-spot well patterns.

[e]Greater than 250 wells.

Mine operators contend that the increase in concentration of an indicator in a monitor well can be caused by factors other than in situ mining. For example, natural fluctuations in ground-water quality, contamination from well cement, and contamination from open exploration holes in communication with aquifers of poor ground-water quality have been cited as possible causes of UCL exceedance. Furthermore, a monitor well provides a pathway for oxygen to reach the subsurface environment. The introduction of oxygen to a mineralized reducing zone may slowly solubilize trace elements, produce acid, and oxidize native sulfur and pyrite to sulfate, creating a false alarm where these constituents are selected as indicators.

Deutsch et al. (1985) state that unstable ions cannot necessarily be relied on as excursion indicators. For example, $SO_4^{=}$ precipitates as gypsum in the presence of excess Ca^{+2}, HCO_3^{-} precipitates as aragonite in the presence of Ca^{+2} in alkaline formation water, and U^{+6} and other trace elements become unstable as they pass through an oxygen deficient mineralized zone on their way to a monitor well.

3.5.2 Selection of Excursion Indicators

Deutsch et al. (1985) provides the general methodology for selecting excursion indicators. This methodology is based on statistical comparison of constituents in natural ground water and lixiviant and on the potential for interfering chemical reactions. Those constituents having low variances in both baseline ground water and lixiviant and much larger mean value concentrations in lixiviant are tentatively selected for use as excursion indicators. The final choice of indicators depends on the possibility that some unstable ions would be removed from solution by chemical reactions (including exchange reactions) before reaching monitor wells and on the possibility that minerals in the vicinity of a monitor well would be placed in solution by continuous exposure to the atmosphere.

The selection of indicators by Deutsch's method is aquifer dependent. The geochemistry of a confined and mineralized ore zone aquifer will normally be quite different from that of an unconfined and non-mineralized aquifer. The chemistry of confined ground water in mineralized zones is not likely to be subject to significant seasonal variations but will vary from one well to the next because of the wide range in mineralization. The chemistry of a non-mineralized and unconfined aquifer is more likely to vary in time than in space because of the seasonal introduction of oxygenated and slightly acidic rain water.

The selection of indicators is also time dependent because the chemistry of the lixiviant changes as it is recycled between the ore zone and uranium extraction exchange resins. The concentration of active complexing agents ($SO_4^{=}$ or HCO_3^{-}) will be initially high but may remain within a narrow range while the concentrations of CL^{-}, TDS, common ions and trace elements may be initially low but will rise as a function of time.

To reduce the possibility of a false alarm, at least two excursion indicators should exceed their UCLs (Section 3.5.3). Ideally, natural

concentrations of selected indicators should vary independently of one another. Further research is necessary to establish the degree of independence among selected indicators. The goal in the selection of indicators is to monitor the concentrations of a number of constituents in, or solubilized by, the lixiviant.

Horizontal Excursions

The selection of horizontal excursion indicators is influenced indirectly by uranium market conditions. In the late 1970's the high price of uranium permitted the mining of lower grade mineralized zones and monitor wells were likely to be placed outside the mineralized zone. Although under the current poor market conditions there is little mining activity, in situ mines will reopen as market conditions improve. It is unlikely, however, that the lower grade mineralized zones will be mined in the near future. Therefore, the areal extent of near future ore zones is likely to be much smaller and monitor wells are more likely to be placed inside the mineralized zone.

Trace elements, pH, alkalinity, HCO_3^-, and $SO_4^=$ are poor excursion indicators inside a mineralized zone. The increase in concentrations of these constituents could be caused by interaction with the atmosphere surrounding the monitor well. Thus, reliance on these indicators alone has the potential for causing false alarms.

Trace elements, pH, alkalinity, HCO_3^-, and $SO_4^=$ are somewhat more useful as excursion indicators in non-mineralized zones. Their increased concentrations are not likely to be caused by interaction with the local atmosphere because trace elements, native sulfur, pyrite and organic debris are not generally present in non-mineralized zones. Although carbonate minerals are present in non-mineralized zones, excess acid would not be available for solubilizing them. Therefore, the appearance of higher concentrations of trace elements, HCO_3^- and $SO_4^=$ means that an excursion is taking place. Failure to record increased concentrations of these constituents does not necessarily mean, however, that no excursion has taken place because trace elements may have precipitated out of solution in a reduction zone lying between the field and monitor wells. Similarly, $SO_4^=$ may be removed from solution in the presence of gypsum or anhydrite in the mineralized zone. Alkalinity, pH, and HCO_3^- concentrations are influenced by buffering by sediments and natural ground water both in and out of a mineralized zone.

Two other excursion indicators are far more stable and would be suitable for use in either mineralized or non-mineralized zones and oxidizing or reducing environments. These indicators are Cl^- and TDS (which increase in concentration in a lixiviant as a function of time). The TDS can be measured directly by total chemical analysis or indirectly by specific conductivity. The use of Cl^- and TDS are site-specific, however, depending on the chemical constituents of natural ground water. For example, Cl^- would not be very useful on the Texas gulf coast because natural ground water there has a high salt concentration. Some sites in Texas are also naturally high in TDS.

A summary of recommended excursion indicators is given in Table 3.2. Conductivity or TDS and Cl^- are generally suitable for monitoring horizontal excursions at specific sites as determined by comparing the chemistry of natural ground water and lixiviant. Trace elements, HCO_3^-, alkalinity, pH, and $SO_4^=$ are possible conditional indicators in non-mineralized zones. Their use in mineralized zones is problematical because of the likelihood of creating a false alarm through the introduction of oxygen during well development and sampling.

Dedicated monitor well equipment has been recently developed to prevent the introduction of oxygen and other contaminants to screened sampling intervals. However, the reliability of this equipment for use in monitoring mineralized zones has yet to be demonstrated. Further research is needed to show that oxygen can be effectively prevented from reaching sample intervals from a monitor well located in a mineralized zone.

Water level has never been used formally as an excursion indicator at any of the 8 mine sites in this study. Under site-specific conditions, however, it may be the best indicator of an early excursion. Whether or how it could be used would depend on external influences on the groundwater regime. Seasonal fluctuations of water level in a confined aquifer are not likely to occur unless there is a heavy demand on the aquifer for irrigation water or a nearby conventional mine is being dewatered. Falling water levels in monitor wells might suggest that field well injection and production rates are properly balanced and the bleed rate is sufficient. A rise in water level in one or more monitor wells would suggest that either an excursion is taking place, the monitor well is in direct communication with an overlying aquifer, irrigation has ceased in one area, or mine dewatering has ceased. If there is a nearby off-site pumping well, an unusually large rate of fall in the water level at a given monitor well might be an early indication of a horizontal excursion. Obviously, the use of water level as a horizontal excursion indicator is not clear- cut. Nevertheless, observation of water level is very promising because it can be used both inside and outside a mineralized zone and it is applicable during the initial stage of mining. Therefore, water level has been tentatively added to the list of recommended indicators of horizontal excursion in Table 3.2. Although water level changes may not always be good indicators of an excursion, they should always be observed closely once an excursion has been declared.

Data obtained during a pilot test may be very useful in establishing appropriate excursion indicators for later commercial scale mining. The appearance of trace elements, HCO_3^-, $SO_4^=$ or alkalinity in a monitor well known to be outside the mineralized zone would demonstrate that these indicators are not being detained by interfering side reactions.

Despite the uncertainty associated with the use of indicators, the mining industry's recent record of reducing and controlling horizontal excursions has been good. The above discussion was intended to organize the indicator selection process and suggest some ways in which it could be improved. Deutsch's statistical comparisons between natural ground water and lixiviants, the use of Table 3.2, and the results of pilot tests

Table 3.2. Generalized suitability of excursion indicators.

Indicators	Potential Unique Interfering Chemical Reactions or Conditions[a]	Time Dependency			
		Initial Lixiviant		Chemically Stabilized Lixiviant	
		Acid	Alkaline	Acid	Alkaline
TDS[b]	None			X	X
Alkalinity	Subject to buffering by host rock and natural ground water		X		X
Cl^-	None			X	X
HCO_3^-	Subject to buffering by host rock and natural ground water		X	X	X
pH	Subject to buffering by host rock and natural ground water	X		X	X
$SO_4^=$	Subject to chemical precipitation as gypsum	X		X	X
Trace Elements[c]	Subject to chemical precipitation and adsorption on clay minerals			X	X
Water Level	Subject to external influences such as mine dewatering and irrigation	X	X	X	X

[a]All indicators are subject to seasonal variations in unconfined aquifers. All chemical indicators are unsuitable where concentrations in natural ground water are high.

[b]Total chemical analysis (preferred) or specific conductance (an alternative at relatively low concentrations of TDS).

[c]For example, arsenic, selenium, molybdenum, vanadium and uranium.

provide a systematic methodology for selecting horizontal excursion indicators. It is recommended that water level be added to the list of excursion indicators and that trace elements, pH, alkalinity, HCO_3^-, and $SO_4^=$ be retained as positive indicators of excursion in non-mineralized zones (subject to the limitations imposed by the interfering side reactions).

Vertical Excursions

The methodology for selecting vertical excursion indicators for an aquifer underlying the ore zone is essentially the same as described above. Indicator selection strategy will depend on whether the monitored aquifer is in a mineralized zone and/or a reducing environment. Typical excursion pathways are a leaky aquitard, fault, or fracture system. Chemical interactions between lixiviant and transport media are similar so that Table 3.2 is directly applicable to monitoring an underlying aquifer.

In contrast, monitoring an overlying aquifer must be modified because the pathways of excursion are drastically different. Figure 3.1 illustrates the difference in pathways for horizontal excursions as opposed to excursions in shallow aquifers. In the latter case, chemical interactions between lixiviant and transport media are insignificant because the pathways through ruptured casings and open exploration holes are short and isolated. Furthermore, assuming upper aquifer monitor wells are not screened in a strongly reducing environment, oxidizing agents in the lixiviant are unlikely to be "used up" as they are during horizontal transport through an ore zone and vertical transport through an underlying leaky aquitard. In retrospect, it appears that it was unwise to screen monitor wells in a coal unit overlying the ore zone at Irigaray. The intent to screen monitor wells as close as possible to the ore zone is understandable. However, strata containing coal are strongly reducing environments as well as being rich sources of sulfur and trace elements. If open exploration holes and ruptured casings are the principle causes of excursion in shallow aquifers, it makes little difference how stratigraphically close to the ore zone that monitor wells are screened.

The following modifications are suggested for the monitoring of excursions in upper aquifers. First, monitor wells should not be screened in a strongly reducing environment or mineralized zone. These are far less difficult requirements to meet in an upper aquifer than in an ore zone aquifer. If these requirements can be met, trace elements and $SO_4^=$ will be, and alkalinity, HCO_3^-, and pH may be, considered as good indicators of excursion in all upper aquifer monitor wells. The possibility of a false alarm or precipitation of excursion indicators are eliminated by careful selection of screened intervals. Oxidizing agents in the lixiviant will prevent the precipitation of trace elements and $SO_4^=$. Whether alkalinity, HCO_3^-, and pH could be used depends on buffering action in natural ground water and host rock.

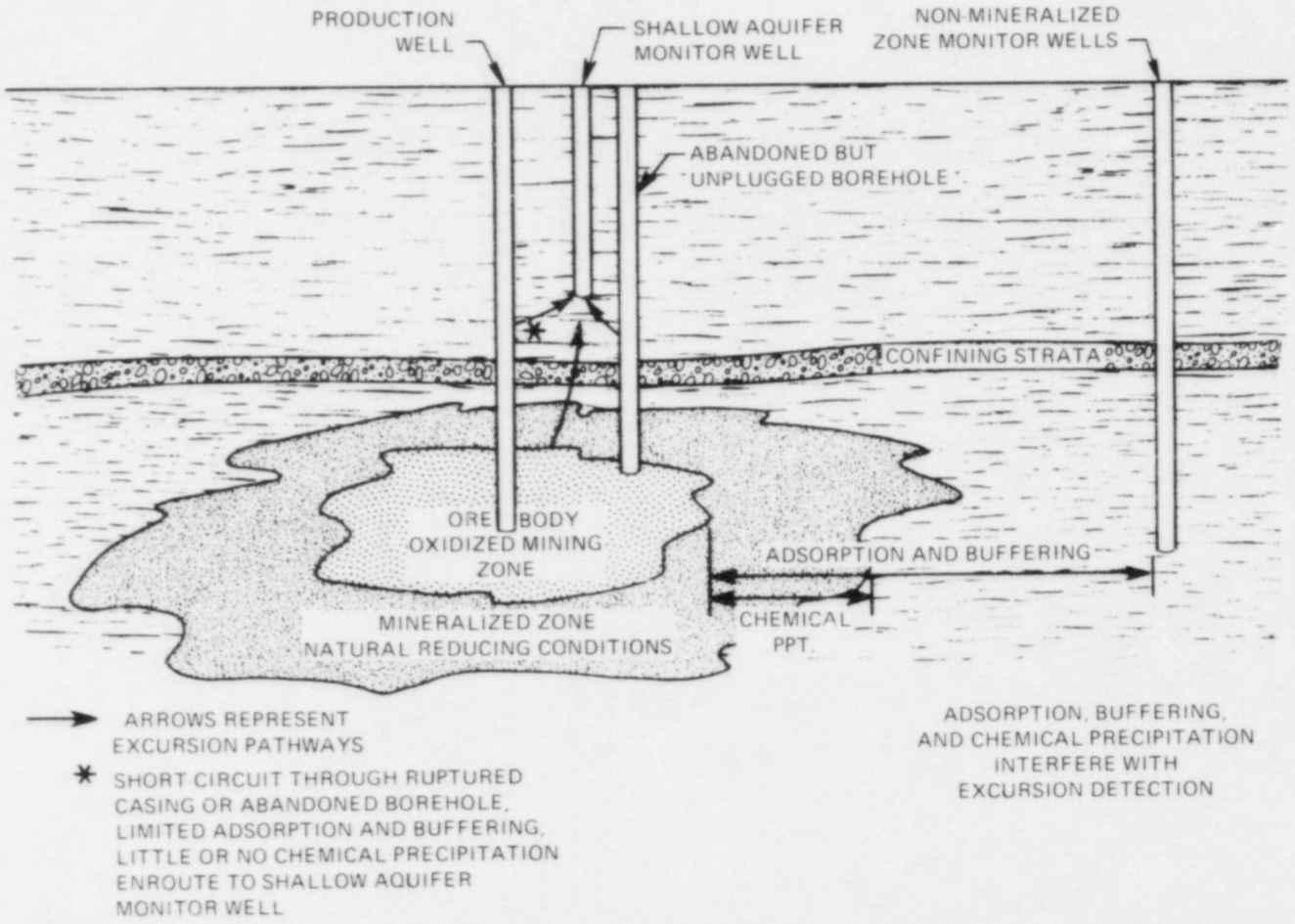


Fig. 3.1. Comparison of pathways for horizontal and vertical excursions.

3.5.3 Selection of Upper Control Limits

The methodology for setting upper control limits (UCLs) requires flexibility. The UCLs for horizontal excursions or vertical excursions in a lower (deep) aquifer need not be as conservative as those for vertical excursions in an upper (shallow) aquifer. Horizontal excursions are easier to prevent or control and vertical excursions in a deep aquifer are less likely to occur and are of minimal consequence in comparison to vertical excursions in a shallow aquifer. Furthermore, statistical methodologies are more suitable for use during commercial scale mining and for common, stable chemical indicators such as Cl^- and TDS. Judgment will be required in the use of statistical procedures for alkalinity and pH and for geochemically sensitive common anions such as HCO_3^- and $SO_4^=$. The latter indicators would be suitable in the absence of interfering chemical reactions. Statistical procedures will not be generally suitable for trace elements because of the difficulty in measuring concentrations precisely nor for water level because of external influences on the ground water regime. Also, the chemistry of the lixiviant is not generally known well enough for the application of statistical procedures at the initiation of a pilot test.

Statistical analysis assumes there are no sources of systematic error at the well being monitored. Among the potential sources of systematic error are: dissolution of minerals at the monitor well through the introduction of oxygen, poor sampling technique, deterioration of cement around the well casing, and incidental direct hydraulic contact with another aquifer of poor baseline water quality. Most of the sources of systematic error can be eliminated through the proper choice of excursion indicators and aquifers to be monitored, properly purging the well before samples are taken for analysis, careful selection of casings and cement, and using specially designed and dedicated down-hole sampling equipment where appropriate.

In the absence of sufficient statistical control (most likely in the case of a pilot test) UCLs are usually set at some fixed value above maximum baseline (MBL) values for either each individual monitor well or for a group of monitor wells surrounding a given production unit. In the case of common anions, TDS, and alkalinity (where baseline concentrations range from 10s to 1000s of mg/L) the UCLs typically are set at percentages ranging from 10 to 25% above MBL depending upon the range of baseline variability.

The above formula for setting UCLs for trace elements has led to a number of false alarms. With MBL concentrations of only a few parts per billion (ppb), a rise of a few ppb in trace element concentrations is probably meaningless. Therefore, it is recommended that the UCLs for trace elements be set at 1000 ppb (1 mg/L) above MBL or an order of magnitude above MBL, whichever is less. For example, if the MBL concentration for uranium is 1000 ppb (1 mg/L) the UCL would be 2000 ppb (2 mg/L) and if its MBL concentration is 20 ppb (0.020 mg/L) its UCL would be 200 ppb (0.200 mg/L). Such increases in uranium concentration should be sufficient indicators of an excursion.

There are several constraints in setting control limits for water level. For example, the water level in an unconfined aquifer is more subject to seasonal variation and all aquifers are affected by external sources and sinks. The use of control limits or some other mechanism for identifying an excursion by water level measurements is site-specific.

4. REFERENCES

Allen, J. R. L., 1965. A Review of the Origin and Characteristics of Recent Alluvium Sediments, Sedimentology, Vol. 5, No. 3, p. 91-191.

Bailey, R. V. and Childers, M. O., 1977. Applied Mineral Exploration With Special Reference to Uranium, Westview Press, Boulder, Colorado, 542 p.

Bishop, A. L., 1980. Non-Production Zone Excursion, Fourth Annual Uranium Seminar, South Texas Minerals Section, SME of AIME, p. 75-80.

Blatt, H., Middleton, G., and Murray, R., 1980. Origin of Sedimentary Rocks, Prentice-Hall, Inc., Englewood Cliffs, New Jersey, 782 p.

Campbell, M. D., and Biddle, K. T., 1977. Frontier Uranium Exploration in the South-Central United States, in Geology of Alternate Energy Resources, ed. M. D. Campbell, Houston Geol. Soc., p. 26-85.

Cedergren, H. R., 1977. Seepage, Drainage, and Flow Nets, 2nd ed., John Wiley and Sons, Chap. 2, p. 26-85.

Cooper, H. H., Jr. and Jacob, C. E., 1946. A Generalized Graphical Method for Evaluating Formation Constants and Summarizing Well Field History, Trans. Am. Geophys. Union, Vol. 27, No. 4, p. 526-534.

Deutsch, W. J., Martin, W. J., Eary, L. E. and Serne, R. J., 1985. Methods of Minimizing Ground-Water Contamination from In Situ Leach Uranium Mining, NUREG/CR-3709, U.S. Nuclear Regulatory Commission.

DeVoto, R. H., 1978. Uranium Geology and Exploration, Colorado School of Mines, Golden, Colorado.

Dickinson, K. A., 1976. Sedimentary Depositional Environments of Uranium and Petroleum Host Rocks of the Jackson Group, South Texas, J. U.S. Geol. Survey Res., 4, p. 615-626.

Dickinson, K. A. and Sullivan, M. W., 1976. Geology of the Brysch Uranium Mine, Karnes County, Texas, J. U.S. Geol. Surv. Res., 4, p. 397-404.

DOI, 1974. Earth Manual, Bureau of Reclamation, U.S. Department of Interior.

Durler, D. L. and A. L. Bishop, 1980. "In Situ Uranium Leach Mining: Considerations for Monitor Well Systems." Paper No. SPE 9505, Soc. Petroleum Eng., Dallas, Texas.

Fisher, R. P., 1970. Similarities, Differences and Some Genetic Problems of the Wyoming and Colorado Plateau Types of Uranium Deposits in Sandstones, Econ. Geol., 65, p. 778-784.

Fisher, R. P, 1974. Exploration Guides to New Uranium Districts and Belts, Econ. Geol., 69, p. 362-376.

Fisher, W. L., Proctor, C. V., Jr., Galloway, W. E., and Nagle, J.S., 1970. Depositional Systems in the Jackson Group of Texas--Their Relationship to Oil, Gas, and Uranium, The University of Texas at Austin, Bureau of Economic Geology Circular 70-4. Reprinted from Gulf Coast Assoc. of Geol. Soc. Trans., Vol. 20, p. 234-261.

Freeze, R. A., and Cherry, J. A., 1979. Groundwater, Prentice-Hall, Inc., Englewood Cliffs, New Jersey.

Friedman, G. M., and Sanders, J. E., 1978. Principles of Sedimentology, John Wiley & Sons, New York, 792 p.

Galloway, W. E., Murphy, T. D., Belcher, R. C., Johnson, B. D., and Sutton S., 1977. Catahoula Formation of the Texas Coastal Plain: Depositional Systems, Composition, Structural Development, Ground Water Flow History and Uranium Distribution, Report of Investigations No. 87, Bureau of Economic Geology, University of Texas at Austin.

Hantush, M. S., 1960. Modifications of the Theory of Leaky Aquifers, J. Geophys. Res., Vol. 65, No. 11, p. 3713-3725.

Hantush, M. S., and Jacob, C. E., 1955. Nonsteady Radial Flow in an Infinite Leaky Aquifer, Trans. Am. Geophys. Union, Vol. 36, No. 1, p. 93-100.

Humenick, M. J., L. J. Turk and M. P. Colchin, 1980. "Methodology for Monitoring Ground Water at Uranium Mines." Ground Water 18(3):262-273.

Kasper, D. R., Martin, H. W., Munsey, L. D., Bhappu, R. B., and Chase, C.K., 1979. Environmental Assessment of In Situ Mining, Final Report by PRC Toups and Mountain States Research and Development, under Contract No. J0265002 for U.S. Dept. Of Interior, Bureau of Mines.

Keys, W. S. and MacCary, C. M., 1971. Application of Borehole Geophysics to Water-Resources Investigations, Techniques of Water-Resources Invest. of the U.S. Geol. Surv., Chapter E1, Book 2.

Klein, G. dev., 1980. Sandstone Depositional Models for Exploration for Fossil Fuels, Burgess Publishing Co., Minneapolis, Minnesota, 149 p.

Larson, W. C., 1980. Uranium In Situ Leach Mining-Environmental Controls, Proceedings of the First International Conference on Uranium Mine Waste Disposal, Vancouver, B.C., May 19-20, 1980.

Neuman, S. P. and Witherspoon, P. A., 1972. Field Determination of the Hydraulic Properties of Leaky Multiple Aquifer Systems, Water-Resources Research, Vol. 8, No. 5, pp. 1284-1298.

NRC, 1978. Final Environmental Statement Related to Operation of Irigaray Uranium Solution Mining Project, Wyoming Mineral Corporation, Docket No. 40-8502, NUREG-0481, U.S. Dept. of Interior.

NRC, 1981. Groundwater Monitoring at Uranium In Situ Solution Mines, Staff Technical Position Paper WM-8102. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1982. Hydrogeologic Characterization of Uranium Solution Mine and Mill Tailings Disposal Sites, Draft Staff Technical Position Paper WM-803. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

O'Rourke, J. E., Essex, R. J., and Ranson, B. K., 1977. Field Permeability Test Methods with Applications to Solution Mining, Report PB 272 452 for U.S. Dept. of Interior, Bureau of Mines, Contract No. J0265045, with Woodward-Clyde Consultants, 180 p.

Osiensky, J. L., Winter, G. V., and Williams, R. E., 1984. Monitoring and Mathematical Modeling of Contaminated Ground-Water Plumes in Fluvial Environments, Ground Water, Vol. 22, No .3, p. 298-306.

Pirson, S. J., 1970. Geologic Well Log Analysis, Gulf Publishing Company, Houston, Texas.

Rackley, R. I., 1972. Environment of Wyoming Tertiary Uranium Deposits, Am. Assoc. of Pet. Geol. Bull., 56, p. 755-774.

Rackley, R. I., 1975. Problems of Converting Potential Uranium Resources into Minable Reserves: Mineral Resources and the Environment, Natl. Acad. of Sci., Washington, D.C., p. 120-140.

Reinson, G. E., 1980. Barrier Island Systems, Chapter 6, Facies Models, Geoscience Canada, Reprint Series 1, Geol. Soc. of Canada, Waterloo, Ontario, p. 57-89.

Selley, R. C., 1978. Ancient Sedimentary Environments, 2nd ed., Cornell University Press, Ithaca, New York, 287 p.

Sharp, W. N. and Gibbons, A. B, 1964. Geology and Uranium Deposits of the Southern Part of the Powder River Basin, Wyoming, U.S. Geological Survey Bulletin 1147-D.

Soister, P. E., 1968. Stratigraphy of the Wind River Formation in South-Central Wind River Basin, Wyoming, U.S. Geological Survey Professional Paper 594-A.

Spearing, D. R., 1975. Summary Sheets of Sedimentary Deposits, Am. Geol. Soc. Special Publiction MC-8.

Theis, C. V., 1935. The Relation Between the Lowering of the Piezometric Surface and the Rate and Duration of Discharge of a Well Using Groundwater Storage, Trans. Am. Geophys. Union, Vol. 16, No., 2, pp. 519-524.

Verruijt, A., 1969. Elastic Storage of Aquifers, in Flow Through Porous Media, ed. R. DeWiest, Academic Press, New York, p. 331-376.

Visher, G. S., 1965. Use of Vertical Profiles in Environmental Reconstruction, AAPG Bull., 49, p. 41-61.

Walker, R. G., and Cant, D. J., 1980. Sandy Fluvial Systems, Chapter 3, Facies Models, Geoscience Canada, Reprint Series 1, Geol. Assoc. of Canada, Waterloo, Ontario.

APPENDIX A

CASE HISTORIES

APPENDIX A. CASE HISTORIES

Before licenses are granted for commercial-scale operations at in situ uranium mines, applicants are required to conduct research and development for uranium leaching and demonstration-scale projects for aquifer restoration. Case histories of seven in situ uranium mine sites in Wyoming and one site in Texas wherein a variety of lixiviants have been tested and used are presented in this appendix. The sites reflect a variety of hydrostratigraphic conditions that must be relied upon to contain the lixiviant within the ore zone. Legal excursions have occurred at six of the mine sites. An excursion is defined, for the purpose of this report, as any uncontrolled migration of leaching fluid out of the leaching zone to an individual monitor well. Each monitor well that detects leaching fluid is considered to represent a separate excursion.

The operator and name of each uranium mine described below are as follows:

1. Wyoming Mineral Corporation (now operated by the parent company, Westinghouse Electric Corporation) - Irigaray Project,
2. Ogle Petroleum, Inc. - Bison Basin Project,
3. Rocky Mountain Energy Company - Nine Mile Lake Project,
4. Exxon Minerals Company (now operated by Everest Minerals Company) - Highland Project,
5. Teton Exploration Drilling Company - Leuenberger Project,
6. Rocky Mountain Energy Company - Reno Creek Project,
7. Cleveland Cliffs Iron Company - Collins Draw Project, and
8. Wyoming Mineral Corporation (now operated by the parent company, Westinghouse Electric Corporation) - Bruni Project.

The general locations of the mine sites are shown on Fig. A.1 and a summary of major characteristics is presented on Table A.1. The descriptions below are based on information available in 1984. Effort was made during this study by R. E. Williams, F. Anastasi, J. Osiensky, and D. Rogness of the University of Idaho to obtain all of the data that have been collected at each site.

A.1 WYOMING MINERAL CORPORATION - IRIGARAY PROJECT

A.1.1 Description of the Site

Wyoming Mineral Corporation (WMC) has operated the Irigaray mine in Johnson County, Wyoming. The mine is now operated by Westinghouse Electric Corporation, the parent company of Wyoming Mineral Corporation. The mine

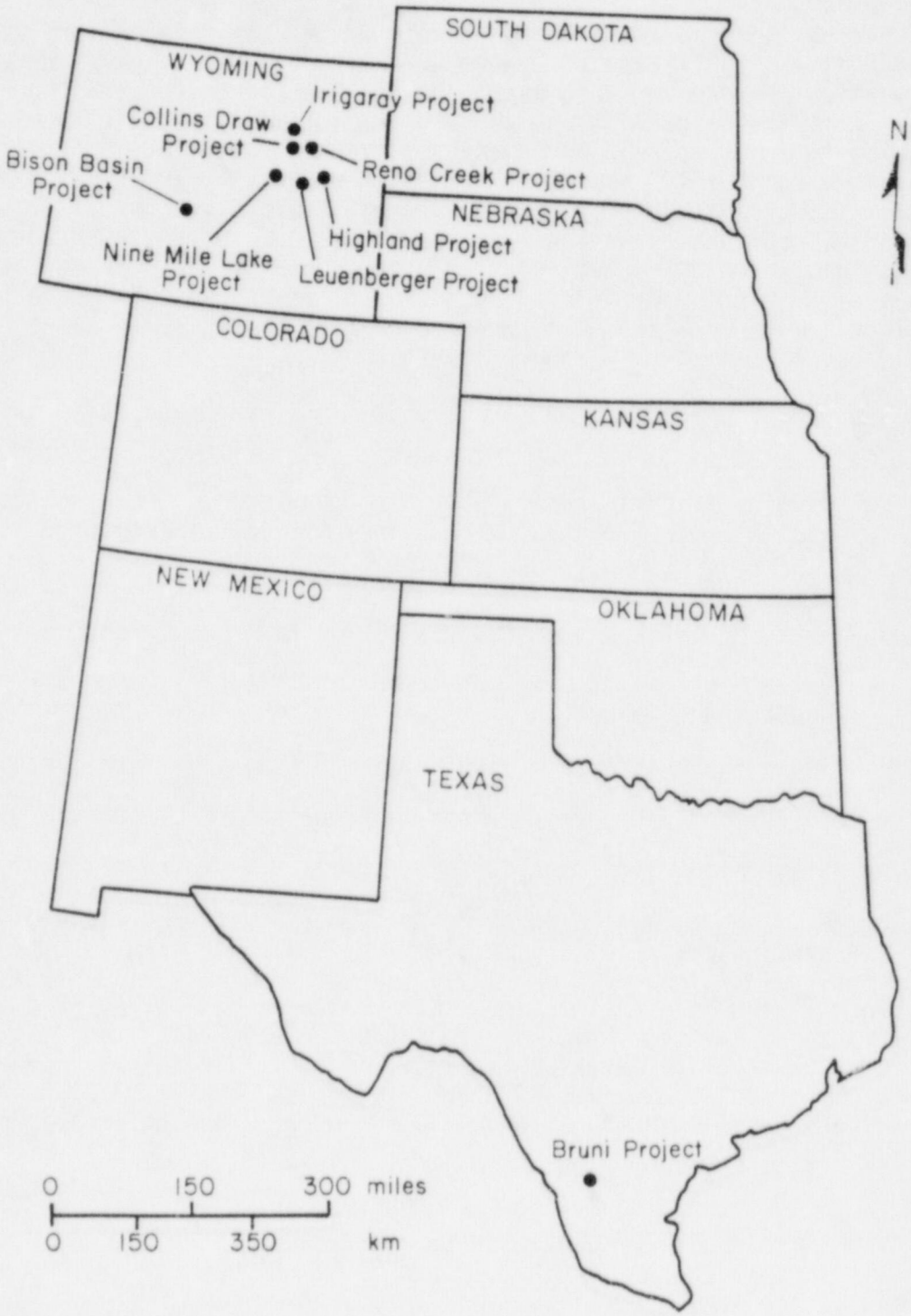


Fig. A.1. Location map of the eight in situ uranium mine sites.

Table A.1. Summary of major characteristics at seven in situ uranium mines in Wyoming and one mine in Texas.

Mine Operator and Name	Lixiviant	Geologic Setting	Ore Zone	Number of Monitor Wells Reported on Excursion[a]
Wyoming Mineral Corporation, Irigaray	NH_4HCO_3 and $NaHCO_3$	SW Powder River Basin. No faults or folds.	Sandstone in Wasatch Formation (Eocene).	10
Ogle Petroleum, Inc., Bison Basin	$NaCO_3/HCO_3$	Bison Basin. Folding and several normal faults.	Sandstone in Green River Formation (Eocene).	10
Rocky Mountain Energy Company, Nine Mile Lake	H_2SO_4 and $NaCO_3/HCO_3$	SW Powder River Basin. Folding; no faults.	Sandstone in Mesa Verde Formation (Cretaceous).	5
Exxon Minerals Company, Highland	$NaCO_3/HCO_3$	S. Powder River Basin. No faults or folds.	Sandstone in Fort Union Formation (Paleocene).	11
Teton Exploration Drilling Company, Leuenberger	$NaCO_3/HCO_3$	S. Powder River Basin. No faults or folds.	Two sandstones in Fort Union Formation (Paleocene).	0
Rocky Mountain Energy Company, Reno Creek	H_2SO_4 $NaCO_3/HCO_3$	SE Powder River Basin. No faults or folds.	Sandstone in Wasatch Formation (Eocene).	0
Cleveland Cliffs Iron Company, Collins Draw	NH_4CO_3	S. Powder River Basin. No faults or folds.	Sandstone in Wasatch Formation (Eocene).	5
Wyoming Mineral Company, Bruni	NH_4CO_3/HCO_3	Texas Coastal Plain. No faults or folds.	Clayey Sandstone in Catahoula Formation (Miocene).	>60

[a]Individual wells may have been placed on excursion status more than once.

site is located about 67 km (42 mi) southwest of Gillette, Wyoming in the west-central portion of the Powder River Basin (Fig. A.2). Topography of the mine site is characterized by gently rolling hills with deeply dissected drainages; elevations range from about 1280 to 1430 m (4200 to 4700 ft). The Powder River is the only perennial surface water in the vicinity; it flows north and is located approximately 3.2 km (2 mi) west of the mine site. Willow Creek, an intermittent stream, drains westward into the Powder River across the project area. Normal annual precipitation at the Irigaray site is between 27.9 and 40.6 cm (11 and 16 in).

Research and development testing began in November, 1975, with two pilot-scale mining operations at the 517 site and at Well Field A in Sections 5, 9, and 16, T45N, R77W (Fig. A.3). WMC was issued permits to mine uranium commercially in 1978. The first commercial production began at Production Unit 1 in November, 1978, followed by Production Units 2 through 5. Production Units 6 through 9 were developed but never entered the production phase. WMC conducted an alternate site test at Well Field E beginning in April, 1979. Demonstration restoration projects were conducted at the 517 test field and Well Field E during 1977 and 1980, respectively. Mining operations in all production units and pilot test sites ceased by mid-1981 due to excursions and a poor uranium market. WMC currently is in "standby mode" with interim shutdown of their production operations at the Irigaray site.

A.1.2 History of Mining Operations

Pilot operations at the 517 site and Well Field A used a lixiviant of ammonium bicarbonate with oxygen as the oxidizing agent. Layouts of the research and development fields are shown on Fig. A.4. The 517 site consists of three adjoining five-spot well patterns each having a central production well surrounded by four injection wells. Each 517 pattern utilized a different well spacing: 12x12 m (40x40 ft), 12x15 m (40x50 ft), and 15x15 m (50x50 ft). Leaching was conducted for a period of 75 to 100 days at each 517 test pattern. Various lixiviant concentrations were evaluated during that time.

Well Field A consists of ten adjacent, seven-spot well patterns; each pattern consists of six injection wells equally spaced 12 m (40 ft) from a central recovery well. Well Field A is located within Production Unit 1. Leaching in this well field was conducted from November, 1975, to mid-1976.

Three non-ammonium based lixiviants were tested at Well Field E, including: 1) carbon dioxide (CO_2), 2) natural ground water and 3) sodium bicarbonate. The leach tests were conducted from April, 1979, to March, 1980. Well Field E consists of seven adjacent seven-spot well patterns having a central production well spaced 12 m (40 ft) from six surrounding injection wells.

Commercial operation consisted of nine production units within a 12 hectare (ha) (30 acre) well field (Fig. A.3). Each production unit consists of a geometric array of six injection wells located about 12 m (40

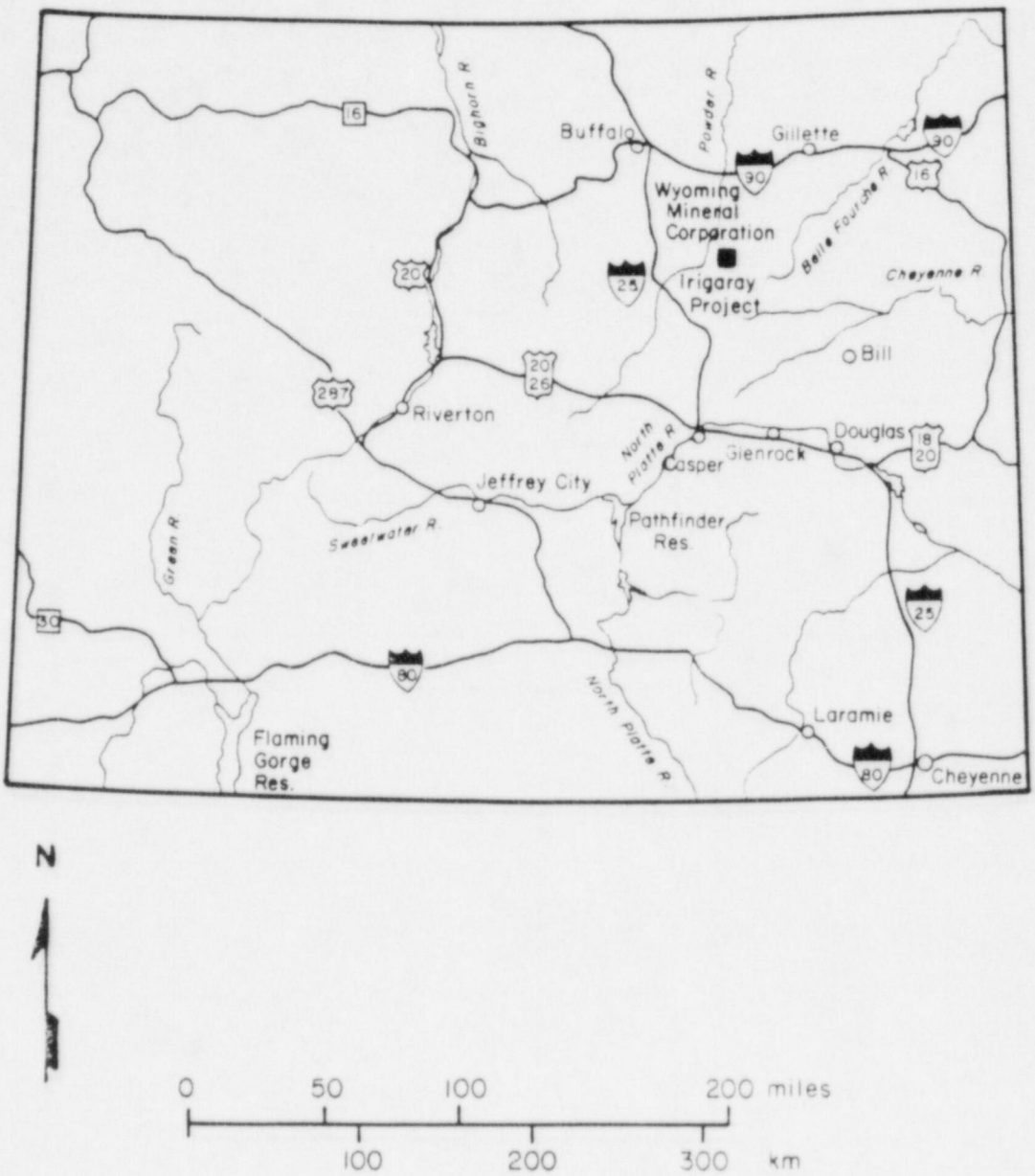


Fig. A.2. Location map of the Wyoming Mineral Corporation Irigaray mine.

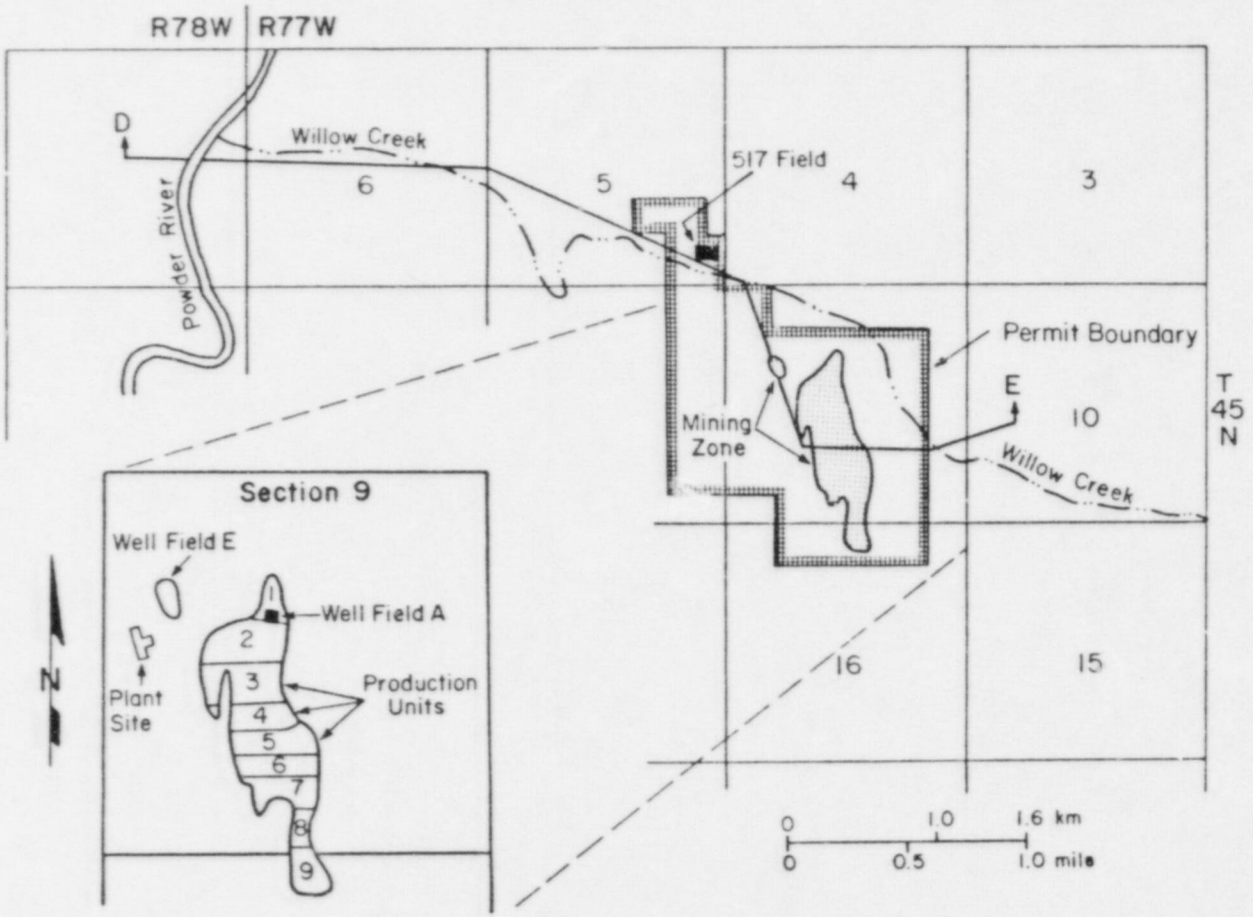


Fig. A.3. Irigaray uranium mine site and immediate vicinity.

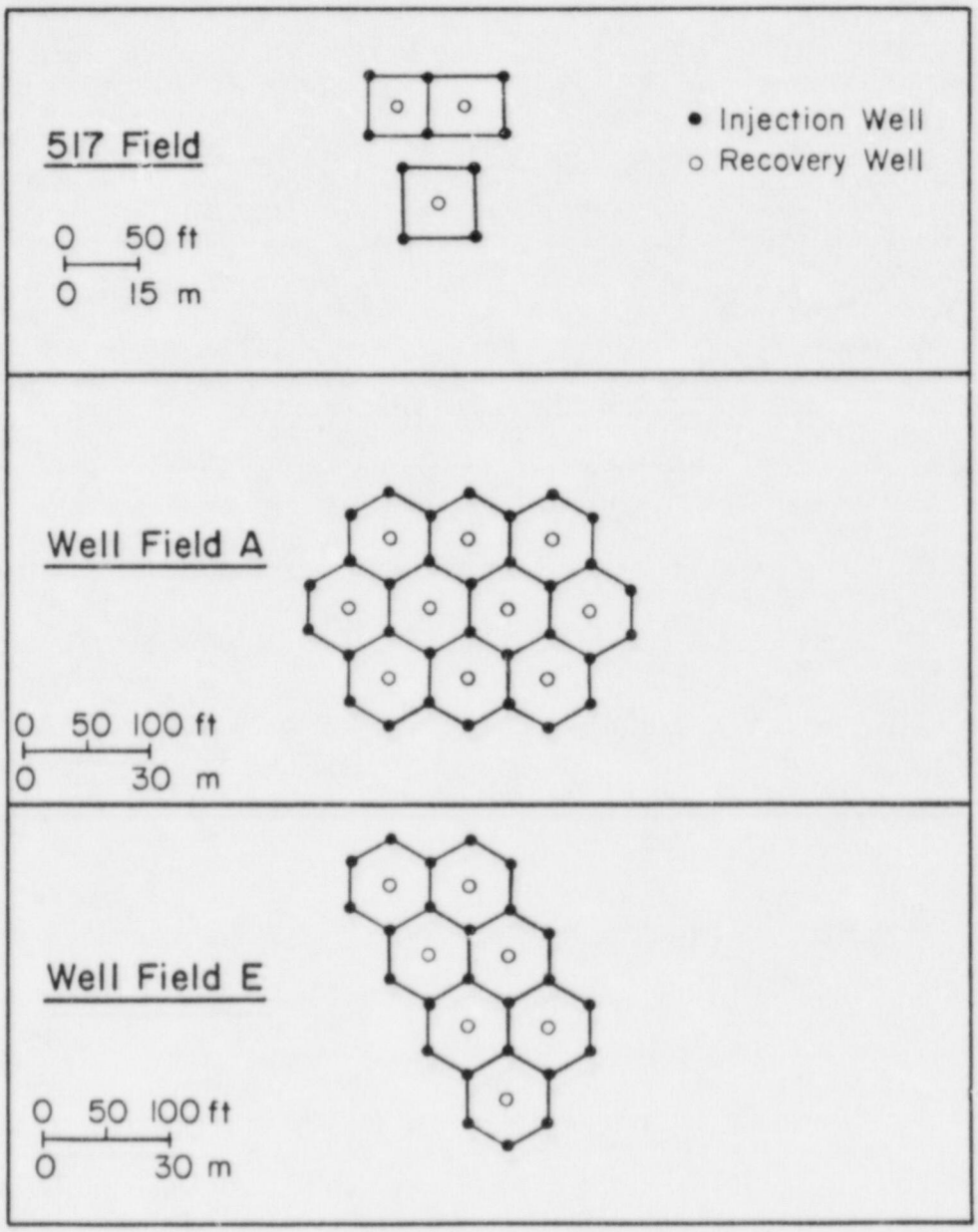


Fig. A.4. Research and development well fields at the Irigaray site.

ft) from a central production well (seven-spot pattern). Production Units 1 through 5 began operation in November, 1978, using an ammonium bicarbonate lixiviant in a concentration of 2 to 5 grams/liter (g/L); an oxidant of hydrogen peroxide or dissolved oxygen in a concentration range of 0.5 to 1.5 g/L; and a solution pH (8 to 10) that was similar to that of the host aquifer (WMC, 1981a). WMC was required by the NRC to halt use of ammonium bicarbonate as the lixiviant in May, 1980, because of excursion problems.

Injection pressures at each well were equal to or less than 9.6×10^5 Newtons/square meter (N/m^2) [140 pounds/square inch (psi)] measured at the surface. Pressures were set conservatively due to the limitation of the fracture pressure of the formation, which was estimated to be about 1.9×10^6 N/m^2 (270 psi) (WMC, 1981a). It is not known whether the pressures were maintained at less than 1.4×10^4 $N/m^2/m$ (0.63 psi/ft) of well depth, which is the minimum pressure that can initiate hydraulic fracturing. Overproduction, or bleed rate, of the well field was intended to range from about 2.5% to 8.5% above the design injection rate.

Uranium was stripped from the recovered lixiviant in ion exchange resin columns. While the uranium-bearing solution and ion exchange resin are in contact, the uranium anionic complex is sorbed by the resin, thereby displacing chloride anions sorbed on the resin during elution. Displaced chloride anions migrate into the solution; the solution is then refortified with sodium bicarbonate and hydrogen peroxide or oxygen and is reinjected into the ore zone.

Uranium sorbed by the ion exchange resins is stripped in the elution/precipitation circuit with a sodium bicarbonate and sodium chloride eluant. While the solution and resin are in contact, the eluting chloride ions are sorbed from the solution by the resins, displacing the uranium (WMC, 1981a).

A.1.3 Regional Setting

A.1.3.1 Regional Geology

The Irigaray mine site is located on the west flank of the Powder River Basin on strata that dip slightly to the northwest. The synclinal axis of the Powder River Basin is about 19 km (12 miles) west of the mine site (Fig. A.5). The surface and ore zone strata at Irigaray are part of the Wasatch Formation of Eocene age; they consist of cyclical sequences of poorly indurated sandstones, siltstones, claystones, and coals. Solution mining at Irigaray occurred in the Upper Irigaray Sandstone of the Wasatch Formation. Little tectonic activity has occurred in the Powder River Basin; no major folding or faulting has been reported.

Sediments of the Wasatch Formation were deposited in a fluvial environment of large meandering streams and coalescing braided streams. Late Paleocene uplift produced increased stream gradients which resulted in rapid erosion of the mountains (Sharp et al, 1964). Rivers overflowed their banks during flood stage and deposited layers of silt and clay over extensive areas of the basin. Coarse channel sands also were deposited,

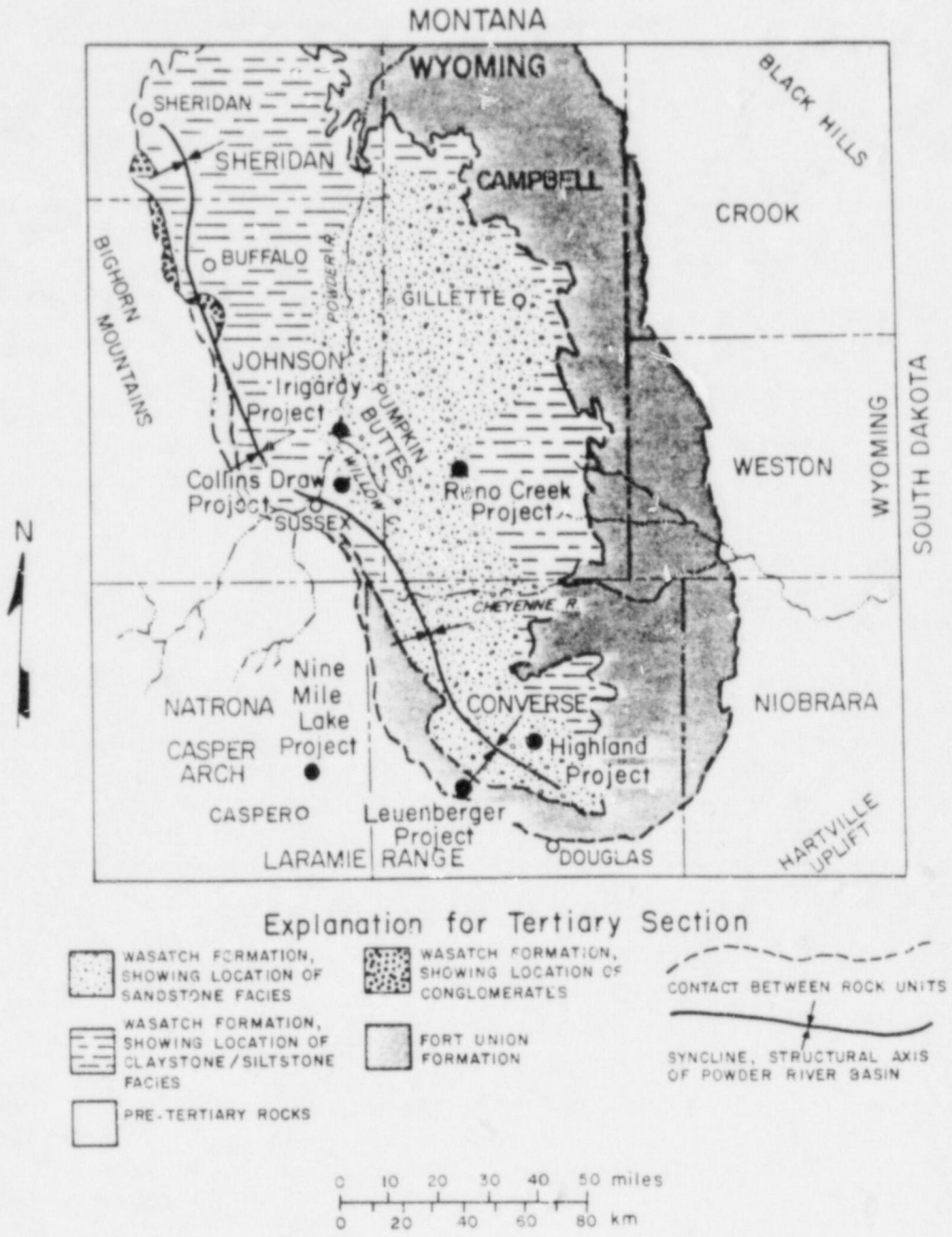


Fig. A.5. Regional geology of the Powder River Basin. *Source:* Modified from Sharp and Gibbon, 1954.

resulting in highly variable lithology (sands, silts, and clays) over short lateral distances. Matrix material in sandstones ranges from carbonate concretions to variable amounts of silt and clay.

A.1.3.2 Regional Ground-water Flow Systems

The Wasatch Formation crops out throughout the Irigaray property; its stratification controls the regional ground-water flow systems. Sandstone beds in the Wasatch Formation produce moderate quantities of water; occasional coal beds also yield significant water. Claystone aquitards form confining layers for the sandstone aquifers. Water level data for 23 wells completed in different hydrostratigraphic units within the Wasatch Formation in the vicinity of the mine site indicate that water levels are at an elevation of about 1310 m (4300 ft). Fifteen of the wells have artesian flows (WMC, 1981a).

Ground-water movement is in a northwesterly direction across the mine site area. The regional ground-water gradient is estimated to be about 0.005 (WMC, 1977a). Infiltration of direct precipitation and ephemeral streamflow is the primary source of natural recharge to the hydrostratigraphic units of the Wasatch Formation in the Powder River Basin (Hodson et al, 1973). The channel of the Powder River is believed to be a discharge area for many of the shallow Wasatch aquifers including the ore zone in this area (Hydrology Associates, Inc., 1980).

Although irrigation accounts for the largest part of total consumptive water use in the Powder River Basin, ground water is used only for livestock watering and private water supply near the Irigaray mine site. WMC (1981a) lists 28 ground-water wells within 5 km (3 mi) of the permit area. Twenty-three of these wells provide water for livestock, three are for domestic use and livestock, and two provide water only for domestic purposes. The nearest wells used for drinking water are located 6.5 km (4.4 mi) from the plant site (NRC, 1978a).

A.1.4 Description and Analysis of Data Collected

A.1.4.1 Exploration

Exploration boreholes were drilled throughout the Irigaray mine area to evaluate the extent and quality of the ore zone. WMC (1981a) lists approximately 2200 exploration boreholes and wells drilled at Irigaray. According to D'Appolonia Consulting Engineers, Inc. (1980), over 500 boreholes remained open to varying depths after drilling thereby providing the potential for hydraulic connection between various hydrostratigraphic units. The majority of exploration holes were drilled between 1970 and 1975 by exploration teams from WMC and other mining companies to evaluate the Irigaray property (WMC, 1980). The number of exploration holes drilled by other mining companies at Irigaray is not known.

An effort was begun by WMC in 1979 to locate, redrill, and plug both exploration boreholes and abandoned wells. The method adopted to locate drill holes involved removal of the surface layer of earth in the vicinity

of the surveyed location of each hole. The borehole was located by sight and cemented from the total depth to the surface.

Approximately 250 boreholes were known to exist in Production Units 1 through 6 based on exploration and well field maps; an estimated 80% of these boreholes were located and plugged as of mid-1980 (WMC, 1980). Approximately 335 abandoned wells have been plugged at the Irigaray site (WMC, 1981a).

Geophysical logs (gamma, resistivity, and induced potential) were recorded for all production, injection, and monitoring wells drilled at the Irigaray site (WMC, 1980). Approximately 100 exploration boreholes are known to have been logged geophysically at Irigaray for gamma, resistivity, and induced potential responses. In addition, boreholes NRC-1C, NRC-2C and and NRC-3C (WMC designation) were drilled and logged geophysically in 1980 to test the response of logs to different lithologies. Boreholes for the NRC monitor wells were cored, and a suite of geophysical logs including natural gamma, spontaneous potential (SP), resistivity, induced potential (IP), neutron (porosity), caliper, and gamma gamma (density) were recorded for each hole.

Identification of lithologic units was based on both the signature (shape) of the log in the vicinity of a particular rock type, and by the magnitude of the response of the resistivity, gamma, induced potential, and other logs as determined from the NRC boreholes (WMC, 1980). The Upper Irigaray Sandstone, Upper Claystone Unit, and carbonaceous/coal units exhibited unique signatures which were traceable throughout the site (WMC, 1980). D'Appolonia Consulting Engineers, Inc. (1980) correlated log response to lithology by comparing the lithologies of the three NRC boreholes with the geophysical logs.

Cross sections were constructed by D'Appolonia Consulting Engineers, Inc. (1980) using the gamma, resistivity, and induced potential logs of selected production, injection, and monitor wells. Characteristic responses of claystones, sandstones, carbonaceous claystones, and coals to each geophysical log were identified using the lithological and geophysical logs recorded for the NRC holes. Similar responses were identified on geophysical logs for wells used in the cross sections. A geologic column was drawn for each cross section borehole based on its geophysical log (WMC, 1980).

A.1.4.2 Aquifer Tests

WMC conducted several aquifer tests in various well fields and production units at the Irigaray mine site. Fourteen constant discharge pumping tests were conducted using two or more observation wells. In addition, injection tests of zones isolated by hydraulic packers were conducted in

three boreholes. Table A.2 presents a summary of the aquifer tests conducted at the Irigaray site. Locations of the aquifer tests are shown on Fig. A.6.

A summary of aquifer and aquitard characteristics estimated by WMC and independently by the authors from the aquifer test data are presented in Table A.3. The methods used to analyze the aquifer test data are also identified in this table. Drawdown data or graphical representations of test data are not available for aquifer tests 1, 2, 15, 16 and 17 to conduct further analysis.

The Hantush and Jacob (1955) leaky artesian method was also used by WMC to analyze drawdown data from aquifer tests 6, 7, 9, 11, 12, and 13 (Table A.3); however, vertical hydraulic conductivity of the Upper Claystone Unit was estimated only from aquifer tests 11 and 12 using this method of analysis. Drawdown data for the aquifer tests 3 through 14 were also analyzed by the authors using the Hantush and Jacob (1955) leaky artesian method. Drawdown data curves for the observation wells were matched with reasonable certainty to the Hantush and Jacob (1955) solution for aquifer tests 3 and 6 through 13. Tests 4, 5, and 14 could not be analyzed by the leaky artesian method because the tests were conducted for insufficient periods of time.

Injection tests (aquifer tests 15, 16 and 17) of zones isolated by hydraulic packers were conducted by WMC in three boreholes (Table A.2). The saturated horizontal hydraulic conductivity of the hydrostratigraphic units overlying the Upper Irigaray Sandstone were estimated from these tests (Table A.3). The straddle packer assemblies consisted of two inflatable packers set 1.5 m (5.0 ft) apart, fixed on perforated pipe. Tests were conducted at 1.5 m (5.0 ft) intervals over the entire length of the saturated portion of the hole. Water was pumped under pressure through the drop pipe into the tested section of the hole. Measurements of the elapsed time, and volume of water pumped at a constant pressure were recorded after the pressure and flow rate stabilized. The hydraulic conductivity values were assumed to be equal to or below $1x10^{-6}$ cm/s for all tested sections where flow into the section was not detectable. This lower limit was established due to the limited accuracy of the flowmeter used in the tests. No observation wells were used during the three injection tests.

Measurement of water levels in wells completed in aquifers overlying and underlying the Upper Irigaray Sandstone during the aquifer tests helped to locate some damaged wells and unplugged boreholes; water level declines of up to 1.4 m (4.5 ft) were attributed to the damaged wells and unplugged boreholes. These wells and boreholes were repaired or plugged by WMC. Water level declines of up to 0.5 m (1.5 ft) in overlying and underlying observation wells were not attributed to any particular phenomenon by WMC. These declines may also have been the result of leakage.

The horizontal hydraulic conductivity of the Upper Irigaray Sandstone (ore zone) ranges from about $4x10^{-5}$ to $5x10^{-4}$ cm/s (0.1 to 1.6 ft/d); storativity averages about $2x102^{-4}$. The direction of major

Table A.2. Summary of aquifer tests conducted at the Irigaray mine.

Test No.	Pumping/ Injection Well	Production Unit or Well Field	Aquifer[a]	Total Depth of Tested Well m (ft)	Number of Observation Wells	Units Monitored[a]	Type of Test[b]	Date of Test	Length of Tests (hrs)	Discharge Rate L/s (gal/min)
1	912-1	6	UISS	?	2	UISS	CD	8/75	27	1.3 (20)
2	517-6	517	UISS	?	8	UISS	CD	11/75	19	0.9 (15)
3	AP-1	1	UISS	81 (265)	5	UISS, 1-Sand	CD	2/77	15	0.6 (10)
4	BI-2	1	UISS	85 (280)	4	UISS	CD	10/18/77	5	1.1 (17)
5	CI-29	2 and 3	UISS	91 (300)	4	UISS	CD	10/18/77	5	0.9 (15)
6	BP-4	1	UISS	82 (268)	6	UISS, Coal, LISS	CD	7/19/78	24	1.6 (25)
7	DP-17	2 and 3	UISS	79 (260)	8	UISS, Coal, LISS	CD	8/29/78	23	0.9 (15)
8	EI-2	E	UISS	74 (244)	6	UISS, Coal, LISS	CD	1/05/79	24	1.3 (20)
9	FP-13	3 and 4	UISS	91 (300)	6	UISS, Coal, LISS	CD	8/31/78	18	1.3 (20)
10	9T-16	5	UISS	88 (290)	7	UISS, Coal, LISS	CD	3/16/79	48	1.0 (16)
11	HP-54	6 and 7	UISS	84 (275)	28	UISS, Coal, LISS	CD	11/13/79	48	0.8 (12)
12	KI-128	8 and 9	UISS	86 (282)	13	UISS, 1-Sand, Coal, LISS	CD	6/25/80	48	0.6 (10)
13	SSM-10	6	1-Sand	?	4	1-Sand	CD	7/01/80	24	1.6 (25)
14	SSM-21	3	1-Sand	?	3	1-Sand	CD	7/10/80	0.5	0.1 (1.0)
15	NRC-1C	6	Over B 1-Sand Inter B Coal U Clay	61 (200)	0		I/P	1980	?	?
16	NRC-2C	1	Over B 1-Sand Inter B Coal U Clay	58 (190)	0		I/P	1980	?	?
17	NRC-3C	5	1-Sand Inter B Coal U Clay	46 (150)	0		I/P	1980	?	?

[a]Over B = Overburden Unit; 1-Sand = Unit 1 Sand; Inter B = Interburden Unit; Coal = Coal Unit; U Clay = Upper Claystone; UISS = Upper Irigaray Sandstone; LISS = Lower Irigaray Sandstone.

[b]CD = Constant discharge pumping test; I/P = Injection tests of zones isolated by hydraulic packers (these tests commenced from the total hole depth upwards in 5 ft intervals until the water table or unsaturated zone was reached).

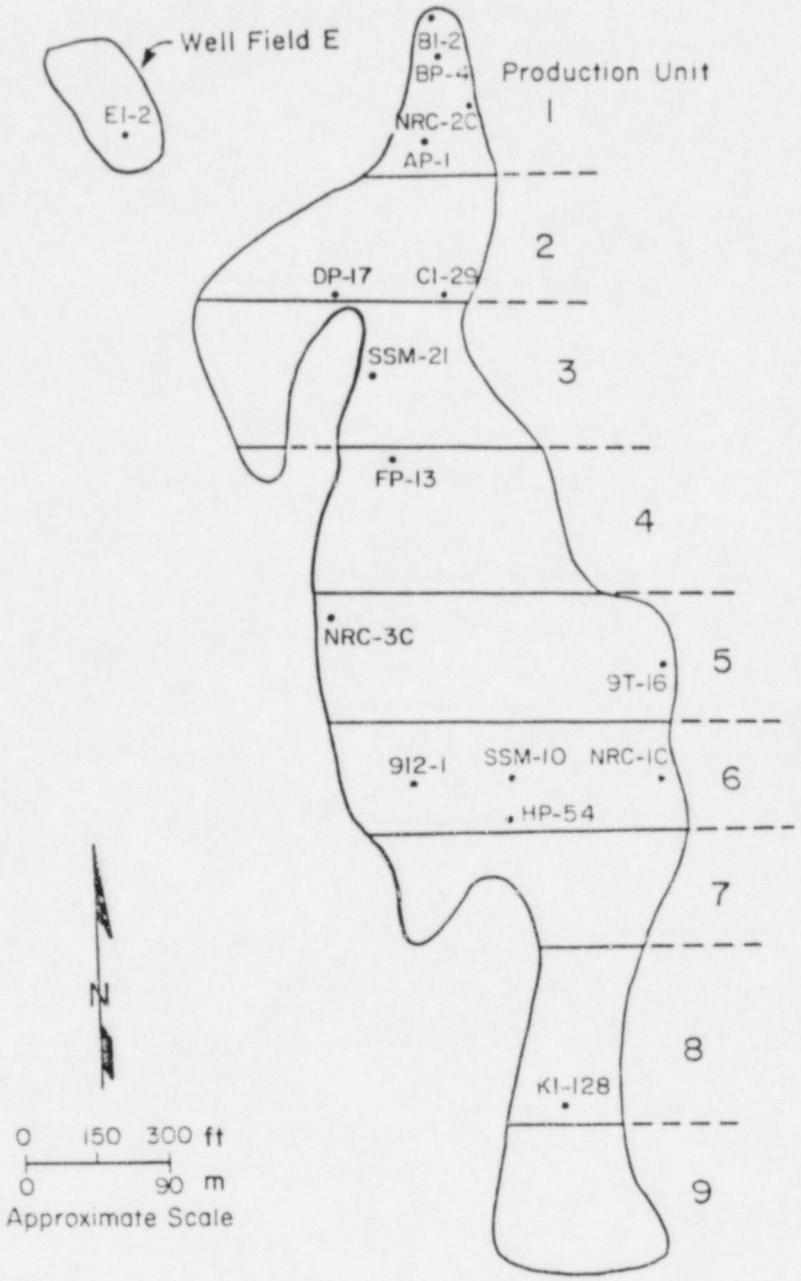


Fig. A.6. Location map of pumping wells for the aquifer tests conducted at the Irigaray mine site.

Table A.3. Summary of aquifer and aquitard characteristics estimated by Wyoming Mineral Corporation and by the authors from pumping/injection test data for the Irigaray mine.

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1	912-1	UISS	WMC	$3x10^{-4}$ to $4x10^{-4}$ (1.0 to 1.2)	11 to 14 (120 to 150)	$5.0x10^{-5}$	--	CJ	CD
2	517-6	UISS	WMC	$3x10^{-4}$ to $5x10^{-4}$ (0.9 to 1.6)	12 to 17 (130 to 180)	$1.3x10^{-4}$	--	CJ	CD
3	AP-1	UISS	WMC	--	5 to 10[f] (50 to 100)	$1.3x10^{-4}$[f]	--	CJ	CD
			Authors	$4x10^{-5}$ (0.1)	0.6 to 1.0 (7 to 10)	$3x10^{-5}$	$2x10^{-6}$ to $3x10^{-6}$ ($5x10^{-3}$ to $7x10^{-3}$)	HJ	CD
4	BI-Z	UISS	WMC	--	5 to 12[f] (56 to 130)	$2.7x10^{-4}$[f]	--	CJ,P	CD
5	CI-29	UISS	WMC	--	4 to 29[f] (40 to 310)	$2.0x10^{-3}$[f]	--	CJ,P	CD
6	BP-4	UISS	WMC	--	2 to 25 (23 to 270)	$1.8x10^{-4}$	--	HJ,P	CD
			Authors	$2x10^{-4}$ to $4x10^{-5}$ (0.5 to 1.1)	5 to 10 (50 to 110)	$1.9x10^{-4}$	$8x10^{-7}$ to $1x10^{-5}$ ($2x10^{-3}$ to $3x10^{-2}$)	HJ	CD
7	DP-17	UISS	WMC	--	2 to 15 (2 to 160)	$2.3x10^{-4}$	--	HJ,P	CD
			Authors	$1x10^{-4}$ (0.3)	2.7 to 3.1 (29 to 33)	$2.5x10^{-4}$	$7x10^{-6}$ to $2x10^{-5}$ ($2x10^{-2}$ to $5x10^{-2}$)	HJ	CD
8	EI-Z	UISS	WMC	--	1 to 4[f] (12 to 40)	$1.7x10^{-4}$[f]	--	H,S,P	CD
			Authors	$7x10^{-5}$ to $1x10^{-4}$ (0.2 to 0.3)	2 to 4 (20 to 41)	$2x10^{-4}$	$1x10^{-6}$ to $7x10^{-6}$ ($4x10^{-3}$ to $2x10^{-2}$)	HJ	CD
9	FP-13	UISS	WMC	--	1 to 10 (12 to 100)	$2.4x10^{-4}$	--	HJ,P	CD
			Authors	$7x10^{-5}$ to $2x10^{-4}$ (0.2 to 0.6)	2 to 5 (25 to 58)	$2x10^{-4}$	$3x10^{-6}$ to $7x10^{-6}$ ($6x10^{-3}$ to $2x10^{-2}$)	HJ	CD

Table A.3. (Continued)

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
10	9T-16	UISS	WMC	--	1 to 40 (11 to 430)	3.7×10	--	H,S,P	CD
			Authors	1×10^{-4} to 2×10^{-4} (0.3 to 0.6)	3 to 6 (30 to 65)	2×10^{-4}	1×10^{-7} to 6×10^{-7} (5×10^{-4} to 2×10^{-3})	HJ	CD
11	HP-54	UISS	WMC	1×10^{-4} to 3×10^{-4} (0.3 to 0.9)	3 to 10 (33 to 100)	1.4×10^{-4}	0 to 2×10^{-5} (0 to 5×10^{-2})	CJ,HJ	CD
			Authors	2×10^{-4} to 4×10^{-4} (0.6 to 1.2)	5 to 10 (50 to 100)	1×10^{-5}	1×10^{-7} to 8×10^{-7} (5×10^{-4} to 3×10^{-3})	HJ	CD
12	KI-128	UISS	WMC	6×10^{-5} to 2×10^{-4} (0.2 to 0.6)	2 to 5 (17 to 57)	1.6×10^{-4}	3×10^{-7} to 5×10^{-7} (9×10^{-4} to 2×10^{-3})	HJ	CD
			Authors	7×10^{-5} to 2×10^{-4} (0.2 to 0.6)	2 to 5 (17 to 51)	2×10^{-4}	3×10^{-7} to 5×10^{-7} (9×10^{-4} to 2×10^{-3})	HJ	CD
13	SSM-10	1-Sand	WMC	2×10^{-3} to 4×10^{-3} (4.8 to 11.6)	15 to 24 (160 to 260)	1.5×10^{-4}	--	HJ	CD
			Authors	2×10^{-3} to 3×10^{-3} (5 to 10)	14 to 23 (150 to 250)	2×10^{-4}	1×10^{-9} to 8×10^{-7} (3×10^{-6} to 2×10^{-3})	HJ	CD
14	SSM-21	1-Sand	WMC	No drawdown occured in any of the observation wells.					CD
15	NRC-1C	Over B	WMC	10^{-6} to 10^{-4} (0.003 to 0.3)	--	--	--	E	I/P
		1-Sand	WMC	1×10^{-4} (0.3)	--	--	--	E	I/P
		Inter B	WMC	$<10^{-6}$ (0.003)	--	--	--	E	I/P
		Coal	WMC	4×10^{-5} (0.1)	--	--	--	E	I/P
		U Clay	WMC	$<10^{-6}$ (0.003)	--	--	--	E	I/P

Table A.3. (Continued)

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
16	NRC-2C	Over B	WMC	10^{-6} to $5x10^{-4}$ (0.003 to 1.3)	--	--	--	E	I/P
		1-Sand	WMC	$3x10^{-5}$ to $5x10^{-4}$ (0.1 to 1.3)	--	--	--	E	I/P
		Inter B	WMC	10^{-6} to $2x10^{-4}$ (0.003 to 0.5)	--	--	--	E	I/P
		Coal	WMC	$3x10^{-5}$ (0.1)	--	--	--	E	I/P
		U Clay	WMC	$<10^{-6}$ (0.003)	--	--	--	E	I/P
17	NRC-3C	1-Sand	WMC	$1x10^{-5}$ (0.03)	--	--	--	E	I/P
		Inter B	WMC	$<10^{-6}$ to $3x10^{-5}$ (0.003 to 0.1)	--	--	--	E	I/P
		Coal	WMC	$1x10^{-4}$ (0.4)	--	--	--	E	I/P
		U Clay	WMC	$<10^{-6}$ (0.003)	--	--	--	E	I/P

[a]Over B = Overburden Unit; 1-Sand = Unit 1 Sand; Inter B = Interburden Unit; Coal = Coal Unit; U Clay = Upper Claystone; UISS = Upper Irigaray Sandstone.

[b]WMC = Wyoming Mineral Corporation.

[c]Values are based, in part, on the following assumptions: 1) thickness of the Upper Irigaray Sandstone is 100 ft, 2) thickness of the Unit 1 Sand is 25 ft, 3) thickness of the aquitard overlying the upper Irigaray Sandstone is 20 ft, and 4) the aquitards underlying the Upper Irigaray Sandstone and the Unit 1 Sand are impermeable.

[d]CJ = Cooper and Jacob (1946); H = Hantush (1960); P = Papadopulos (1965); HJ = Hantush and Jacob (1955); S = Stallman (1963) E = equation of the following form: $K = Q/2\pi LH \ln L/r$, where K = hydraulic conductivity, Q = constant rate of flow into the hole, L = length of the portion of the hole tested, H = differential head of water and r = radius of the hole tested (DOI, 1974).

[e]CD = Constant discharge pumping test; I/P = Injection tests of zones isolated by hydraulic packers.

[f]Results are questionable or not valid because method(s) of analysis were used improperly.

Source: WMC, 1977a, 1980, 1981a, and 1981b.

hydraulic conductivity estimated by WMC from nine aquifer tests conducted in the Upper Irigaray Sandstone is approximately north-south; this direction coincides with the major trend of the uranium ore body. The Upper Irigaray Sandstone appears to be anisotropic and heterogeneous to some extent on the basis of the aquifer test results.

Results from eight aquifer tests (3, 6, 7, 8, 9, 10, 11, and 12) conducted in the Upper Irigaray Sandstone (Table A.3) suggest that water probably leaks through the overlying and/or underlying confining units under pumping conditions. Use of the Hantush and Jacob (1955) leaky artesian method by the authors allowed an estimate of vertical hydraulic conductivity of the overlying confining bed (Upper Claystone Unit). These vertical hydraulic conductivity values ranged from $1x10^{-7}$ to $3x10^{-5}$ cm/s ($3x10^{-4}$ to $6x10^{-2}$ ft/d). WMC estimated vertical hydraulic conductivity of the confining bed using the Hantush and Jacob (1955) method for aquifer tests 11 and 12; their estimates range from about $3x10^{-7}$ to $2x10^{-5}$ cm/s ($9x10^{-4}$ to $6x10^{-2}$ ft/d). Specific storage of the confining bed cannot be estimated using the Hantush and Jacob (1955) method of analysis. The authors estimated the vertical hydraulic conductivity of the aquitard overlying the Unit 1 Sand from data for aquifer test 12; the resulting values range from $1x10^{-9}$ to $8x10^{-7}$ cm/s ($3x10^{-6}$ to $2x10^{-3}$ ft/d).

Although direct pathways for leakage cannot be determined from results of the aquifer tests conducted at Irigaray, reasonably certain matches of drawdown data to the Hantush and Jacob (1955) solution indicate that water may be leaking through the aquitards from the overlying and/or underlying aquifers. Rapid leveling responses of the data curves indicate that leakage likely is occurring through 1) abandoned boreholes, 2) leakage through damaged wells, 3) a thin confining layer, 4) a permeable confining layer, and/or 5) fracture/fault zones. These pathways of leakage will be discussed in the following sections.

A.1.5 Hydrogeology of the Mine Site

Uranium mineralization at the Irigaray site occurs as roll-front deposits in the fluvial "Upper" and "Lower" Irigaray sandstones of the Wasatch Formation; however, only the Upper Irigaray Sandstone has been mined as of 1984. The Upper Irigaray Sandstone was deposited as a channel sand by a north-flowing stream (NRC, 1978a). The Upper Irigaray Sandstone averages 30 m (100 ft) in thickness and is present 23 to 150 m (75 to 500 ft) below land surface due to topographic relief. Sands in this unit grade vertically from coarse-grained at the bottom to fine-grained at the top; at least two periods of downcutting and subsequent filling occurred at the mine site (NRC, 1978a). Uranium has been deposited in the ore roll-front as uraninite and coffinite. Arsenic, selenium, and barite are present within and/or adjacent to the uranium mineralization. Calcite content in the ore zone ranges from 1% to 3%; pyrite content generally is less than 0.5%.

Fig. A.7 is a geologic column of a part of the Wasatch Formation at the site. Detailed stratigraphic and lithologic relationships for the site

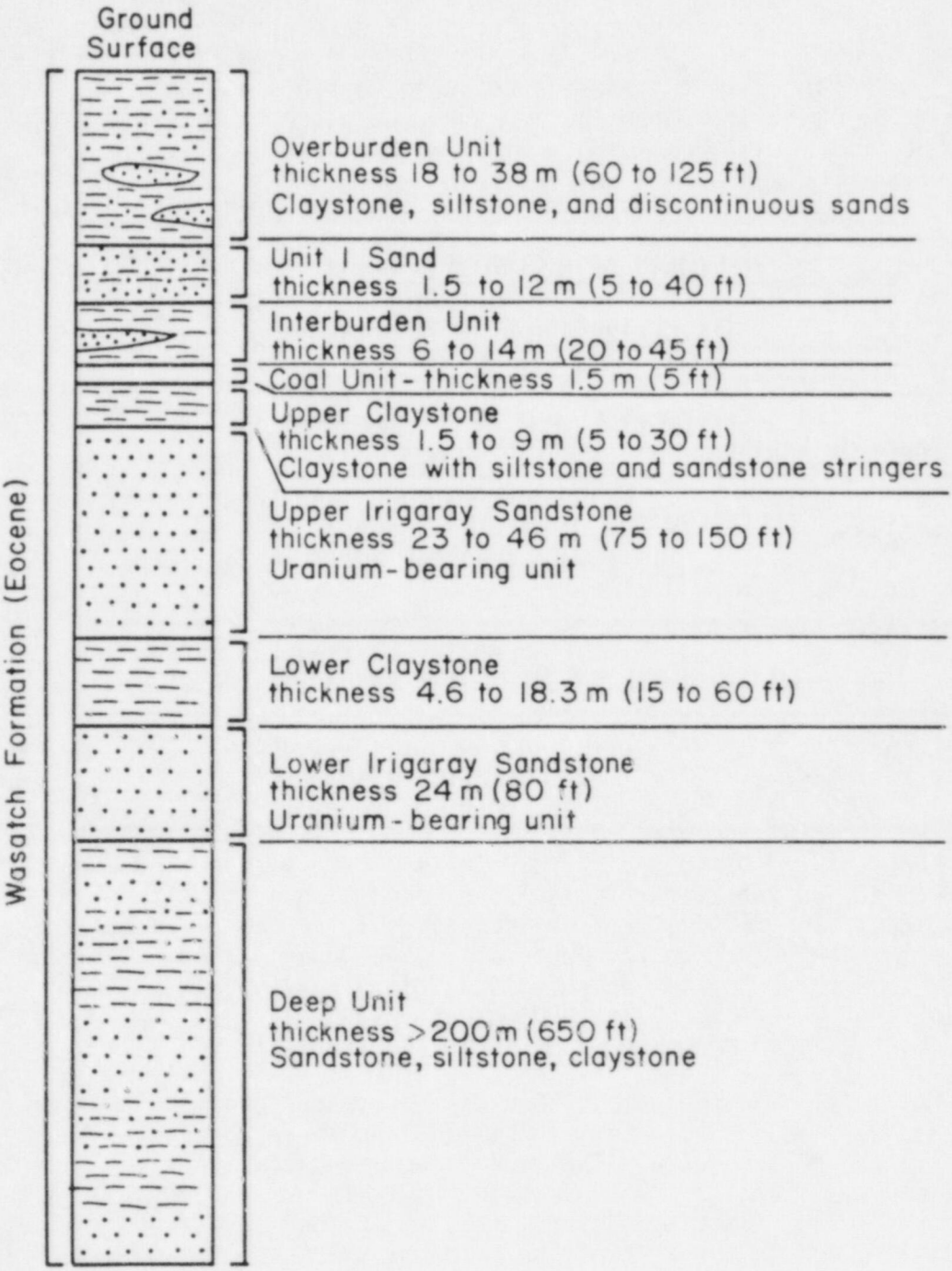


Fig. A.7. Stratigraphic section for the Irigaray project area. *Source:* Modified from NRC, 1978a.

have been developed by WMC (1981a) and D'Appolonia Consulting Engineers, Inc. (1980) based on direct correlations from boreholes, geophysical logs, and lithological logs. The Lower Claystone Unit that separates the Upper and Lower Irigaray sandstones is believed to pinch out somewhere to the east of the site whereupon the two sandstones become a single hydrostratigraphic unit.

The Wasatch Formation crops out at the mine area; it dips to the northwest from 1 to 2 degrees. A simplified cross section at Irigaray based on a limited number of wells is shown in Fig. A.8. The actual hydrostratigraphy is believed to be much more complex with scour and fill deposits and lenses. No faults or folds have been recognized in the mine area. D'Appolonia Consulting Engineers, Inc. (1980) drew a structure contour map of the Coal Unit at Irigaray; no geologic discontinuities were apparent in this unit. Outcrop areas of the Upper Irigaray Sandstone have not been identified.

Ground water occurs under both confined and unconfined conditions at the Irigaray site. The water table is located approximately 15 to 21 m (50 to 70 ft) below ground surface in the Overburden Unit of the Wasatch Formation. The Overburden Unit has relatively low hydraulic conductivity due primarily to the claystones occurring at depths of from 15 to 30 m (50 to 100 ft). The confined aquifers that occur below the Overburden Unit in descending order are as follows: Unit 1 Sand, Coal Unit, Upper Irigaray Sandstone, Lower Irigaray Sandstone, and the Deep Unit (Fig. A.7). These aquifers are confined by claystone units that contain sandstone and siltstone stringers.

WMC (1981b) reported variability in the lithologic character and thickness of the Upper Claystone Unit immediately above the Upper Irigaray Sandstone (ore zone), particularly in Production Units 2 and 3. Hydrology Associates, Inc. (1979) examined geophysical and lithological logs for the Irigaray site and found the Upper Claystone Unit to vary in thickness from 0 to 7.6 m (0 to 25 ft) in Production Units 1 through 5. D'Appolonia Consulting Engineers, Inc. (1980) constructed an isopach map of the Upper Claystone Unit based on selected holes in Production Units 1 through 9; the Upper Claystone Unit was shown to vary in thickness from about 5.5 to 7.6 m (18 to 25 ft). An adequate number of geophysical and/or lithological logs are not available to identify specific areas where the Upper Claystone Unit is thin and/or permeable. The predominant premining vertical ground-water gradient was directed upward from the Lower Irigaray Sandstone to the Coal Unit (Hydrology Associates, Inc., 1980). The horizontal hydraulic gradient averages 0.033 to the west at the Irigaray pilot-scale test site 517. Local variations in the flow rate are to be expected in the fluvial sedimentary deposits at Irigaray. Ground water at Irigaray moves northwest approximately parallel to the long axis of the ore body.

A.1.6 Monitoring Well System and Well Construction

Monitoring wells were placed in the Upper Irigaray Sandstone (ore zone), Lower Irigaray Sandstone, Coal Unit, and the Unit 1 Sand. Trend wells were utilized in the Upper Irigaray Sandstone and Coal Unit. Trend

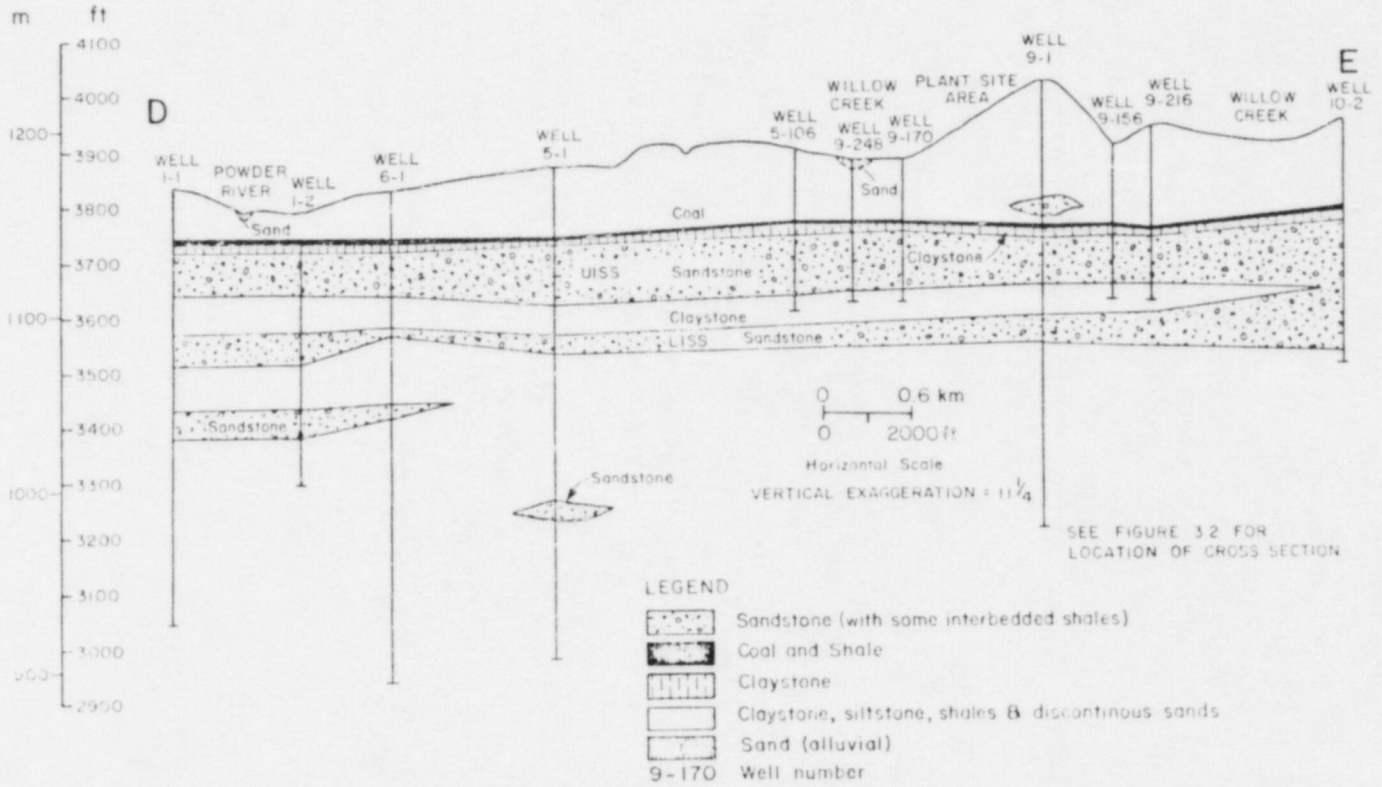


Fig. A.8. Simplified cross section through the Irigaray site. *Source:* NRC, 1978a.

wells were utilized to provide early warning of possible lixiviant movement away from the well field toward the monitor wells. Selected monitoring wells and trend wells continue to be sampled routinely during standby status at Irigaray. The following discussion describes the more extensive monitoring well system used during production operations.

Monitoring and trend wells were sampled every two weeks for the excursion indicators a) electrical conductivity (conductivity), b) chloride, and c) alkalinity. Conductivity provides a general measure of ion concentration. Chloride is introduced to lixiviant by the ion exchange process to replace uranium. A carbonate/bicarbonate lixiviant is used for leaching. Uranium and pH were added to the assay list for monitoring during an excursion status, with an initial analysis for arsenic and selenium; no data are available concerning concentrations of these parameters during excursion and clean-up status. Upper control limits (UCL's) were established by WMC (1981a,b,c) for the excursion indicators for each monitoring well.

UCL's for the Upper and Lower Irigaray sandstones and the Coal Unit were set at 5 mg/L above baseline for chloride, and 15% and 10% above baseline for alkalinity and conductivity, respectively. UCL's for the Unit 1 Sand were established at 10 mg/L above baseline for chloride, and 20% above baseline for conductivity and alkalinity; these higher UCL's for the Unit 1 Sand were allowed by the NRC due to the apparent natural variability of water quality at the Irigaray site (WMC, 1981a). This variable water quality is due mainly to high levels of TDS and sulfate in some areas of the well field; the reason for these high levels is not known. Monitoring wells are placed on excursion status when any two of the three indicators exceed their UCL concentration as verified by a second sample taken within 48 hours.

Monitoring wells completed in the Upper Irigaray Sandstone are located no farther than 120 m (400 ft) from the well field boundary. A distance ranging between 120 to 180 m (400 to 600 ft) separates each monitoring well. Trend wells are located no farther than 60 m (200 ft) from the well field boundary. Deep monitoring wells were drilled into the Lower Irigaray Sandstone with at least one well for each production unit 0.8 to 2 ha (2 to 5 acres) in size; the monitoring well generally is located in the center of and beneath the extraction well field.

The Coal Unit was established originally as the shallow monitor zone above the production zone. Later, due to low hydraulic conductivity values, D'Appolonia Consulting Engineers, Inc. (1980) convinced the NRC that the Coal Unit should be considered part of the confining layer overlying the Upper Irigaray Sandstone. WMC was allowed to establish a new upper monitor zone in the Unit 1 Sand in November, 1981 (WMC, 1981a).

Wells to monitor vertical excursions were completed into the Unit 1 Sand and the Coal Unit within the perimeter of the well field at a density of about one well per 0.4 ha (1 acre). Coal Unit wells inside the mining perimeter are designated as trend wells; all others are monitoring wells. These monitoring wells are located approximately 120 m (400 ft) from the well field perimeter and are spaced about 180 m (600 ft) apart. The Coal

Unit monitoring wells provided an early warning system and detected horizontal excursions outside the well field area through this unit.

Wells at Irigaray were drilled using a standard rotary rig and inorganic drilling mud. Holes were drilled through the target zone. The holes were logged geophysically to evaluate the thickness and grade of uranium mineralization. Well screen or casing usually was not used in the Upper Irigaray Sandstone because of its stability. Casing of PVC plastic pipe with a plug at the bottom was set to the top of the designated interval; cement was then pumped into the well annulus through weep holes at the base of the casing. The well annulus from the base of the casing upward to the ground surface. Once the cement hardened, the cement remaining in the casing was drilled out along with the plug attached to the bottom of the casing. The target zone below the casing often was underreamed to increase injectivity of fluid into the ore zone. Each well was cleaned and developed upon completion. WMC generally tested well integrity by using high pressure injection tests between packers, and/or by televiewing.

A.1.7 Analysis of Excursions

WMC's Irigaray mine has a history of numerous excursions. Most excursions have been detected in the Coal Unit which was designated originally as the shallow monitor zone. Fluid migration was first detected in the Coal Unit during the spring of 1979. On April 21, 1980, the NRC ordered WMC to cease operations and identify the cause of the shallow zone excursions. Mining was allowed to resume on May 23, 1980. Table A.4 summarizes the excursions that have occurred at Irigaray as of June, 1984. A total of seven Coal Unit monitoring wells have been on excursion status; three additional monitoring wells, one in the Unit 1 Sand and two in the Upper Irigaray Sandstone, have been reported on excursion status since mining operations began. All excursions except the one in Upper Irigaray Sandstone well 9M-2 have shown elevated concentrations of chloride. Excursions have been reported in Production Units 1 through 5 and in Well Field E. Locations of monitoring wells reported on excursion status at the Irigaray mine are shown in Fig. A.9.

The injection of sodium bicarbonate and oxygen into Production Units 1 through 5 was discontinued on March 27, 1981, due to poor uranium market conditions. Uranium extraction continued with the circulation of existing solutions in the well fields and the addition of carbon dioxide for pH control. This circulation phase continued until July, 1981, for all mining units (WMC, 1981d).

Three Coal Unit monitoring wells (SM-1, SM-6, SM-7) in Production Units 1 and 2 were confirmed on excursion status during the period of March 27 to April 12, 1979, based on elevated chloride and alkalinity. Overpumping from the production zone was initiated immediately in the vicinity of contamination. Direct pumping from two Coal Unit monitoring wells began in July, 1979. Concentrations of excursion indicators were reduced below the UCL's by August 23, 1979, for wells SM-1 and SM-7. Well SM-6 was plugged and later replaced by well RS-19. The new well was initially on excursion status; it was taken off excursion status near the end of 1982.

Table A.4. Summary of excursions at the Irigaray mine site during mining operations.

Well	Location	Aquifer	Excursion Parameters				Excursion History		
			Date of Sample[a]	Chloride mg/L	Conductivity µmhos/cm	Alkalinity mg/L	On	Off	Duration
SM-1	Production Unit 1	Coal Unit	UCL	28.8	791	129.3	3/27/79	9/23/79	5 months
			3/27/79	--	--	--	6/04/81	8/03/81	2 months
			6/15/81	43.8	733	130.0			
SM-7	Production Unit 2	Coal Unit	UCL	17.8	742	115.9	3/27/79	8/21/79	5 months
			3/27/79	--	--	--	1/05/81	8/03/81	7 months
			1/05/81	32.3	789	109.0			
RS-19 (SM-6)	Production Unit 3	Coal Unit	UCL	17.0	828	124.8	4/10/79	10/82	6 months
			4/10/79	--	--	--			
			3/29/82	32.6	684	133.6			
SM-12 (SM-8)	Well Field E	Coal Unit	UCL	16.7	1014	139.4	9/05/79	10/82	3 years
			9/05/79	--	--	--			
			4/28/80	62.4	797	203.0			
SM-4	Production Unit 4	Coal Unit	UCL	17.0	779	149.3	3/21/80	7/28/80	4 months
			3/21/80	21.7	912	114.0	1/05/81	5/30/82	17 months
			6/16/80	153.7	3187	124.0			
			2/22/82	60.5	1434	165.6			
SM-9	Production Unit 5	Coal Unit	UCL	18.8	897	119.9	6/30/80	7/22/80	1 month
			6/30/80	21.5	787	122.0	1/10/82	5/30/82	4 months
			7/01/80	59.8	892	179.0			
			1/10/82	19.8	635	138.2			
SM-14	Production Unit 5	Coal Unit	UCL	18.5	794	137.2	4/08/80	7/02/80	8 months
			4/08/80	27.6	690	238.0			
			6/16/80	34.4	709	215.0			
9M-22	Production Unit 5	Upper Irigaray SS	UCL	16.9	646	101.2	5/05/80	8/81	15 months
			5/05/81	12.0	1442	113.2			
			7/20/81	11.6	1245	109.6			
SSM-5	Production Unit 3	Unit 1 Sand	UCL	31.8	883	252.0	5/29/81	9/08/81	3 months
			5/29/81	45.3	?	?			
			6/08/81	65.0	?	?			
9M-1	Production Unit 1	Upper Irigaray SS	UCL	26.0	780	112.0	1/08/80	3/31/80	3 months
			1/08/80	20.8	1005	135.6			
			1/21/80	139.8	1710	406.0			

[a]UCL = Upper Control Limit; the first sample date is when the well went on excursion status; dates that follow are representative of well on excursion status giving highest concentrations achieved or giving concentrations of interest.

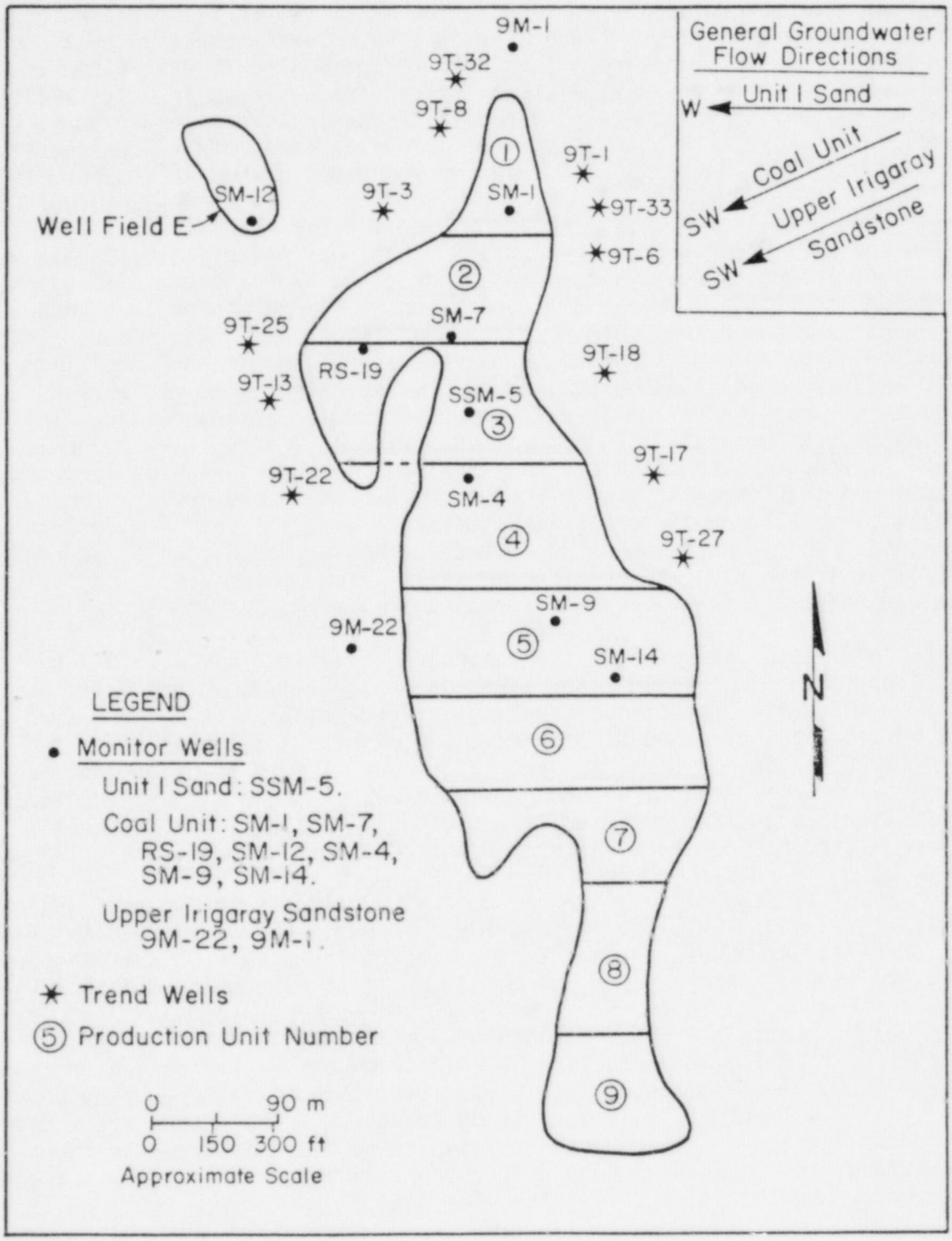


Fig. A.9. Location of trend wells and monitoring wells reported on excursion status at the Irigaray mine.

Three Coal Unit monitoring wells (SM-4, SM-9, SM-14) in Production Units 4 and 5 exceeded their UCL's for at least two of the three excursion indicators between March 21 and June 5, 1980. Over production of the Upper Irigaray Sandstone and direct pumping from the Coal Unit was initiated. All three of these wells were taken off excursion status in July, 1980. Wells SM-4 and SM-9 again were placed on excursion status in January, 1981, and January, 1982, respectively. Well SM-9 was taken off excursion status on May 30, 1982, and well SM-4 returned to normal status in October, 1982.

WMC investigated possible reasons for the excursions in wells SM-1, SM-6 and SM-7 beginning in April, 1979. Geologic and hydrologic data were studied, including geophysical logs, core data, geologic cross sections, and pump test data. WMC (1980) could find no evidence of natural hydraulic connection between the Upper Irigaray Sandstone and the Coal Unit. Injection tests between packers and televiewer scanning were conducted in over 800 wells in Production Units 1 through 4 beginning in April, 1979. These investigations identified 62 wells as having major casing damage. The wells were either repaired or they were cemented and replaced by April, 1980. Based on data collected and examined in these investigations, WMC concluded that the most probable cause of excursions observed in Production Units 1 and 2 was movement of lixiviant into the Coal Unit via damaged well casings. This conclusion is supported further by reduced chloride concentrations in the vicinity of affected wells after correction of casing damage and the initiation of overpumping.

Information gained from the excursion investigation in 1979 was applied to the analysis of excursions in wells SM-4, SM-9 and SM-14 in 1980. All wells suspected of leaking in Production Units 4 and 5 were either repaired or cemented and replaced prior to initiation of mining (WMC, 1980). This led WMC to conclude that the cause of excursions in Production Units 4 and 5 was not casing damage. Geologic and hydrologic data were reviewed again by WMC to assess the possibility of natural hydraulic connection between the production unit and coal unit.

Potential for fluid migration due to thinning of the Upper Claystone Unit was examined by plotting chloride concentrations versus bed thickness; no correlation between the two factors was observed (WMC, 1980). According to WMC (1980), the Upper Claystone Unit is relatively uniform in thickness; it varies from 5.2 to 7.3 m (17 to 24 ft). However, Hydrology Associates, Inc. (1979) found the Upper Claystone Unit to vary in thickness from 0 to 7.6 m (0 to 25 ft) in Production Units 1 through 5. Examination of available lithological and geophysical logs reveals that the Upper Claystone Unit is from about 1.5 to 9 m (5 to 30 ft) thick. The entire Upper Claystone Unit often is interbedded with sandstone and siltstone stringers. Therefore, the Upper Claystone Unit appears to be variable in lithologic character and thickness across the mine site.

WMC (1980) drew a structural contour map of the Coal Unit to ascertain the existence of irregularities, such as faults or folds, in the beds overlying the Upper Irigaray Sandstone. The structure map shows that the Coal Unit dips gently to the northwest. The map does not indicate the existence of folds or faults within the well field. As a result of this

investigation, WMC (1980) concluded that structural discontinuities probably are not the cause of excursions within the mine area.

Hydraulic fracturing of the overlying hydrostratigraphic units due to excessive fluid injection pressures may have occurred allowing the upward movement of lixiviant. WMC (1980) compared the areal extent of excursion migrations and chloride concentrations against the overburden isopach; no definite or strong correlation was apparent between the factors. It was decided that no further conclusions regarding this relationship could be drawn (WMC, 1980).

If excursions are occurring through open boreholes, and all boreholes are equal in their ability to transmit fluids, areas of higher open borehole density should show greater concentrations of migrated lixiviant. WMC (1980) attempted to correlate known boreholes with the 1979 and 1980 Coal Unit excursions. An effort was begun to locate, redrill and cement all abandoned open boreholes. Concurrent with over production of selected production zone wells in Production Units 4 and 5, the plugging of boreholes appeared to help control the excursions in wells SM-4, SM-9 and SM-14.

As a result of these diagnostic tests, WMC (1980) concluded that the most likely pathways for lixiviant migration to the Coal Unit in Production Units 4 and 5 during 1980 were unplugged exploration boreholes. Exploration holes probably contributed to excursions in Production Units 1 and 2 in addition to contributions by damaged well casings already identified. Even though casing damage and exploration boreholes appear to be significant causes for excursions into the Coal Unit, the relatively thin nature and lack of lithologic uniformity of the Upper Claystone Unit may have been additional factors in vertical lixiviant movement. Results from several aquifer tests in the Upper Irigaray Sandstone indicated the occurrence of leaky conditions which may be a result of natural conditions in the Upper Claystone Unit or from mining activities such as open boreholes and casing damage.

Chloride and alkalinity were detected above their UCL's in Coal Unit monitoring well SM-8 in Well Field E on September 5, 1979. WMC (1980) believed that well SM-8 was being affected by lixiviant migration from the adjacent production units and by possible cement contamination. Well SM-12 later replaced well SM-8 and fluctuated on and off excursion status until late 1982 when well SM-12 was taken off excursion status. Leaching processes using sodium bicarbonate occurred from April, 1979, to March, 1980, in Well Field E; restoration was conducted from March 15 to August 27, 1980.

The UCL's for chloride, conductivity, and alkalinity were exceeded on January 8, 1980, in Upper Irigaray Sandstone monitoring well 9M-1 in Production Unit 1. The excursion lasted only about three months, although concentrations of the three elevated indicators reached very high levels (Table A.4). WMC did not address the excursion in well 9M-1 and information about possible corrective actions is not available. It is not possible to ascertain the cause of this excursion due to lack of information.

Well 9M-1 is located directly north of the Production Unit 1 well field which is nearly opposite the direction of natural ground-water flow in Upper Irigaray Sandstone (southwest).

A water sample taken from monitoring well 9M-22, completed in the Upper Irigaray Sandstone in Production Unit 5, exceeded UCL's for conductivity and alkalinity on May 5, 1981. This is the only well reported on excursion status that has not shown elevated chloride concentrations. Conductivity was the only parameter elevated significantly, due primarily to a high level of sulfate. Ground water in this well had a two year history of high fluctuating sulfate concentrations prior to the reported excursion (WMC, 1981d). WMC tested the integrity of the casing in well 9M-22, located old exploration holes in the vicinity, and analyzed geologic information to ascertain the cause of the excursion. One abandoned exploration borehole was found adjacent to 9M-22. The hole was redrilled and cemented. Pressure testing the well casing between packers and review of geologic logs of the area did not reveal additional causes of the excursion (WMC, 1981d). Well 9M-22 remained on excursion status after sealing of the exploration hole; therefore, WMC (1981d) believes the incident of high sulfate levels in well 9M-22 is a very localized phenomenon. Sulfate contamination may have been introduced through the nearby abandoned borehole from an overlying sulfate-rich aquifer.

Well 9M-22 is located approximately 120 m (400 ft) west of Production Unit 5. Another Upper Irigaray Sandstone monitoring well identified as 9M-15 is located between well 9M-22 and the well field. Well 9M-15 indicated elevated conductivity and chloride concentrations during January and February, 1981. Although well 9M-15 was never placed on excursion status.

Unit 1 Sand monitoring well SSM-5 in Production Unit 4 was reported on excursion status for chloride on May 29, 1981. The chloride level stabilized above the UCL for about one month and then began to decline. No corrective actions were taken although an investigation into the cause of this excursion was conducted by WMC (1981e). Water quality data suggest that well SSM-5 may not be completed in the Unit 1 Sand or the Coal Unit (WMC, 1981e). A local, discontinuous sandstone unit was discovered in the vicinity of well SSM-5 within the Interburden Unit; this sandstone unit is approximately 1.5 m (5 ft) below the Unit 1 Sand and 0.6 m (2 ft) above the Coal Unit. WMC (1981e) believes that well SSM-5 was drilled and completed accidentally in this sandstone unit below the Unit 1 Sand.

A seven-hour pumping test was conducted in Coal Unit 1 to detect communication between well SSM-5 and the Coal Unit. No drawdown was observed in well SSM-5 during pumping. WMC (1981e) believes that there is communication between well SSM-5 and the Coal Unit, but that seven hours was not long enough to detect the connection. This incident in well SSM-5 suggests that vertical excursions have the potential to reach the Unit 1 Sand due to localized sandstone stringers in the Interburden Unit. No Unit 1 Sand monitoring wells, except well SSM-5, have been reported on excursion status at Irigaray.

Water samples from 12 trend wells completed in the Upper Irigaray Sandstone have exceeded UCL's for two or more excursion indicators between 1980 and 1982. Although trend wells cannot be put on excursion status, they aid in detecting the movement of leaching solutions. The 12 trend wells are located on both the east and west sides of the well field in no regular pattern of distribution. These wells are located adjacent to Production Units 1 through 4 and are about 30 to 60 m (100 to 200 ft) from the well field perimeter. Elevated excursion indicators in the trend wells indicate that leaching solutions have migrated laterally at least 60 m (200 ft) from the well field in many areas, but the lixiviant generally has not moved beyond 120 m (400 ft) from the well field. Only two monitoring wells completed in the Upper Irigaray Sandstone (9M-1 and 9M-22) have been reported on excursion status; these two wells are located 120 m (400 ft) from the well field boundary. The lateral migration of lixiviant away from the well field may be the result of improper quantification of leakage leading to imbalances between injection and pumping rates.

A small-scale pumping program was initiated in Production Unit 1 in January, 1982, to draw back a lixiviant migration detected in several trend wells. The pumping program continued until October, 1982; however, two trend wells (9T-8 and 9T-32) continued to show elevated chloride levels. WMC intended to continue pumping in the vicinity of these trend wells until chloride levels decreased below UCL's.

A.1.8 Restoration

Demonstration restoration projects have been conducted at the 517 test field and Well Field E. Lixiviants used during leaching at the 517 site and Well Field E were ammonium bicarbonate and sodium bicarbonate, respectively. Three restoration techniques were utilized at the 517 sites: 1) ground-water sweeping, 2) clean water recycle, and 3) chemical treatment. Clean water recycle was the primary restoration method used for Well Field E. The reader is referred to WMC (1978b and 1981c) and NRC (1978a) for more detailed discussions of the restoration tests.

Ground-water restoration at the 517 site was conducted from May 1, 1977, to February 18, 1978. Ground-water sweeping consisted of pumping water from the aquifer to create energy gradients toward the pumping well. This resulted in the movement of uncontaminated ground water from beyond the mine area into the leached zone. In the process, residual lixiviant in the leached zone was swept from the aquifer. This sweeping process continued for one month at the 517 site with removal of the equivalent of 12 aquifer pore volumes of ground water. During clean water recycle, ground water was pumped from the leached zone, treated for impurity removal via reverse osmosis, and then reinjected into the aquifer. A total of about five aquifer pore volumes of treated ground water (conductivity of 600-700 umhos/cm) was injected into the aquifer during the one-week clean water recycle period at the 517 site.

A chemical treatment restoration test was conducted at the 517 site to reduce further the levels of residual ammonium (NH_4^+) in the sediments of the aquifer. This chemical test was conducted for three and one-half

months and consisted of the circulation of calcium, sodium, and magnesium ions through the aquifer to replace ammonium adsorbed on the clays. Ammonium was stripped from the recirculation solution through the use of a conventional water treatment plant. The type of treatment equipment used in this facility (e.g., ion exchange, reverse osmosis) is not known. Upon completion of the chemical restoration test at the 517 site, a reverse osmosis unit was added to the circuit. The unit was operated for about two months to reduce further the TDS content of the ground water and the residual ammonium from the sediments of the aquifer.

Concentrations of selected constituents in the ground water before and after the restoration tests at the 517 site are shown on Table A.5. These results indicate that the levels of all constituents were reduced by ground-water sweeping, but not to a significant degree considering the removal of 12 aquifer pore volumes of ground water. Clean water recycle reduced all constituents below the levels achieved by ground-water sweeping with the removal of an additional five aquifer pore volumes of ground water; however, ammonium decreased only 3 mg/L more than during ground-water sweeping.

The chemical treatment restoration test appears not to have accomplished its objective of reducing further the ammonium level significantly. However, during the test the concentration of ammonium in the recovered ground water initially increased from about 120 to 230 mg/L, indicating that residual ammonium was being removed from the sediments of the aquifer. The conductivity increased considerably during chemical treatment, most likely due to the addition of calcium, sodium and magnesium. Recirculating the solution and operating the treatment facility for longer periods of time most likely would have reduced the ammonium level below its final concentration of 120 mg/L. The final two-month period of recirculation of ground water through a reverse osmosis unit reduced the concentrations of all constituents significantly except carbonate. The concentration of ammonium was reduced from 120 to 35 mg/L; however, long-term stability of the reduced ammonium level has not been documented.

Significant amounts of ammonium (NH_4^+) can be adsorbed on calcium clays by ion-exchange (Buma et al, 1980). During restoration, desorption of ammonium from the clays occurs at a relatively slow rate. Ammonium will continue to "bleed" into the ground water for a relatively long period of time. A computer simulation of clean water recycle restoration (Riding and Rosswog, 1979) demonstrated that as many as 96 aquifer pore volumes of ammonium-free water are required to reduce an initial ammonium concentration of 2000 mg/L to a concentration of 0.45 mg/L. Ammonium is not particularly toxic to man, but may be oxidized to nitrate which is potentially carcinogenic to humans (Kasper et al, 1979).

The three restoration tests completed at the Irigaray site indicate that clean water recycle using a reverse osmosis treatment unit probably was the most water efficient method of reducing concentrations of major ions in the ground water to acceptable levels. Use of ammonium-based lixiviants in clay-rich ore zones may require the recirculation of many aquifer pore volumes of water. Chemical treatment with calcium, sodium,

Table A.5. Water quality[a] before and after restoration tests at the 517 well field of the Irigaray site.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Leaching/ Prerestoration[b]	Ground-Water Sweeping[c] 6/01/77	Clean Water Recycle[c] 8/05/77	Chemical Treatment[c] 11/30/77	Final Recycle via RO[c] 2/28/78
Ammonium (NH_4^+)	0.5	<1.0	180	123	120	120	35
Chloride	250	10.75	531	230	160	2800	159
Carbonate	--	--	397	267	40	40	60
Uranium(U_3O_8)	5.0	0.098	18	12	2.0	1.0	<1.0
Conductivity, (μmhos/cm)	--	~2000	3300	1950	1400	9000	880

[a]Units are mg/L except as noted.

[b]Average concentration from wells in the 517 Well Field.

[c]Highest concentration observed from any well in the 517 Well Field.

Source: NRC, 1978a.

and/or magnesium prior to clean water recycle may be beneficial with respect to replacing ammonium in the aquifer sediments, especially in clay-rich zones. However, the introduction of salts during chemical treatment may cause additional problems. The expense of clean water recycle and/or chemical treatment may be substantial, especially if long periods of operation are needed to reduce contaminant concentrations to acceptable levels.

Ground-water restoration within the alternate lixiviant test site (Well Field E) began on March 15, 1980, and was completed on August 27, 1980. The intial operation used a clean water recycle technique where water was pumped from the well field, treated via ion exchange, and reinjected into the ore zone. The clean water recycle technique was effective to the extent that only sodium, uranium, and conductivity (the latter was above EPA standards prior to leaching) remained above drinking water standards by the end of June 1980.

The clean water recycle technique was modified on July 22, 1980, by raising the pH (method unknown) of the recycle water to increase the rate of removal of bicarbonate, uranium, and other radionuclide species from the aquifer. According to WMC (1981c), the increase in injection pH accelerated the removal of residual uranium from the aquifer significantly. Table A.6 presents the concentrations of major constituents in the ground water before, during, and after restoration at Well Field E.

Eleven and one-half aquifer pore volumes of ground water were removed from the leached zone during five and one-half months of restoration within Well Field E. Generally, ground water within Well Field E had been restored to the ground-water quality standards of the State of Wyoming, the EPA, or near baseline values; however, elevated pH values were detected during post-restoration monitoring in Well Field E. WMC initiated recirculation of the well field on April 10, 1981, to reduce the pH of the ground water; recirculation lasted about two weeks and was effective in reducing the pH to the baseline value.

Water quality data for major dissolved constituents in Well Field E indicate that restoration was successful in reducing major constituents below drinking water standards (Table A.6). However, restoration was not able to reduce the major dissolved constituents below premining baseline levels. Concentrations of the major constituents have declined significantly after about 15 months of post-restoration monitoring. Water quality data indicate that injection of high pH (9.0- 11.0) water accelerated the removal of residual uranium and bicarbonate and reduced the average concentration of sodium from about 230 mg/L to the drinking water standard of 200 mg/L.

Use of a reverse osmosis unit, or a combination of methods instead of ion exchange, might have increased the rate and improved the success of the restoration process; only ion exchange was used to treat recirculated water in Well Field E. Restoration of Well Field E in which a lixiviant of sodium bicarbonate was used appears to have been more efficient and encountered less difficulties than restoration of the 517 site, rendering

Table A.6. Water quality[a] for the well field recovery solutions before, during, and after restoration at Well Field E of the Irigaray site.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Leaching/ Prerestoration[c] 3/11/80	During Restoration[c] 6/26/80	Termination of Restoration[c] 8/27/80	Post Restoration[c] 11/12/81
Chloride	250	12	210	116	98	39
Bicarbonate	500	90	1670	435	241	196
Sodium	200	120	787	230	200	177
Uranium (U_3O_8)	5.0	0.03	40	7.5	3.1	1.9
Conductivity (μmhos/cm)	--	650	3070	1148	1044	870

[a]Units are mg/L except as noted.

[b]Complete chemical analyses were not performed on the recovery solution samples.

[c]Average concentration from wells in Well Field E.

Source: WMC, 1981c.

any comparisons inconclusive. Sodium-based lixiviants have a tendency to cause swelling of clays and a resulting decrease in aquifer hydraulic conductivity. This swelling has not been a significant problem at Irigaray.

Restoration has not been initiated in any of the production units that have been mined (Production Units 1 through 5) as of June, 1984; restoration is not planned by WMC while the site is on standby status. WMC's proposed restoration program for the production units would incorporate ground-water sweeping and clean water recycle (NRC, 1978a). The water would be adjusted chemically by the addition of ions of calcium or magnesium prior to injection into the mined aquifer to enhance the removal of ammonium adsorbed on clays. When the concentration of ammonium in the ground water increased to a predetermined level, a reverse osmosis unit would be added to the circuit and operated until all ions were reduced to acceptable levels. WMC (1981a) estimates that the withdrawal of at least five aquifer pore volumes would be required to attain acceptable water quality levels.

A.2 OGLE PETROLEUM, INC. - BISON BASIN PROJECT

A.2.1 Description of the Site

The Bison Basin Project is operated by Ogle Petroleum, Inc. (OPI) in Fremont County, south-central Wyoming. The uranium in situ mine site is located approximately 80 km (50 mi) south of Riverton, Wyoming and 48 km (30 mi) southwest of Jeffrey City, Wyoming (Fig. A.10). Gently rolling terrain characterizes the project area; elevations range between 2130 and 2200 m (6990 and 7220 ft). The mine site is located on the southeast flank of the Wind River uplift, within the West Alkali Creek drainage basin (Fig. A.11). West Alkali Creek flows only in response to snowmelt and precipitation events. The mine site area is drained by several small ephemeral channels which discharge either into Grassy Lake (a small closed depression) or into West Alkali Creek. Alkali Creek, located approximately 15 km (9 mi) from the Bison Basin Project site, is the nearest perennial stream. Annual precipitation at the site ranges between 30 and 41 cm (12 and 16 in).

Solution mining research and development at the site began in 1979; this effort was followed in 1981 by commercial mining of Mine Unit 1 (Fig. A.12). OPI intends to mine Units 2 through 4; however, only Mine Unit 1 had been mined as of June, 1984. These commercial mine units are located in Section 25, T27N, R97W, and in Section 30, T27N, R96W (Fig. A.12). Restoration of the research and development site was conducted from August 1 through September 14, 1979.

A.2.2 History of Mining Operations

Research and development testing was conducted from May 1 to July 31, 1979, on a one-acre tract within Mining Unit 1 (Fig. A.13). A sodium carbonate/bicarbonate lixiviant with oxygen or hydrogen peroxide oxidant was used (NRC, 1981a). The test unit consisted of a row of four injection

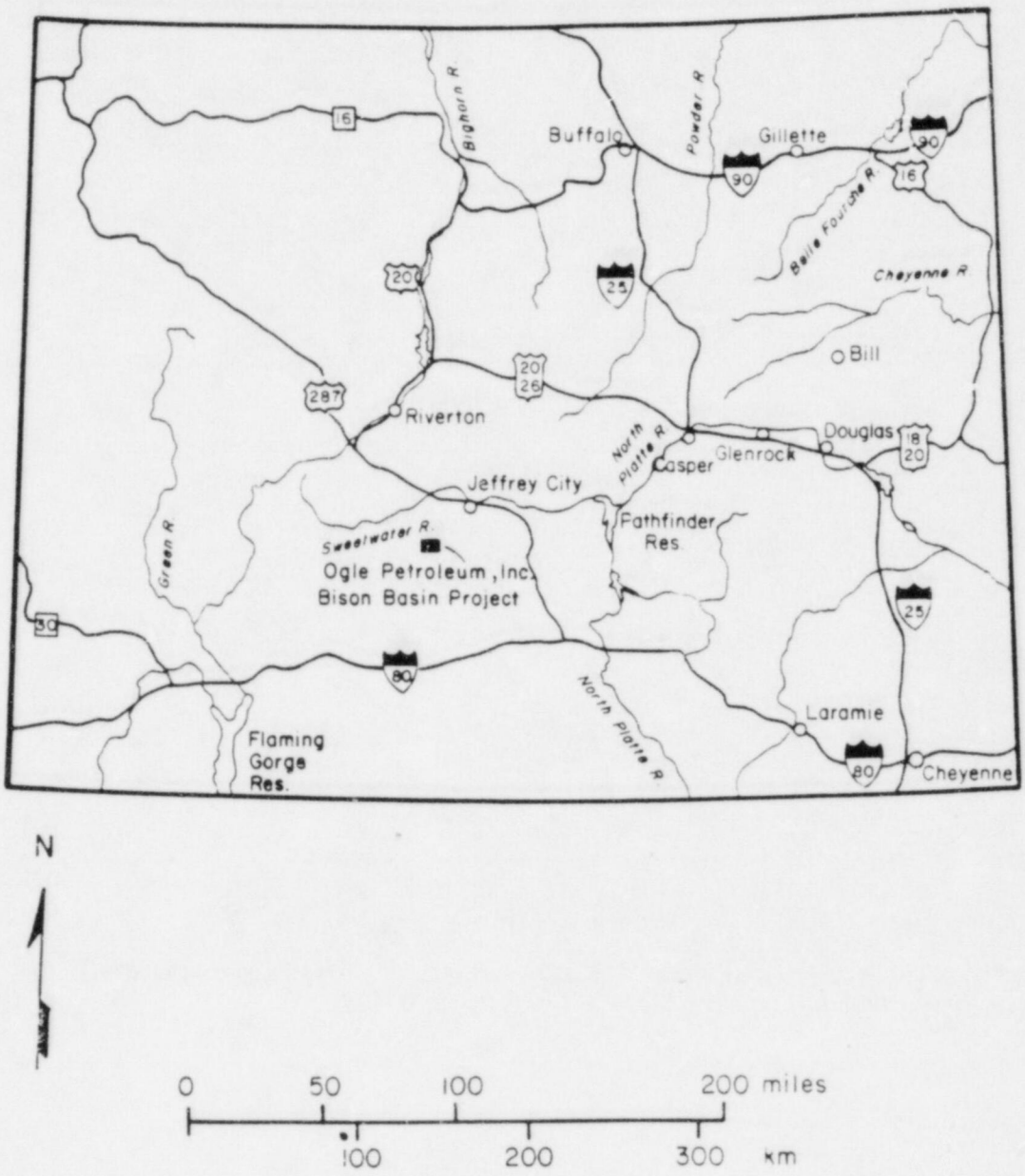


Fig. A.10. Location map of the Ogle Petroleum, Inc., Bison Basin mine.

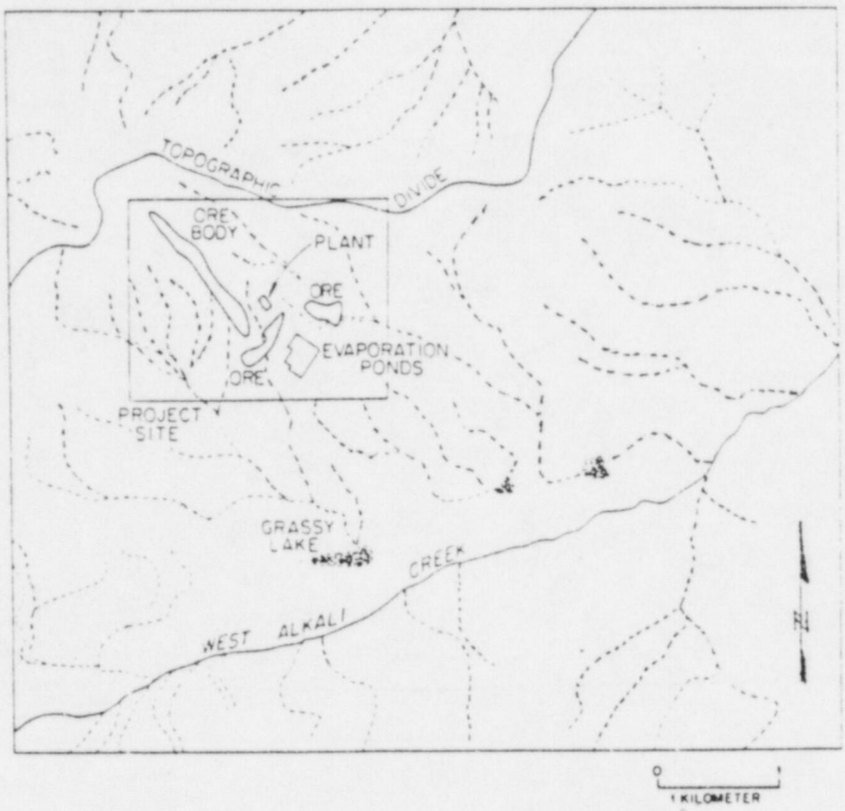


Fig. A.11. Drainage patterns and their relationship to the Bison Basin project site. Dashed lines are ephemeral drainages. *Source:* Modified from NRC, 1981a.

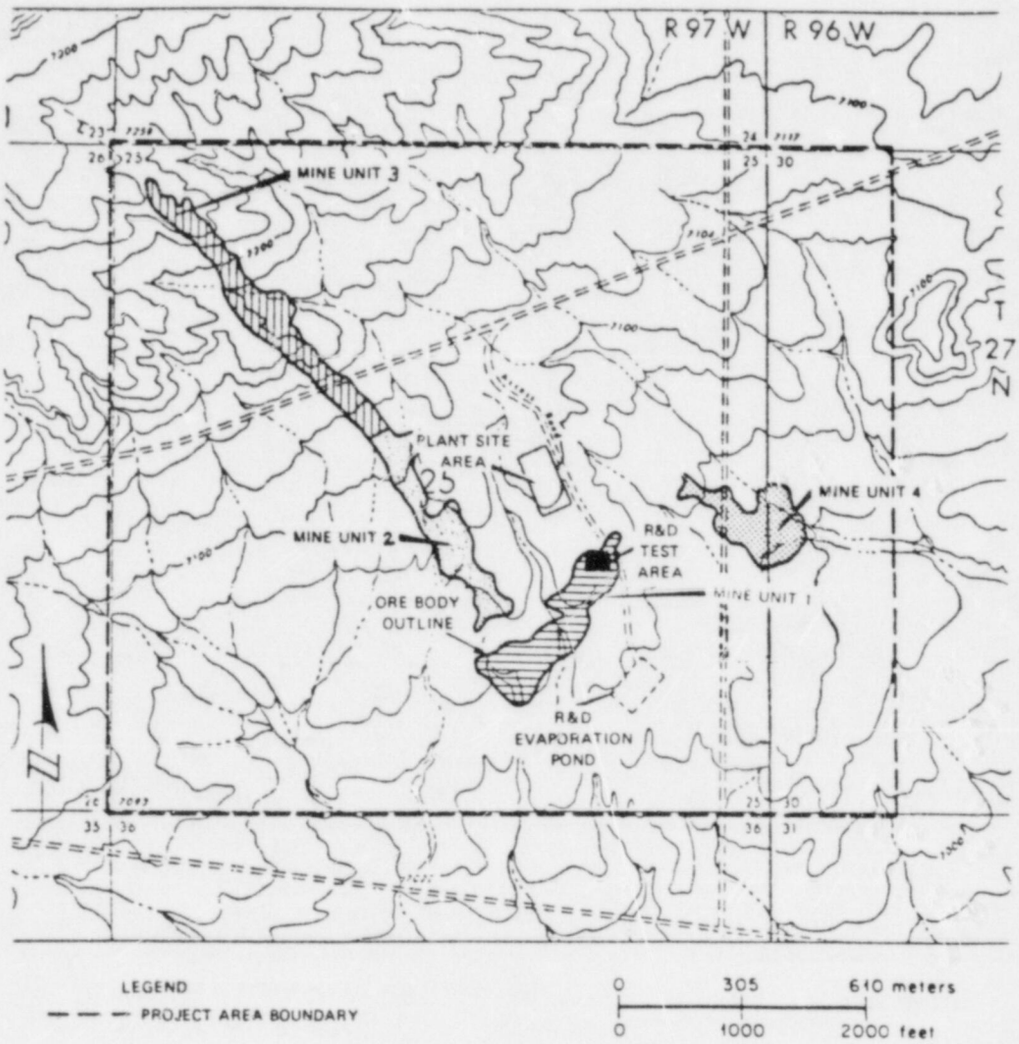


Fig. A.12. Topographical map of the Bison Basin project site. *Source:* Modified from NRC, 1981a.

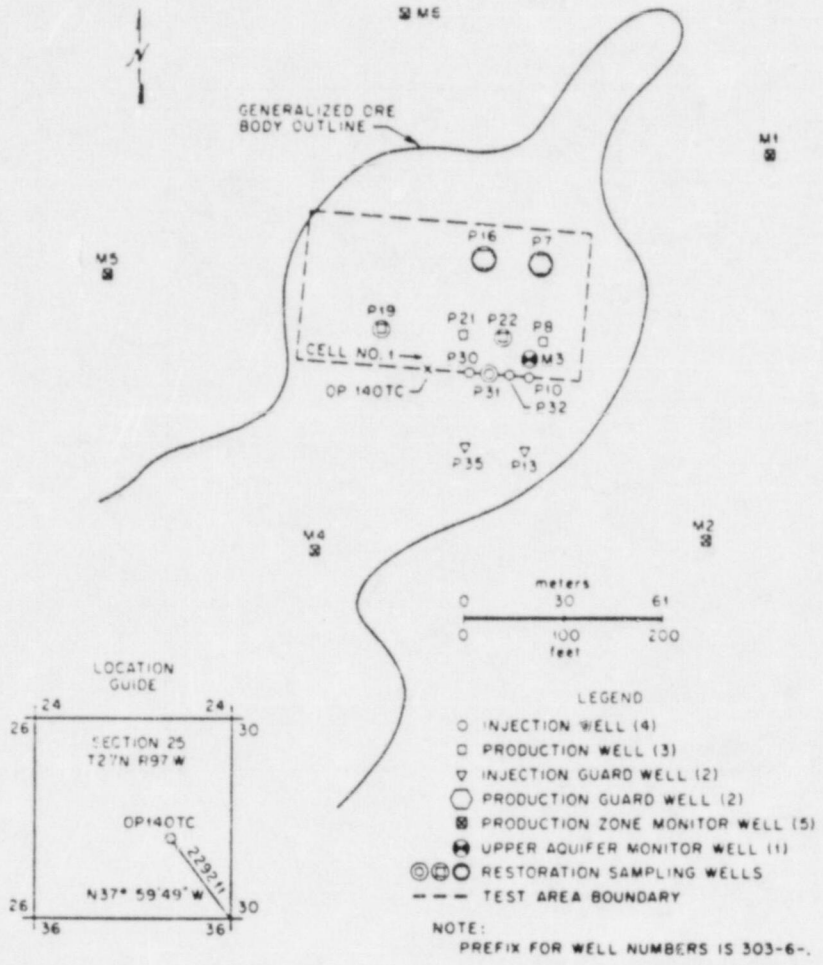


Fig. A.13. Research and development well field at the Bison Basin site. *Source:* NRC, 1981a.

wells separated by a distance of 6 m (20 ft) and a parallel row of three production wells separated by 12 m (40 ft); a distance of 11 m (36 ft) separates the injection and production rows. According to the NRC (1981a), this well pattern is used adjacent to faults. About 25 aquifer pore volumes were circulated through the production zone during the three month pilot testing period.

The commercial well field configuration consists of four mine units (Fig. A.12) covering about 16 ha (40 acres). Commercial production in Mine Unit 1 began on August 21, 1981; the mine was closed temporarily due to economic considerations as of June, 1984. OPI had planned to proceed with commercial-scale mining in Mine Units 2, 3, and 4.

The lixiviant used or to be used for all mine units is a sodium carbonate/bicarbonate solution in water at a concentration of 1 to 3 g/L. An oxidant of oxygen or hydrogen peroxide (concentration unknown) has been used as part of the lixiviant (OPI, 1979). OPI utilizes a seven-spot well pattern comprised of six injection wells surrounding one central recovery well. The distance between wells is 15 m (50 ft). Average pumping rate per production well varies between 0.6 and 0.8 L/s (9 and 12 gal/min). Injection rates per well vary between 0.3 and 0.4 L/s (4 and 7 gal/min) at pressures from 4.1×10^5 to 6.9×10^5 N/m^2 (60 to 100 psi). This pressure range will not exceed a pressure of 1.4×10^4 N/m^2/m (0.63 psi/ft) of well depth, which is the minimum pressure for the initiation of hydraulic fracturing.

The uranium-bearing solution is pumped from the well field to ion exchange resin columns where the uranium is adsorbed preferentially from the solution while chloride ions are released into the solution. Carbon dioxide and/or sodium carbonate are added as necessary to restore the lixiviant to its original strength. The solution is returned to the well field where the oxidant is added immediately prior to injection into the ore zone.

A.2.3 Regional Setting

A.2.3.1 Regional Geology

The Bison Basin Project site is located in the Bison Basin (also known as the Cyclone Basin) area on the southeast flank of the Wind River uplift. The mine site is about 6 km (4 mi) north of the continental divide, which separates the Great Divide Basin and the drainage of West Alkali Creek. Basins in the region generally contain Tertiary sediments of Eocene or Paleocene age. The mountain ranges contain cores of Precambrian granite or metasedimentary strata (NRC, 1981a). Erosion of these mountains provided most of the source material for the Paleocene and Eocene formations. Coarse clastics and feldspar-quartz sands were deposited as alluvial fans and fluvial plains. Finer silt and clay size materials are found in strata deposited in lacustrine environments.

The ore zone (basal sandstone of the "D" unit) is within the Laney Member of the Green River Formation of Eocene age. This formation consists

of alternating strata of sandstone, siltstone, and mudstone. The Laney Member is part of a large system of channel sandstone deposits that coalesce a few kilometers east of the project area (NRC, 1981a). The Laney Member was deposited in a major drainage that transported oxidizing uranium-charged water into the Great Divide Basin (NRC, 1981a). Folds and faults of middle-upper Tertiary age are common throughout the area; they generally trend in an east-west direction.

A.2.3.2 Regional Ground-water Flow Systems

The regional ground-water flow systems in this part of Wyoming are poorly defined. Welder and McGreevy (1966) indicate that both unconfined and confined aquifer systems are present in the Great Divide Basin located just south of the project area. Aquifers of large areal extent having uniform characteristics probably are not present. Structural features in the region influence regional ground-water flow systems. Regional recharge areas for the basins probably are the Wind River and Granite Mountains; these ranges are located northwest and east of Bison Basin, respectively. Localized ground-water basins are formed with recharge and discharge areas controlled by folds and faults.

Limited livestock grazing in the area has not placed a demand on the use of ground-water supplies. Three water wells are located in the four townships that include and surround the project area. One of these wells has been purchased by OPI for use in drilling operations (OPI, 1979). The other two wells, used for stock watering, are located more than 13 km (8 mi) south of the project area in the Great Divide Basin. The nearest public ground-water supply is located in Jeffrey City, approximately 48 km (30 mi) northeast of the project area. OPI has completed an industrial water well to a depth of 187 m (614 ft) in the Battle Springs Formation.

A.2.4 Description and Analysis of Data Collected

A.2.4.1 Exploration

Exploration boreholes were drilled throughout the Bison Basin mine permit area and adjacent sections for evaluation of uranium values in several formations. OPI and its predecessors drilled 820 exploration holes at the project area between 1967 and 1980.

Exploration holes drilled by OPI prior to 1978 (total of 136) were abandoned by leaving a column of drilling mud in the hole and installing a surface cap. The remainder of the holes (320) drilled by OPI were plugged by adding 23 kg (50 lbs) of Shur-gel (Trade name) to the drilling mud per 30 m (100 ft) of hole. Each hole was capped by pouring cement on top of the Shur-gel column. These procedures should provide adequate sealing of their exploration holes. OPI (1981) indicates that the 364 exploration holes drilled by other companies are believed to have been plugged and capped in a suitable manner which included leaving a column of drilling mud in the hole upon abandonment. However, some of these boreholes may not have been plugged or were improperly plugged. Rigorous records of plugging by other companies are not available.

Very little information is available concerning geophysical logging at the Bison Basin site. OPI (1979) states that over 400 exploration drill holes were logged geophysically; however, only seven logs are available to the authors for review. Apparently other logs exist but have not been placed in the public record. Geophysical logs for the seven holes consist of gamma, spontaneous potential, and resistivity. All logs were recorded prior to placement of casing in the holes.

Eight cross sections were constructed and presented by OPI (1979). Identification of lithologic units using the geophysical logs was based on the shape and magnitude of log response in the vicinity of a particular rock type. Alternating sequences of sandstones and claystones exhibit unique signatures and can be differentiated using the geophysical logs. The uranium-bearing zone is identified easily because it causes a distinct deflection in the gamma log response.

A.2.4.2 Aquifer Tests

OPI conducted ten aquifer tests between 1977 and 1982 at their Bison Basin mine site. Seven of these aquifer tests were conducted in Mine Unit 1. One test was conducted in Mine Unit 2 and two tests were conducted in Mine Unit 3. Eight constant discharge pumping tests were conducted using two or more observation wells; recovery was measured during four of these pumping tests. Two constant rate injection tests were conducted at the mine site. Table A.7 summarizes the aquifer tests conducted at the Bison Basin mine site. Locations of the aquifer tests are shown on Fig. A.14.

Table A.8 summarizes the aquifer and aquitard characteristics estimated by OPI, by Hydro-Engineering, Inc., and the authors from the aquifer test data. Methods used to analyze the aquifer test data also are included in Table A.8. The Hantush and Jacob (1955) leaky artesian method was used by the authors to analyze data from aquifer tests 1 and 2. However, there is some uncertainty with the match of the data curves to the Hantush and Jacob (1955) solution because drawdown data for these two tests were not recorded during the first 15 minutes of pumping. The Hantush (1960) modified leaky artesian method was used by the authors to analyze data from aquifer tests 3 and 5. In this case, the drawdown data curves were matched to the Hantush (1960) solution with greater certainty than for tests 1 and 2 because only drawdown data during the first 5 minutes of pumping were not available. Both leaky artesian methods of analysis were utilized by the authors to analyze data from the observation wells for aquifer test 4. Aquifer test 4 was conducted for a period of only five hours. Longer pumping/injection periods for most of the aquifer tests conducted at the Bison Basin mine would have allowed considerably improved and more reliable curve matching results. OPI (1979) concluded that no leakage was occurring based on the results of aquifer tests 3, 4, and 5.

Table A.7. Summary of aquifer tests conducted at the Bison Basin mine.

Test No.	Pumping/ Injection Well	Production Unit or Well Field	Aquifer[a]	Total Depth of Tested Well m (ft)	Number of Observation Wells	Units Monitored[a]	Type of Test[b]	Date of Test	Length of Tests (hrs)	Discharge Rate L/s (gal/min)
1	OP-140-TC	1	D-Sand	?	4	D-Sand	CD	6/13/77	11.5	0.53 (8.4)
2	OP-140-TC	1	D-Sand	?	4	D-Sand	I	6/08/77	11.5	0.59 (9.4)
3	OP-141-TC	3	D-Sand	?	4	D-Sand	CD R	6/22/77	10 ?	0.64 (10.2) --
4	OP-141-TC	3	D-Sand	?	4	D-Sand	I R	6/17/77	5 ?	0.98 (15.5) --
5	303-6-P22	1	D-Sand	?	16	D-Sand, B-Sand	CD	11/03/78	25	0.5 (8)
6	M-19	1	L-Sand	?	3	B-Sand, D-Sand	CD	11/03/80	10	0.25 to 0.57 (3.8 to 9.0)
7	M-63	1	L-Sand	120 (394)	3	B-Sand, D-Sand	CD	11/14/80	0.5	0.28 to 0.8 (3 to 12)
8	M-4	1	D-Sand	?	2	B-Sand, L-Sand	CD R	4/17/81	10.3 27	0.38 (6.1) --
9	A-B	1	D-Sand	?	3	B-Sand, L-Sand	CD R	4/28/81	11 32	0.38 to 0.45 (6.1 to 7.2) --
10	C-70	2	D-Sand	105 (344)	17	D-Sand, B-Sand L-Sand	CD R	5/24/82	74.2 79	0.32 to 0.45 (5.1 to 5.6) --

[a]D-Sand is the uranium-bearing sandstone of the "D" unit; B-Sand is the basal sandstone of the "B" unit; L-Sand is the lower sandstone monitor zone in the sandstone stringer below the ore zone.

[b]CD = constant discharge pumping test; I = constant rate injection test; R = recovery portion of test.

[c]Pumping or injection rate.

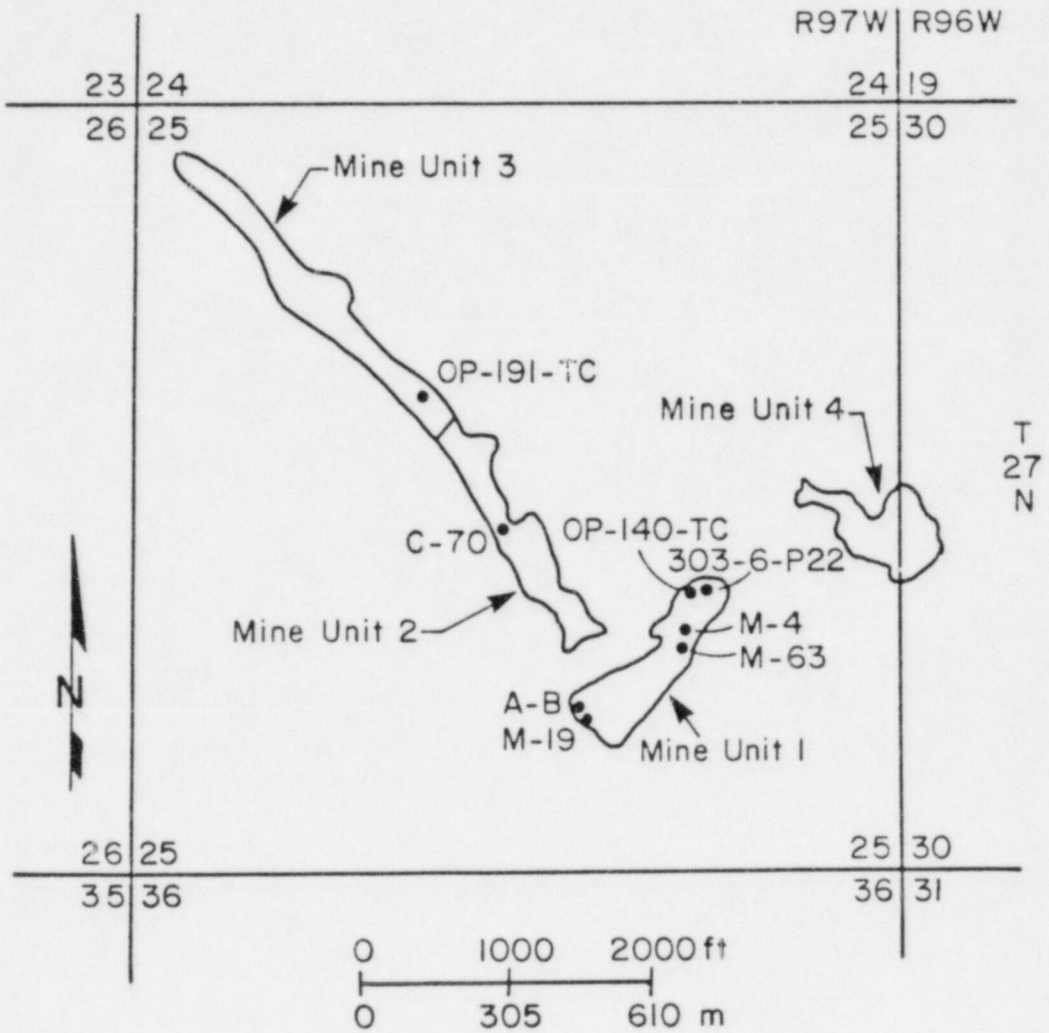


Fig. A.14. Location map of pumping/injection wells for the aquifer tests conducted at the Bison Basin mine site. *Source:* Modified from NRC, 1981a.

Table A.8. Summary of aquifer and aquitard characteristics estimated by Ogle Petroleum, Inc., Hydro-Engineering, and the authors from aquifer test data for the Bison Basin mine.

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1	OP-140-TC	D-Sand	OPI	$6x10^{-5}$ to $4x10^{-4}$ (0.2 to 1.1)	0.3 to 2 (3 to 23)	$1.5x10^{-4}$	--	CJ,HJ	CD
			Authors	$7x10^{-5}$ to $1x10^{-4}$ (0.2 to 0.3)	0.4 to 0.5 (4 to 5)	$9x10^{-5}$	$7x10^{-6}$ to $7x10^{-5}$ ($2x10^{-2}$ to $2x10^{-1}$)	HJ	CD
2	OP-140-TC	D-Sand	OPI	$1x10^{-4}$ to $3x10^{-4}$ (0.3 to 0.9)	0.6 to 2 (6 to 23)	$1x10^{-3}$	--	CJ,HJ	I
			Authors	$2x10^{-4}$ to $4x10^{-4}$ (0.5 to 1.1)	0.7 to 2.0 (7 to 23)	$9x10^{-5}$	$1x10^{-5}$ to $6x10^{-5}$ ($3x10^{-2}$ to $2x10^{-1}$)	HJ	I
3	OP-141-TC	D-Sand	OPI	$4x10^{-4}$ to $7x10^{-4}$ (1.1 to 2.0)	2 to 4 (23 to 44)	$4.8x10^{-5}$	--	CJ,HJ	CD
			Authors	$3x10^{-4}$ to $6x10^{-4}$ (0.9 to 1.7)	1 to 3 (12 to 32)	$5x10^{-5}$	$1x10^{-8}$ to $8x10^{-8}$ ($4x10^{-5}$ to $2x10^{-4}$)	H	CD
			OPI	$4x10^{-4}$ to $6x10^{-4}$ (1.1 to 1.7)	2 to 3 (23 to 32)	--	--	CJ	R
4	OP-141-TC	D-Sand	OPI	$4x10^{-4}$ to $5x10^{-4}$ (1.1 to 1.5)	2 to 3 (20 to 27)	$3x10^{-3}$	--	CJ,HJ	I
			Authors	$4x10^{-4}$ to $6x10^{-4}$ (1.1 to 1.7)	1 to 2 (15 to 24)	$4x10^{-4}$	$1x10^{-8}$ to $2x10^{-7}$ ($4x10^{-5}$ to $2x10^{-4}$)	HJ,H	I
			OPI	$4x10^{-4}$ to $5x10^{-4}$ (1.3 to 1.5)	2 to 3 (23 to 32)	--	--	CJ	R

Table A.8. (Continued)

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
5	303-6-P22	D-Sand	OPI	$2x10^{-4}$ to $3x10^{-4}$[f] (0.7 to 0.9)	1 to 2[f] (15 to 24)	$3x10^{-4}$[f]	--	CJ	CD
			Authors	$3x10^{-4}$ to $5x10^{-4}$ (0.9 to 1.5)	1 to 2 (15 to 24)	$2x10^{-4}$	$2x10^{-9}$ to $2x10^{-8}$ ($6x10^{-6}$ to $5x10^{-5}$)	H	CD
8	M-4	D-Sand	Authors	--	0.3 (3.2)	--	--	CJ	R
9	A-B	D-Sand	Authors	--	0.2 (2.2)	--	--	CJ	R
10	C-70	D-Sand	HE	$5x10^{-4}$ to $8x10^{-4}$ (1.5 to 2.2)	1 to 3 (15 to 32)	$9x10^{-5}$	$1x10^{-6}$ ($3x10^{-3}$)[f]	T,CJ,P,NW	CD
			HE	--	1 to 3) (20 to 25)	--	--	CJ	R

[a]D-Sand is the uranium-bearing basal sandstone of the "D" unit.

[b]Ogle = Ogle Petroleum, Inc.; HE = Hydro-Engineering.

[c]Values are based, in part, on the following assumptions: 1) thickness of the basal sandstone of the "D" unit is 25 ft; 2) thickness of the aquitard overlying the "D" sandstone is 100 ft, 3) the aquitard underlying the "D" sandstone is impermeable, and 4) the specific storage of the overlying aquitard is two orders of magnitude larger than that calculated for the tested aquifer.

[d]CJ = Cooper and Jacob (1946); HJ = Hantush and Jacob (1955); H = Hantush (1960); T = Theis (1935); P = Papadopulos (1965); NW = Neuman and Witherspoon (1972).

[e]CD = Constant discharge pumping test; I = Constant rate injection test; R = Recovery test.

[f]Results are questionable or not valid because method(s) of analysis were used improperly.

Source: OPI, 1979a and Hydro-Engineering, 1982.

The Hantush and Jacob (1955) method was used by OPI to analyze drawdown data from aquifer tests 1 through 4; but, vertical hydraulic conductivity of the confining layer(s) was not estimated. Hydro-Engineering, Inc. (1982) attempted to analyze the results of aquifer test 10 by using the "ratio method" described by Neuman and Witherspoon (1972); however, the aquitard observation well may not have been ideally located.

Hydrogeologic properties of the "D" sandstone estimated by OPI (1979) from aquifer test 5 generally may be in question because the data record was of insufficient length to apply the straight line method of Cooper and Jacob (1946). The variable pumping rate [0.57 to 0.25 L/s (9.0 to 3.8 gal/min)] instead of constant pumping during aquifer test 6 rendered the results questionable. Aquifer test 7 was discontinued after 33 minutes of pumping because the well would not yield a sufficient quantity of water; no indication was presented on what the problem may have been. The authors used recovery data from the pumping wells for aquifer tests 8 and 9 to estimate transmissivity by the Jacob straight line method; drawdown data for the pumping wells were not reported.

Values for transmissivity and storativity estimated from the drawdown data for aquifer test 10 by Hydro-Engineering, Inc. (1982) using the Cooper and Jacob straight line method of analysis are questionable because all of the criteria for applying the method were not satisfied. Actual hydrogeologic properties of the basal sandstone of the "D" unit in Mine Unit 2 probably fall within the range of values that were estimated using the Theis (1935) method of analysis modified to consider image wells (boundaries) (Hydro-Engineering, Inc., 1982). Very little can be determined with respect to potential leakage between aquifers, and properties of aquitards, with the data available from test 10. Boundary effects imposed by the north and middle faults in Mine Unit 2, and recovering water levels in some observation wells from previous pumping in the aquifer, prevent meaningful analysis of these data. The pumping test indicates that both faults are flow retarding, but not impermeable, boundaries. Consequently, pumping in Mine Unit 1 lowered water levels in Mine Unit 2.

Water levels in wells completed in aquifers overlying and underlying the pumped/injected aquifer declined up to 0.1 m (0.3 ft) during the aquifer tests, except for test 10. In the latter case, a rising water level trend was observed in the overlying and underlying aquifers before, during, and after the test, which suggests that there was no response in these aquifers to pump test 10. However, these results alone do not indicate the absence of leakage. Small water level declines cannot be used to support conclusions regarding leakage because antecedent water levels and/or barometric pressure changes were not measured or reported during most of the aquifer tests.

The basal sandstone of the "D" unit (ore zone) is the only aquifer at the Bison Basin mine site that has been characterized from results of the aquifer tests. The horizontal hydraulic conductivity of this unit ranges

from about $6x10^{-5}$ to $8x10^{-4}$ cm/s (0.2 to 2.3 ft/d). The average storativity is about $1x10^{-4}$. The axis of maximum transmissivity was calculated from drawdown data from aquifer test 10 to be N17°W, which is nearly parallel to the general trend of the ore body in Mine Unit 2 (Hydro-Engineering, Inc., 1982). The basal sandstone of the "D" unit appears to be anisotropic and heterogeneous in nature based on the aquifer tests conducted at the Bison Basin mine.

Results of aquifer tests 1 through 5 (Table A.8) conducted in the basal sandstone of the "D" unit indicate that water leaks into this unit under pumping conditions. Data curves matched by the authors to the Hantush and Jacob (1955) leaky artesian type curves (tests 1, 2, and 4) indicate a range of vertical hydraulic conductivity from $2x10^{-6}$ to $7x10^{-5}$ cm/s ($5x10^{-3}$ to $2x10^{-1}$ ft/d). Data curves matched by the authors to the Hantush (1960) modified leaky artesian type curves (tests 3, 4, and 5) indicate a range of vertical hydraulic conductivity from $2x10^{-9}$ to $2x10^{-7}$ cm/s ($6x10^{-6}$ to $5x10^{-4}$ ft/d). These values of vertical hydraulic conductivity estimated using both leaky artesian methods of analysis correspond to the aquitard overlying the basal sandstone of the "D" unit, assuming that the underlying aquitard was impermeable for the purpose of the calculations only.

Aquifer test data that match the Hantush (1960) type curves suggest that most of the leakage water may be derived from storage in the overlying aquitard (also assuming that the underlying aquitard is impermeable). This source is to be expected because the aquitard overlying the basal sandstone of the "D" unit is about 30 m (100 ft) thick. Aquifer test data curves that match the Hantush and Jacob (1955) type curves suggest that water may also be leaking through the aquitard from the overlying aquifer. This aquitard contains siltstone layers and sandstone lenses, which would be expected to transmit water in the vertical direction. However, an adequate number of geophysical and lithological logs are not available to identify specific areas where the aquitard overlying the basal sandstone of the "D" unit is thin or contains numerous sandstone lenses. Abandoned boreholes and/or damaged wells also may contribute to vertical leakage from an overlying aquifer but there is no evidence from observation wells to suggest such conditions exist. However, results from testing and observations were inconclusive.

A.2.5 Hydrogeology of the Mine Site

The uranium ore body being mined at the Bison Basin mine is the basal sandstone of the "D" unit. Average thickness of the basal sandstone of the "D" unit is about 4.6 m (15 ft) at a depth of about 116 m (380 ft) below the land surface; the ore thickness within the basal sandstone averages about 1.9 m (6.3 ft) (NRC, 1981a). Uranium was precipitated in roll-front deposits from ground water that encountered carbonaceous organic matter and metallic pyrite; these reductants are estimated to be 1.5% of the volume of the basal sandstone of the "D" unit (NRC, 1981a).

The sandstones, siltstones, and mudstones of the Laney Member are part of a large system of fluvial deposits formed by meandering and braided

river systems in the Bison Basin (NRC, 1981a). Horizontal and vertical facies change occur over short distances. The existence of occasional fresh water limestone lenses at the mine site indicates localized stagnant water conditions during deposition. The Battle Springs Formation underlying the Laney Member constitutes a large alluvial fan complex (OPI, 1979). The Bison Basin was subjected to large sediment accumulations from large alluvial fans and braided streams on the southeast flank of the Wind River uplift.

The stratigraphic column for the project area is presented in Fig. A.15. Structural features mapped by Denson and Pipiringos (1974) in the Bison Basin project vicinity include the McKay Lake-Daley Lake synclinal basins bounded on the north by the Horsetrack anticline and on the south by the Mesa anticline. Fig. A.16 is a plan view of the project area showing exposed bedrock formations and structural features. The permit area is located on the north side of the McKay Lake-Daley Lake syncline basin. The "D" sandstone dips gently southeastward at 34 m/km (180 ft/mile); it crops out around the Horsetrack anticline and West Alkali Creek. A generalized cross-section of the project area is presented in Fig. A.17. Several normal faults with small but variable displacements transect the 16 ha (40 acre) ore body (Fig. A.18), and generally extend east to west with no apparent surface expression. The maximum displacement at depth determined from geophysical and lithological logs is 15 m (50 ft) (OPI, 1979).

The McKay Lake-Daley Lake syncline (Fig. A.16) locally controls the direction of ground-water flow to the depth of the basal sandstone of the "D" unit. Ground-water flow within this syncline moves at a slow rate to the southeast toward the axis. Ground water in the basal sandstone of the "D" unit is moving under a hydraulic gradient of about 0.003 (NRC, 1981a).

The water table in the project area occurs at a depth of 15 to 20 m (50 to 65 ft) below ground surface in the Bridger Formation. The confined aquifers that occur below the Bridger Formation in descending order are: basal sandstone of the "B" unit, basal sandstone of the "D" unit, and Battle Springs Formation (Fig. A.15). The sandstone aquifers are separated by mudstones, siltstones, and sandstone stringers.

Premining water levels in aquifers at the Bison Basin site decreased with depth. This indicates that the vertical gradient was directed downward from the basal sandstone of the "B" unit through the basal sandstone of the "D" unit. The static head in the basal sandstone of the "B" unit was approximately 12 m (40 ft) higher than that of the basal sandstone of the "D" unit and about 18 m (60 ft) higher than the static head of a sandstone zone below the basal sandstone of the "D" unit.

A.2.6 Monitoring Well System and Well Construction

Monitoring wells at the Bison Basin site are sampled every two weeks for conductivity, carbonate plus bicarbonate, chloride, uranium, sodium, and sulfate (pilot and commercial-scale operations). These constituents are considered primary indicators for the detection of excursions. They were selected because 1) sodium carbonate/bicarbonate is the leaching

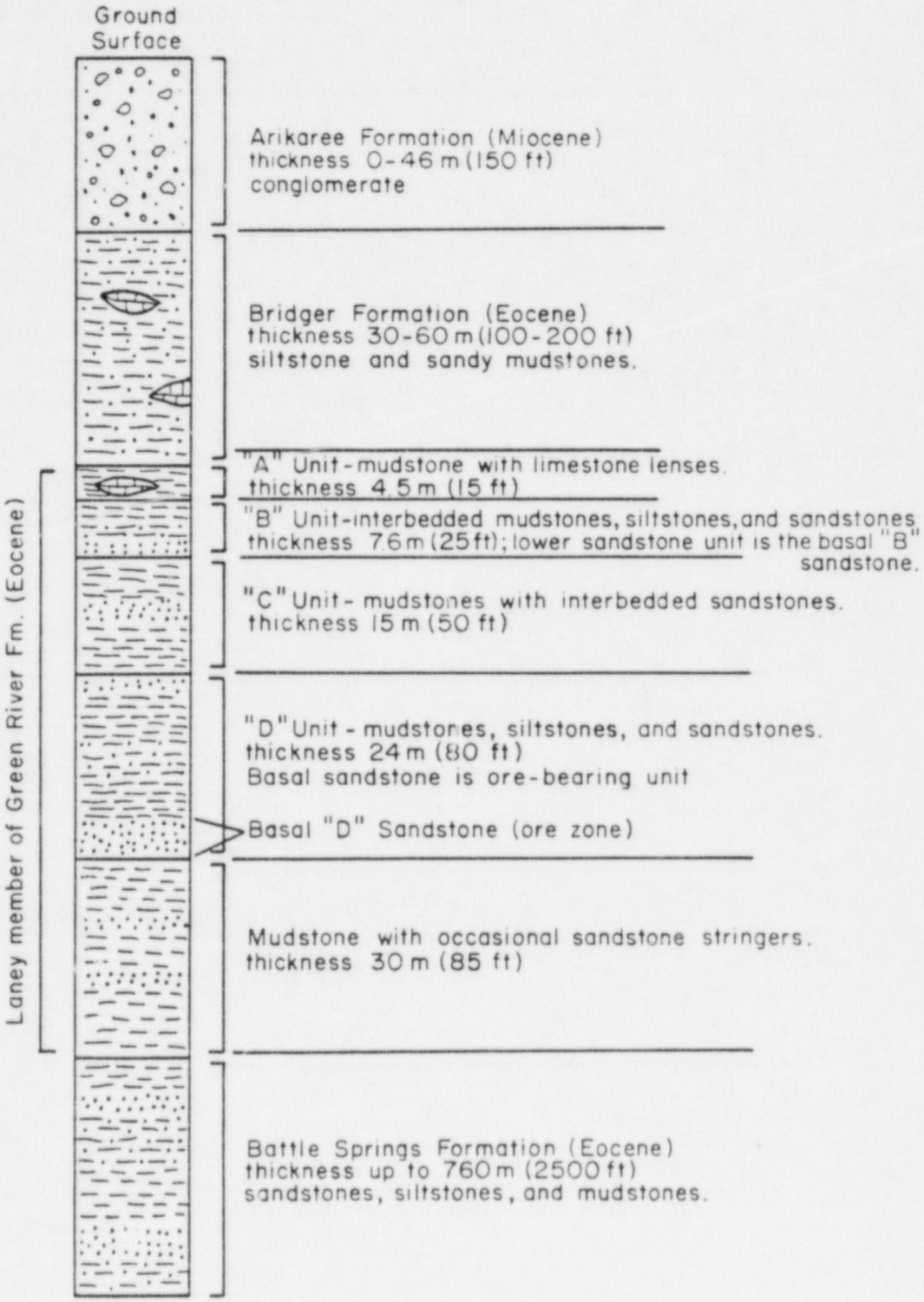


Fig. A.15. Stratigraphic section for the Bison Basin project area. *Source:* Modified from OPI, 1979a.

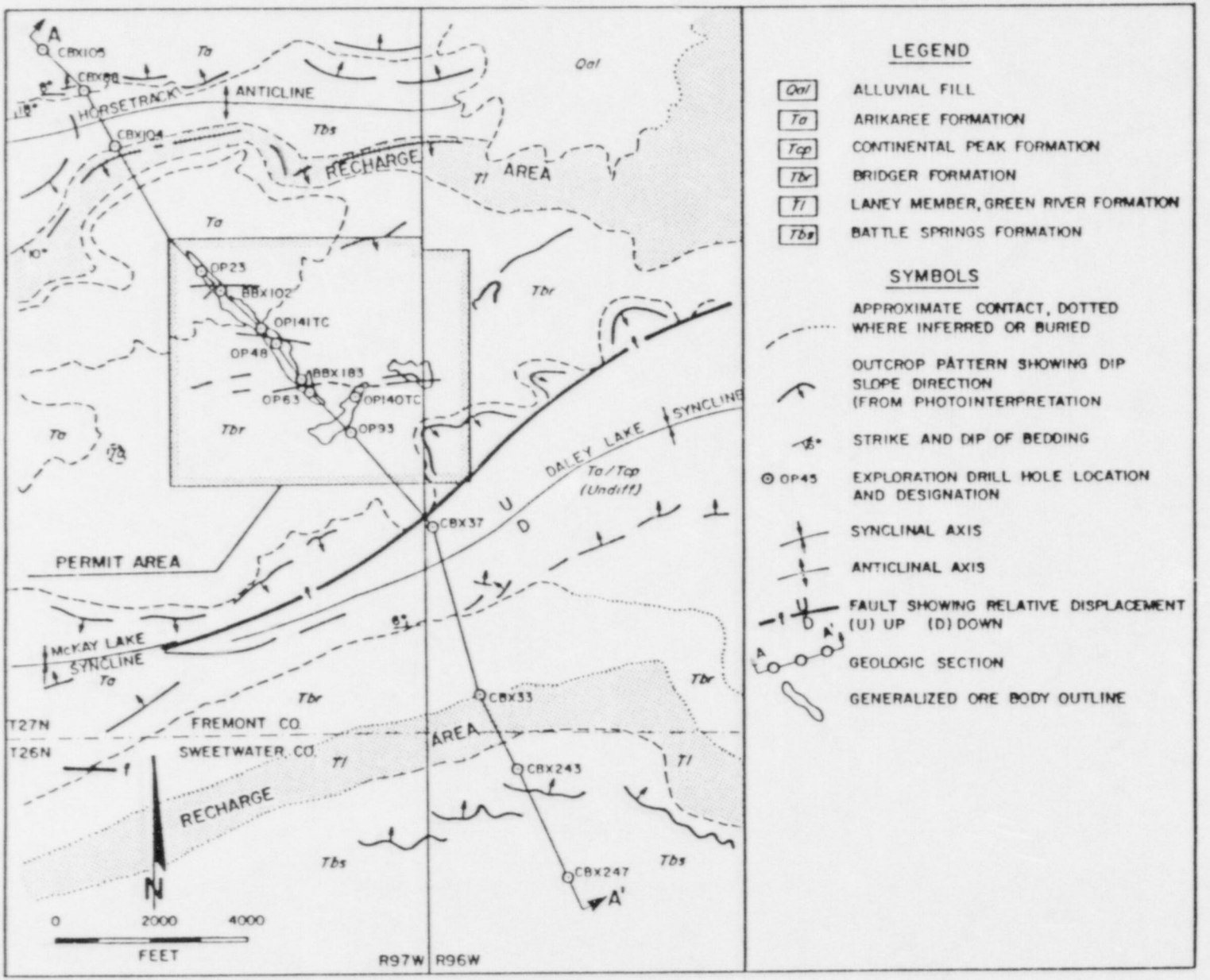
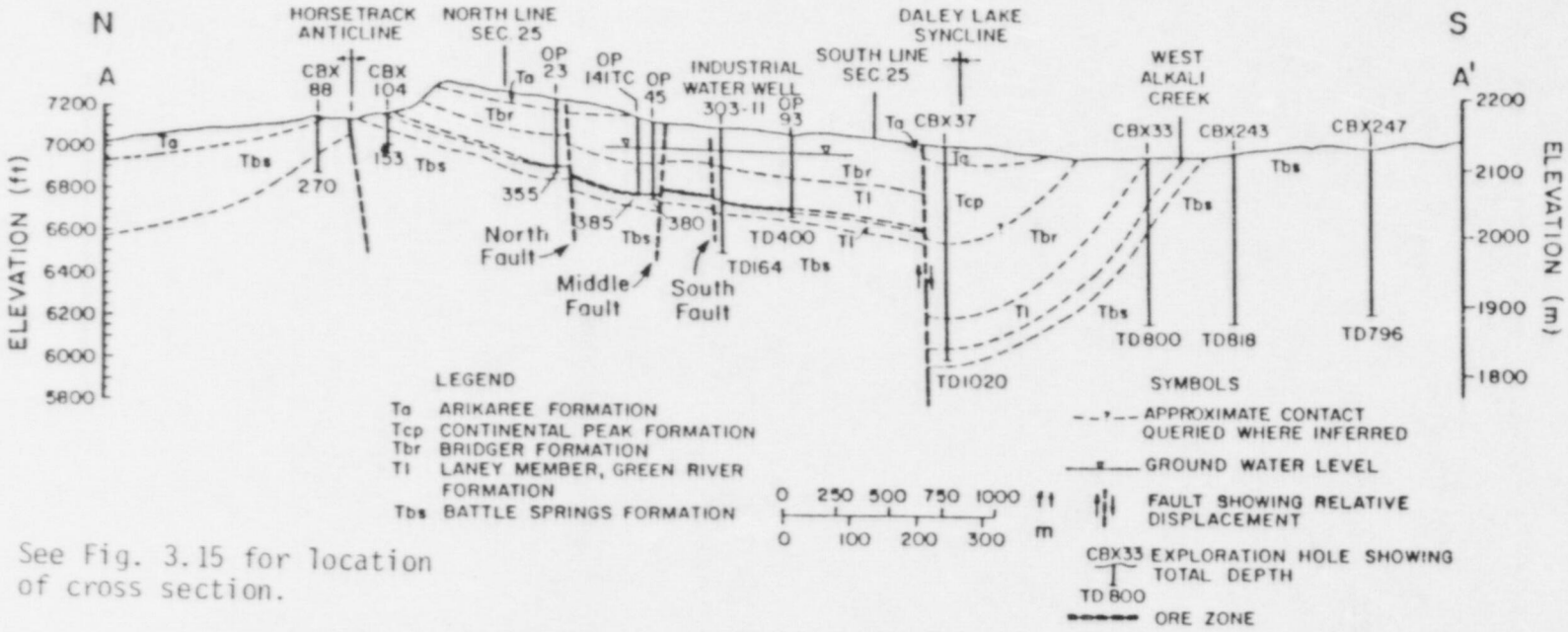


Fig. A.16. Surficial geology and structure map of the Bison Basin project area. *Source:* Modified from NRC, 1981a.



See Fig. 3.15 for location of cross section.

Fig. A.17. Geologic cross section through the Bison Basin mine site. *Source:* Modifed from NRC, 1981a.

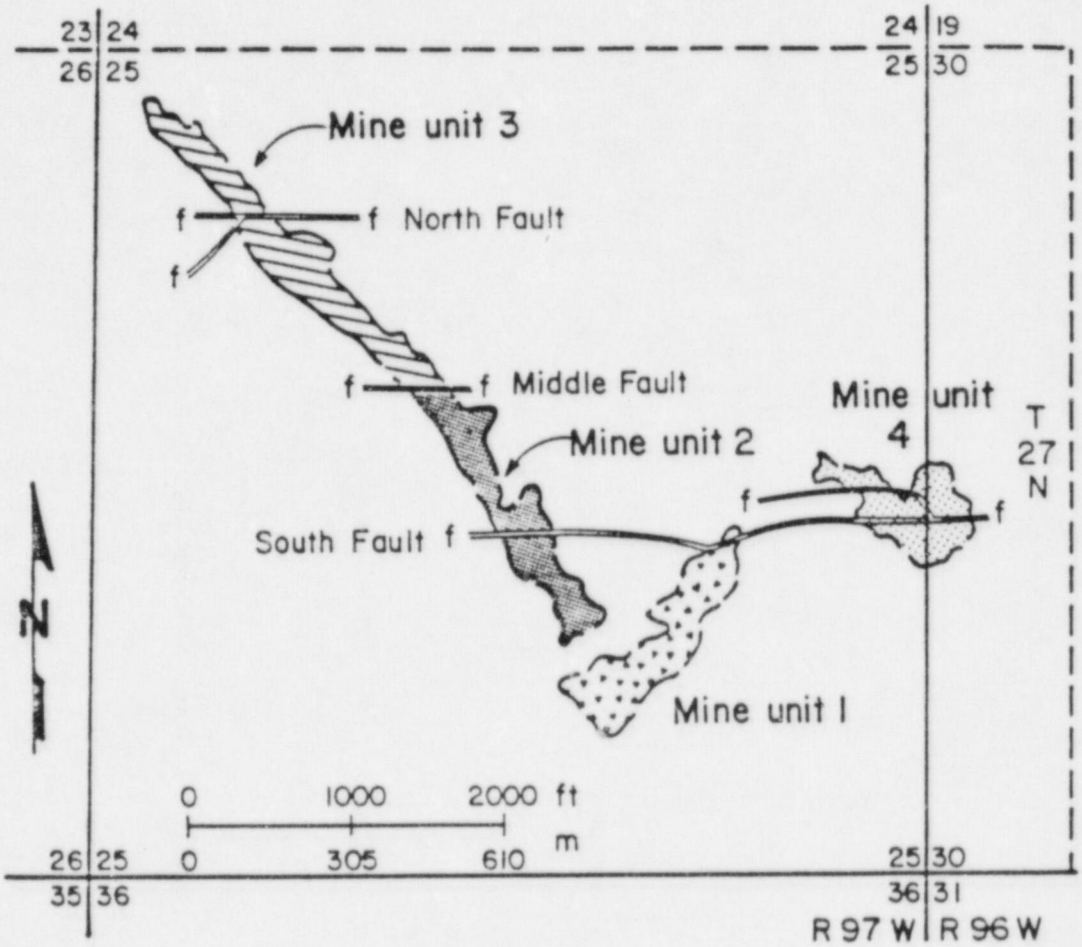


Fig. A.18. Location of faults (f) transecting the ore body at the Bison Basin mine. *Source:* Modified from NRC, 1981a.

agent, 2) chloride is released into solution in the uranium process, and 3) conductivity is a reasonable measure of TDS. It is unclear why sodium and sulfate were selected since they have high baseline concentrations.

UCL's were established by OPI (1979) as 20% above the highest baseline concentration for carbonate plus bicarbonate, sodium, sulfate, conductivity and chloride, and 0.015 mg/L above baseline for uranium. The NRC (1982a) modified the UCL for uranium for all monitoring wells to 1.0 mg/L above the highest baseline concentration. UCL's are established on a well-by-well basis for all parameters. Should a routine sample, and a confirmation sample taken within 48 hours, indicate that two or more indicators exceed their respective UCL's, or one indicator exceeds its UCL by more than 20%, the well is placed on excursion status.

Five monitoring wells in the production zone aquifer (basal sandstone of the "D" unit) surround the 0.4 ha (1 acre) research and development test site at a distance of about 60 m (200 ft) from the well field perimeter. In addition, one monitoring well was completed in the upper aquifer (basal sandstone of the "B" unit) to detect vertical excursions. During commercial-scale operations, ore zone monitoring wells are located about 90 m (300 ft) from the well field boundary and 180 m (600 ft) apart. Shallow monitoring wells are completed in the basal sandstone of the "B" unit and deep monitoring wells are completed in sandstone stringers located about 2.4 to 14 m (8 to 46 ft) below the ore zone (lower sandstone monitor zone). Monitoring well density for commercial-scale operation is about one shallow well for each 0.8 to 1.2 ha (2 to 3 acres); deep monitoring wells are at a density of about half that for shallow monitoring wells. No trend wells have been utilized at the Bison Basin mine.

Wells at the Bison Basin mine are drilled to a depth of approximately 1.5 m (5.0 ft) above the designated sandstone (NRC, 1981a). A PVC or fiberglass casing is cemented in the hole using a predetermined volume of cement slurry sufficient to fill the entire annulus. After the cement has set, the well is drillled past the bottom of the casing and through the target zone using a "nonclogging" drilling mud (not identified). A screen assembly consisting of a blank tail piece, screen, and blank riser pipe connected to a packer is placed in the well. Well development consists of high velocity jetting in combination with washing and air-bubbling cycles. Upon completion, OPI uses high-pressure injection tests between packers to check well integrity.

A.2.7 Analysis of Excursions

Ten monitoring wells at the Bison Basin mine have been reported on excursion status since research and development mining began in May 1979 (Fig. A.19). The excursions reported at two of the wells occurred during research and development operations, and eight wells were reported on excursion status immediately prior to or during commercial-scale mining operations. Seven of the monitoring wells reported on excursion status are completed in the ore-bearing basal sandstone of the "D" unit (horizontal excursions); two wells are completed in the basal sandstone of the "B" unit

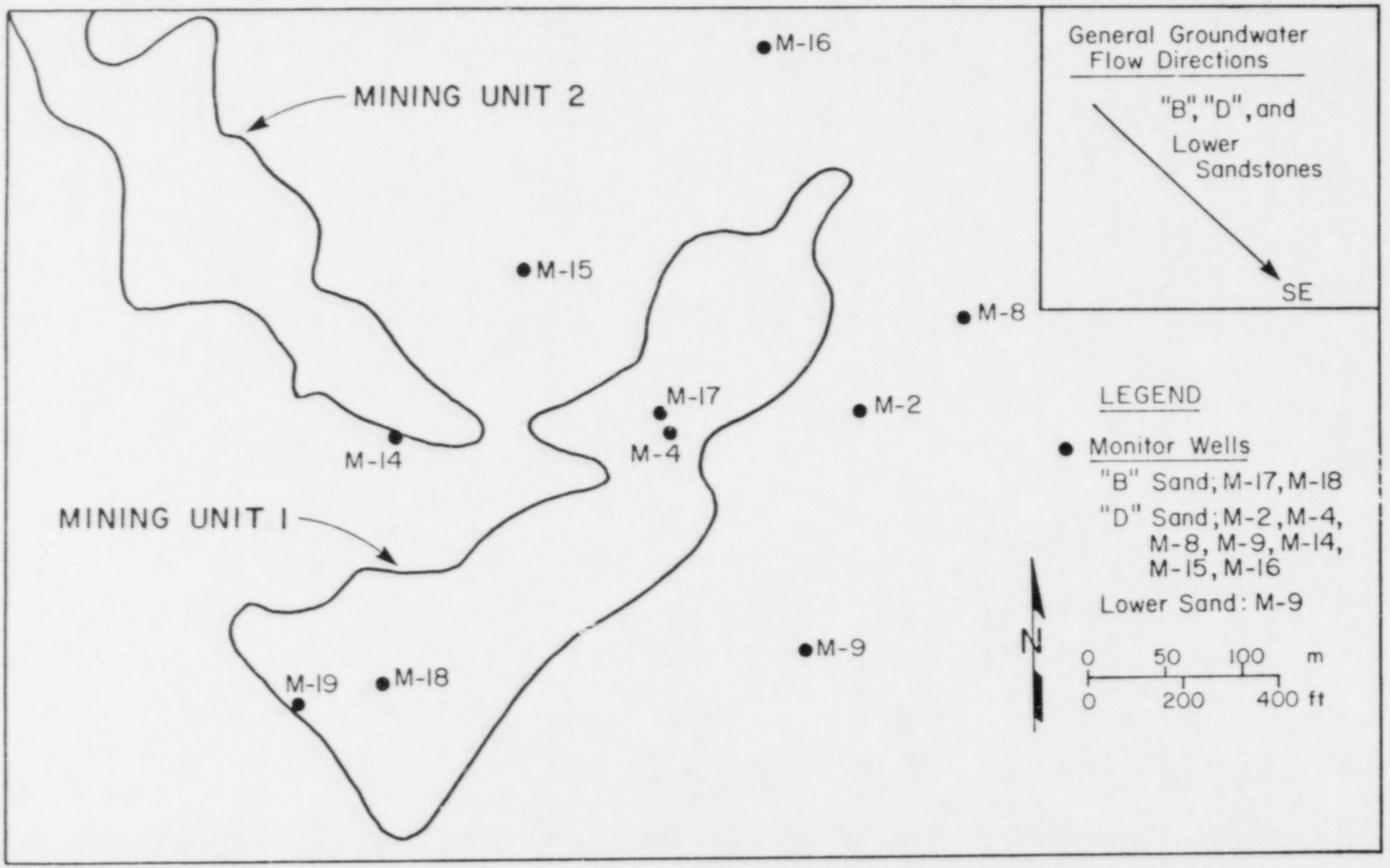


Fig. A.19. Location of monitoring wells reported on excursion status at the Bison Basin mine.

(vertical excursions). Table A.9 presents a summary of excursions that have occurred at the Bison Basin mine site.

Two horizontal excursion monitoring wells (M-2 and M-4) were reported on excursion status on February 4, 1981, during research and development operations (Fig. A.19). The well M-2 excursion was based on carbonate plus bicarbonate and chloride indicators; the well M-4 excursion was based on conductivity, carbonate plus bicarbonate, and chloride indicators. As concentrations of carbonate plus bicarbonate, and chloride rose above their UCL's in well M-2, concentrations of sodium, sulfate, and uranium, as well as conductivity also showed significant increases. Well M-2 was taken off excursion status on April 15, 1981, when all indicators were reduced significantly. Excursion status for well M-4 terminated on March 4, 1981, one month after the excursion was reported. The excursion indicators showed significant decreases by February 18, 1981, indicating the excursion at well M-4 was nearly cleaned up within two weeks of its occurrence.

Information concerning corrective measures used to control the lixiviant migration at the pilot mine site by OPI is not available. Water quality data indicate that a horizontal excursion occurred in 1981 in the vicinity of monitoring wells M-2 and M-4. This southward migration of lixiviant conforms with the general direction of ground-water flow in the basal sandstone of the "D" unit. Corrective measures to control lixiviant migration, such as increasing the pumping rate and/or reducing the injection rate, appear to have been implemented soon after excursion detection because of the rapid decrease in elevated parameters. Possible cause(s) for the excursion cannot be ascertained due to lack of data.

Water quality samples collected from all monitoring wells in Mine Unit 1 on August 12, 1981, one week prior to injection of lixiviant for the commercial-scale operation, indicated that uranium concentrations had exceeded the UCL in monitoring wells M-8 and M-9 (Fig. A.19). These two wells are completed in the basal sandstone of the "D" unit. Two weeks after the initiation of mining (September 3, 1981), monitoring well M-19 in the lower sandstone monitor zone exceeded its UCL for uranium. Wells M-8 and M-19 subsequently fluctuated on and off excursion status, based on uranium concentrations, until June, 1982, when the UCL for uranium was raised by the NRC. The UCL for uranium previously was established as the highest baseline concentration plus 0.015 mg/L, for each monitoring zone or aquifer. As a result of evidence supplied by OPI (1982a), the NRC (1982a) ammended the UCL for uranium to the highest baseline concentration plus 1.0 mg/L for each monitoring well in June, 1982.

Four additional monitoring wells (M-14, M-15, M-16, M-18) were placed on excursion status in 1981 due to elevated uranium concentrations. Wells M-14 and M-18 were taken off excursion status on December 23, 1981; well M-15 was taken off excursion status in January, 1982. Three of these wells (M-14, M-15, M-16) are completed in the ore zone (horizontal excursions); well M-18 monitors the basal sandstone of the "B" unit (vertical excursion). Uranium concentrations in these wells were several thousandths of a mg/L above their respective UCL's. Natural variations in uranium concentrations appear to be the cause of elevated uranium concentrations in

Table A.9. Summary of excursions at Bison Basin mine site during mining operations.

Well	Location	Aquifer[a]	Excursion Parameters							Excursion History		
			Date of Sample[b]	Conductivity (μmhos/cm)	Carbonate + Bicarb. (mg/L)	Chloride (mg/L)	Sodium (mg/L)	Sulfate (mg/L)	Uranium (mg/L)	On	Off	Duration
M-2	South of Mine Unit 1	D-Sand	UCL	2250	131	53	540	962	1.001			
			2/04/81	2050	238	68	495	922	0.001	2/04/81	4/15/81	2.5 months
M-4	Within Mine Unit 1	D-Sand	UCL	2054	131	46	510	934	1.002			
			2/04/81	2125	349	109	503	870	0.002	2/04/81	3/04/81	1 month
M-8	East of Mine Unit 1	D-Sand	UCL	2580	140	40	504	1092	0.018			
			8/12/81	2030	107	34	393	860	0.027	8/12/81	9/10/81	1 month
			9/03/81	2250	120	39	385	760	0.036	12/09/81	1/08/82	1 month
M-9	South of Mine Unit 1	D-Sand	UCL	2628	140	42	492	1037	0.018			
			8/12/81	2000	122	36	375	825	0.838	8/12/81	3/30/82	7.5 months
			8/26/82	1850	261	47	506	792	--	8/26/82	remains	c
M-14	Within Mine Unit 3	D-Sand	UCL	2640	152	35	524	1020	0.018			
			12/09/81	2290	124	26	405	805	0.022	12/09/81	12/23/81	2 weeks
			2/11/82	3400	163	275	592	1110	0.156	2/11/82	3/30/82	1.5 months
M-15	North of Mine Unit 1	D-Sand	UCL	2580	146	50	516	1078	0.018			
			12/23/81	1950	122	29	380	860	0.025	12/23/81	1/82	1 month
M-16	North of Mine Unit 1	D-Sand	UCL	2620	146	48	540	1018	0.018			
			11/25/81	2350	117	28	419	810	0.024	11/25/81	12/23/81	1 month
M-17	Within Mine Unit 1	B-Sand	UCL	3552	124	23	659	1506	0.027			
			3/30/82	2450	176	51	536	1000	0.005	3/30/82	4/27/82	1 month
M-18	Within Mine Unit 1	B-Sand	UCL	3312	112	24	654	1476	0.027			
			11/25/81	3000	80	17	528	1340	0.033	11/25/81	12/23/81	1 month
M-19	Within Mine Unit 1	Lower Sand	UCL	2256	152	64	468	739	0.012			
			9/03/81	1760	119	43	342	580	0.017	9/03/81	9/1/81	1 week
				1900	124	40	354	586	0.016	11/25/81	1/08/82	1.5 months

[a]D-Sand is the uranium-bearing basal sandstone of the "D" unit; B-Sand is the basal sandstone of the "B" unit; Lower Sand is the lower sandstone monitor zone.

[b]UCL = Upper Control Limit; the first sample date is when the well went on excursion status; dates that follow are representative of highest excursion values achieved or giving concentrations of interest.

[c]This well was still in excursion status in 1984.

wells that have high uranium levels but normal levels of other parameters. If an uncontrolled movement of lixiviant had occurred, these wells probably would have shown an increase in the concentration of one or more of the above excursion indicators. Additionally, this problem does not appear to be restricted to a few specific wells or one specific aquifer.

Two excursion monitoring wells (M-9 and M-14) in the basal sandstone of the "D" unit that previously were on excursion status in 1981 based only on uranium, were placed on excursion status on February 11, 1982, based on increased concentrations of all six excursion indicators (conductivity, carbonate plus bicarbonate, chloride, sodium, sulfate, and uranium). OPI adjusted the well field pumping patterns and rates immediately following detection of the excursion and positive results were observed soon thereafter. Well M-14 was taken off excursion status on March 30, 1982. The gradient toward well M-14 may have been caused by over-injection in Mine Unit 1.

Well M-9 returned to excursion status on August 26, 1982, based on elevated concentrations of carbonate plus bicarbonate, chloride, and sodium. The excursion probably was due to over-injection in a portion of the well field near well M-9 (OPI, 1982b). Corrective actions were initiated by decreasing the injection rate and increasing the production rate in wells near well M-9. Although the well remained on excursion status as of September 1, 1982, a sample taken on August 30, 1982, indicated that sodium was below its UCL and carbonate plus bicarbonate was decreasing.

Monitor well M-17, located within Mine Unit 1 and completed in the basal sandstone of the "B" unit (used to monitor for vertical excursions), was placed on excursion status on March 30, 1982. A water sample indicated that carbonate plus bicarbonate and chloride remained slightly above their UCL's until August, 1982. OPI did not report any corrective actions for this excursion but eventually the well was taken off excursion status. Well M-17 is the only monitoring well completed in the basal sandstone of the "B" unit that was placed on excursion status due to excursion indicators other than uranium. Data indicate that hydraulic communication had occurred between the ore zone and the basal sandstone of the "B" unit; however, the reason is not clear. No faults have been detected in the vicinity of well M-17 in Mine Unit 1. Because no other excursions have been detected in the basal sandstone of the "B" unit within Mine Unit 1, leakage through the confining layer(s) appears to be insignificant; however, the vertical excursion in well M-17 may have been caused by casing damage and/or inadequate sealing around the casing.

A.2.8 Restoration

Aquifer restoration testing has been conducted at the OPI in situ pilot project in Mine Unit 1. The mining phase of the pilot test operation lasted from May 1 through August 1 1979, using a sodium carbonate/bicarbonate lixiviant. The restoration phase of the project took place during the period August 1 through September 14, 1979. OPI (1980) presents a complete description of the restoration process and related water quality data. A reverse osmosis unit was used to restore the quality of ground

water affected by the solution mining operation. Background water quality in the basal sandstone of the "D" unit (ore zone) is unsuitable for any use other than industrial, primarily due to high pH and the high concentrations of sulfate, sodium and ^{226}Ra. Approximately eight aquifer pore volumes of ground water were circulated through the reverse osmosis unit and the well field during restoration. Concentrations of all measured constituents were reduced either to less than baseline or to less than drinking water standards.

On August 1, 1979, the addition of leaching chemicals to the lixiviant ceased; however, ground water continued to be circulated through the well field and the processing plant for four days to reduce the uranium concentrations. Between August 5 and August 9, 1979, approximately one aquifer pore volume was pumped from the well field through the reverse osmosis unit to an evaporation pond. This operation simulated the activities that would occur when lixiviant is transferred between well fields during commercial production. From August 10 through September 14, 1979, ground water from the production wells was passed through the reverse osmosis unit and subsequently injected as clean water into the pilot well field. Concentrated brine from the reverse osmosis unit was discharged to the evaporation pond.

Restoration of the ground water affected by the pilot in situ uranium mining project was completed on September 14, 1979. Concentrations of selected constituents from production well P-22 are shown in Table A.10. Ground-water quality in the restored aquifer was monitored for six months to assess the water quality stability; little or no increase in ion concentrations was seen during this period. Concentrations of ^{226}Ra exceeded applicable standards both before and during mining. Post-restoration analytical data indicate that the concentration levels of toxic metals are similar to baseline. In general, the results of restoration and post-restoration sampling show that most of the parameters were returned to less than baseline plus 10% or less than drinking water standards (OPI, 1980). The NRC (1981a) believes that the performance of the reverse osmosis unit can be improved from the 62% reinjection attained during the test to 90% reinjection. This improvement should further reduce both the aquifer pore volumes required for treatment and the evaporation pond volume and surface requirements.

OPI proposed to accomplish restoration of the commercial-scale mining units through a combination of ground-water sweeping and clean water recirculation. Approximately one aquifer pore volume of ground water initially would be transferred from the mining area last mined to the next area to be mined with the simultaneous transfer of an equivalent amount of water in the opposite direction. After this first aquifer pore volume transfer, recirculation of ground water from the leached mine unit through a reverse osmosis treatment unit would begin. Clean water recirculation would continue until sampling indicates that the water quality in the affected aquifer meets established restoration criteria.

Table A.10. Water quality[a] in wells before, during, and after restoration at the research and development pilot site at Bison Basin.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Leaching/ Prerestoration[b] 8/5/79	During Restoration[b] 9/5/79	Post-Restoration[b] 10/5/79	Post-Restoration[b] 3/18/80
Bicarbonate	500	73	360	127	140	143
Chloride	250	34	215	35	35	35
Sodium	200	443	648	397	400	425
Sulfate	250	900	787	720	783	915
Uranium (U_3O_8)	5.0	0.002	0.85	0.026	0.131	0.735
Conductivity (μmhos/cm)	--	1850	2700	1620	1630	1970
pH (units)	6.5-8.5	9.7	8.0	8.1	8.4	7.7

[a]Values are mg/L except as noted.

[b]Reported values are average from five wells (P-7, P-16, P-19, P-22, P-31).

Source: OPI, 1980.

A.3 ROCKY MOUNTAIN ENERGY COMPANY - NINE MILE LAKE PROJECT

A.3.1 Description of the Site

Rocky Mountain Energy Company (RMEC) operates the Nine Mile Lake uranium solution mine about 14.5 km (9 mi) north of Casper in Natrona County, Wyoming (Fig. A.20). The topography of the mine site generally is flat over the mine area. The surrounding area consists of gently rolling hills. Elevation of the mine site is about 1600 m (5300 ft) with 18 m (60 ft) of relief. The mine site is located within the North Platte River drainage basin; runoff from the project area generally is contained in lowland depressions prior to reaching the North Platte River. The North Platte River, the nearest perennial stream, is located approximately 8 km (5 mi) south of the mine site. All streams within the project area flow only in response to snowmelt and precipitation events. Average annual precipitation recorded at Casper is 28.9 cm (11.4 in), but large variations occur from year-to-year.

Research and development testing was conducted at the Nine Mile Lake mine (Fig. A.21) in four well patterns. Leaching continued sequentially in these four patterns from November, 1976, through November, 1980. Restoration tests were conducted at the four pilot sites with final completion of all restoration by February 1, 1982. RMEC applied to the NRC on February 12, 1979, for a license to construct and operate a commercial-scale mine at the Nine Mile Lake site. The ore body to be mined commercially is located in Sections 21, 27, 28, 33, and 34, T35N, R79W, and Section 3, T34N, R79W (Fig. A.21). RMEC has since withdrawn license applications for the development of commercial-scale operations due to poor uranium market conditions and problems encountered in uranium recovery and restoration at the site. In addition, no further research and development testing was planned for Nine Mile Lake as of June, 1984.

A.3.2 History of Mining Operations

Laboratory testing was conducted on drill core samples. Over 80 tests analyzed sulfuric acid (H_2SO_4), sodium carbonate/bicarbonate, and ammonium bicarbonate as lixiviants; results demonstrated that H_2SO_4 leaching solutions gave much better uranium extraction than did carbonate lixiviants (RMEC, 1979). These tests suggested that the optimal lixiviant for the Nine Mile Lake ore body consists of 3.0 g/L of H_2SO_4, has a pH of 1.5, and an oxidation potential of 450 mV achieved by addition of hydrogen peroxide (H_2O_2).

A pilot in situ leaching project began at Nine Mile Lake in 1976 to verify laboratory results. Four test patterns of injection and recovery wells were drilled and a uranium recovery plant was constructed (Fig. A.22). Pattern 1 consisted of a seven-spot well pattern of six injection wells equally spaced at a 15 m (50 ft) radius from a central recovery well. All wells penetrated about 7.5 m (25 ft) of the upper ore sand of the Teapot Sandstone. Composition of the lixiviant averaged about 4 g/L of H_2SO_4 (pH 1.7), 0.5 g/L of H_2O_2, and 0.5 g/L of iron sulfate (Fe_2SO_4). Iron sulfate was added to help oxidize the uranium. Leaching continued

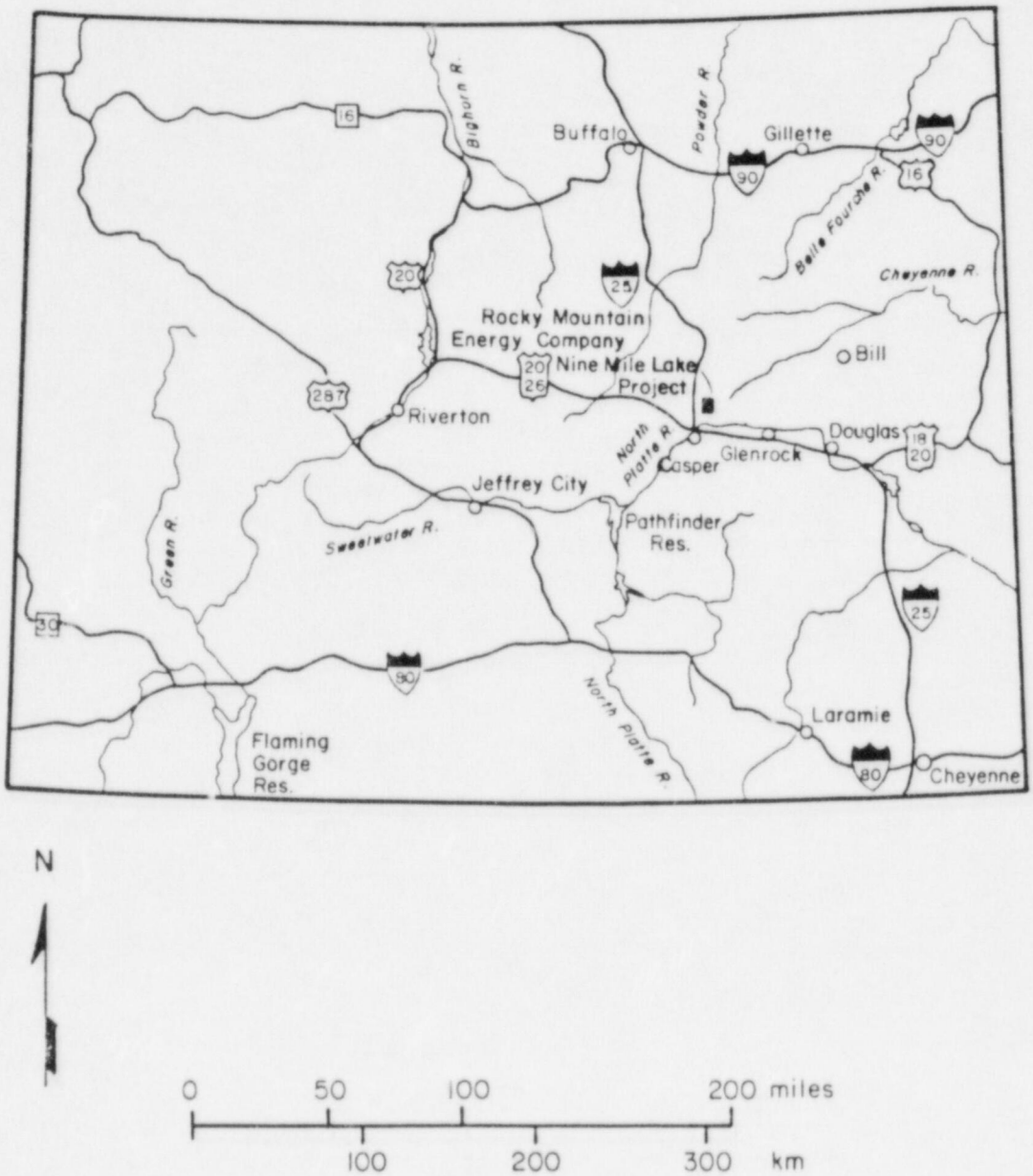


Fig. A.20. Location map of the Rocky Mountain Energy Company Nine Mile Lake mine.

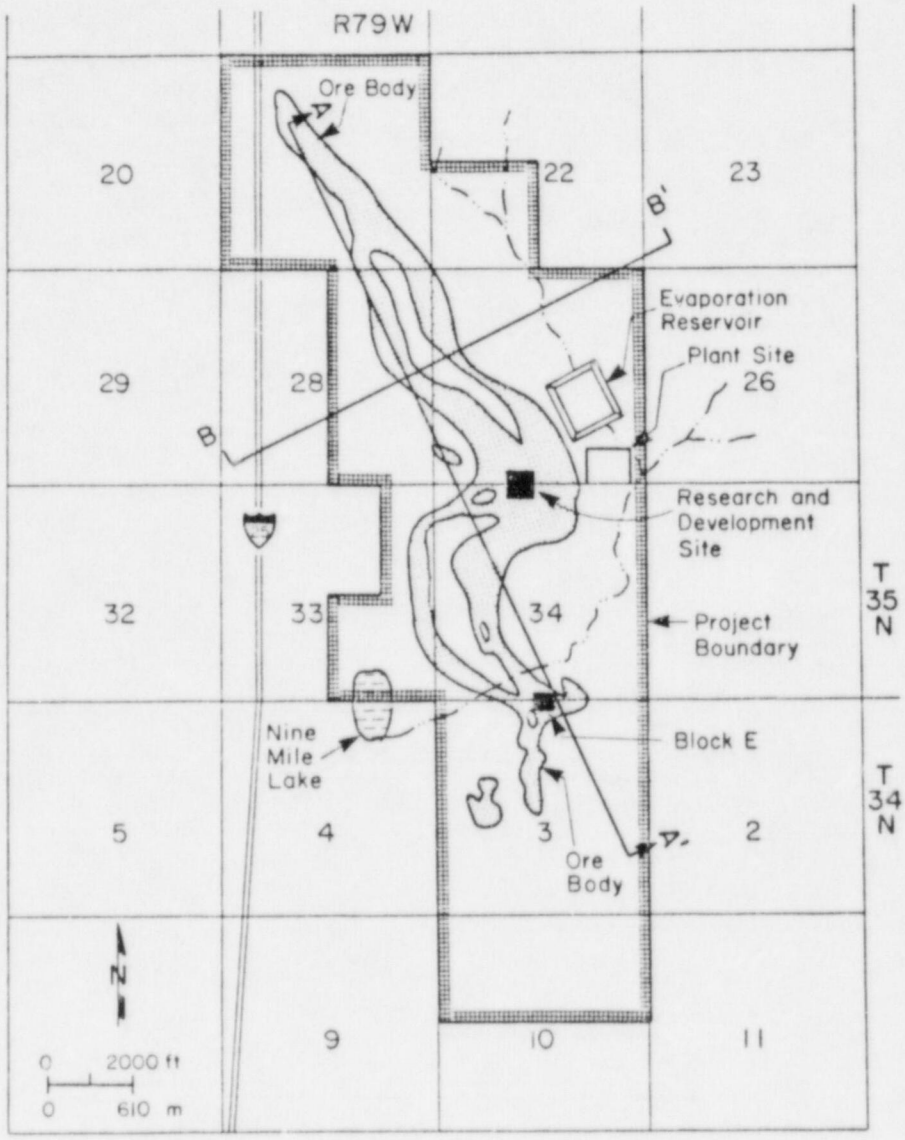


Fig. A.21. Map of the Nine Mile Lake uranium project site and immediate vicinity.

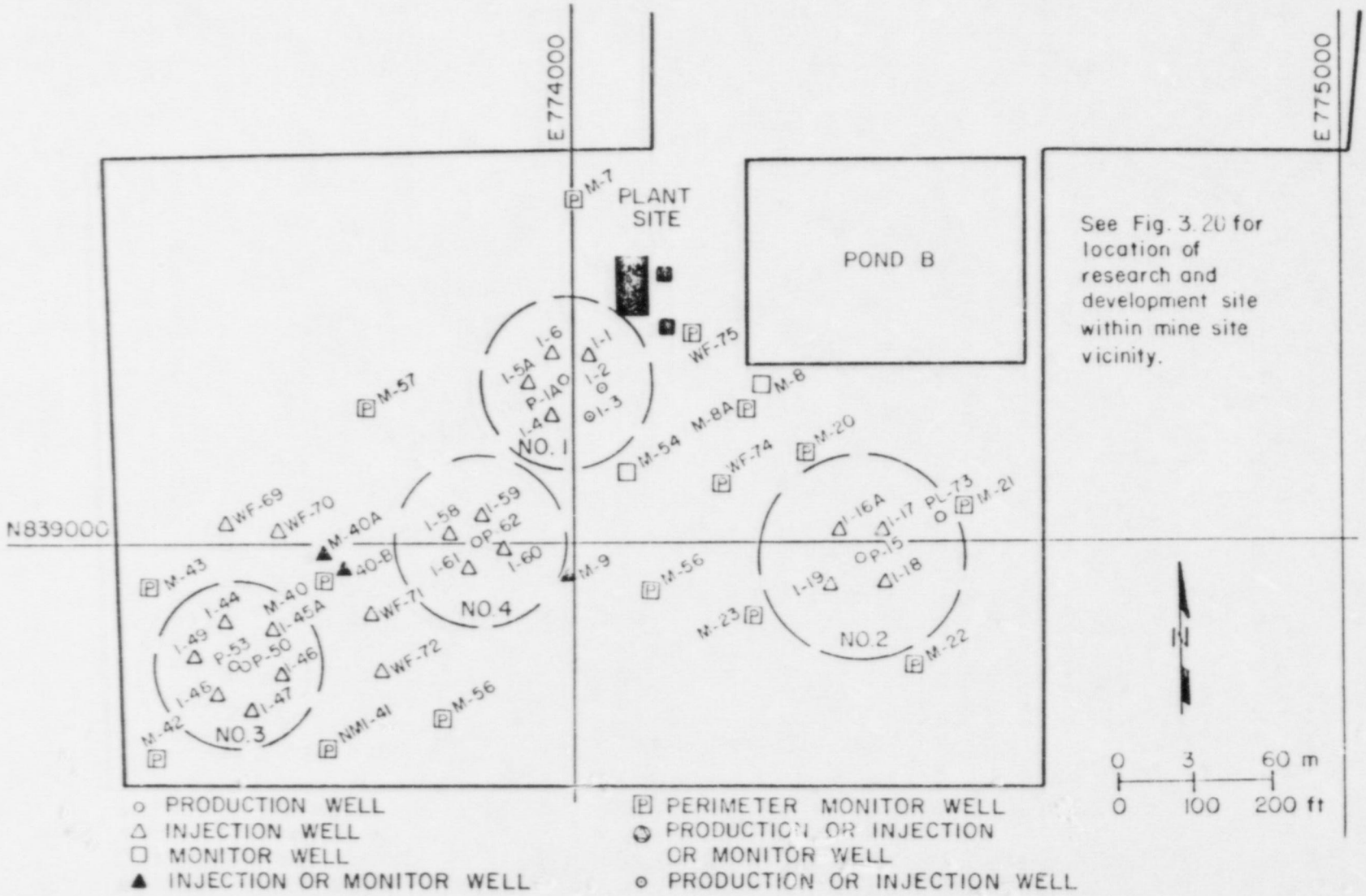


Fig. A.22. Research and development test patterns at the Nine Mile Lake site. *Source:* Modified from RMEC, 1982a.

sporadically at the production rate of about 2.5 L/s (40 gal/min) for ten months (November, 1976, to August, 1977). Approximately seven aquifer pore volumes of lixiviant were injected during this time period. Numerous problems due to well plugging by gypsum, low uranium recovery, and equipment failure plagued leaching operations in Pattern 1.

Pattern 2 was a 15 m (50 ft) radius, five-spot well pattern consisting of four injection wells around a central recovery well. Wells were completed in the lower ore zone of the Teapot Sandstone. Production and total injection rates were about 2.6 L/s (42 gal/min) and 2.5 L/s (40 gal/min), respectively. The leaching solution consisted of 3 to 5 g/L of H_2SO_4 and 1 g/L of H_2O_2. Leaching was conducted from December, 1977, through September, 1978, during which time approximately 13 aquifer pore volumes of ground water were produced. Injectivity generally was good in Pattern 2, although plugging problems due to "fungus growth" and gypsum precipitation were encountered in April, 1978. Neither the details of nor the evidence are provided for the conclusion that a "fungus growth" developed.

Pattern 3 was constructed using an eight-spot pattern consisting of six injection wells located 18 m (60 ft) around two central production wells. Each injection well was completed into the two separate ore zones of the Teapot Sandstone. Production wells were completed independently with one each in the upper and lower ore zones. The primary purpose of the Pattern 3 test was to evaluate the feasibility of mining of the upper and lower ore zones simultaneously. Leaching continued from September, 1979, to April, 1980, using a H_2SO_4 lixiviant and separate oxidants in the form of peroxymono-sulfuric acid (H_2SO_5) or oxygen. Total injection flow rates were set at 3.7 L/s (58 gal/min) and production rates were maintained initially at 3.8 L/s (60 gal/min) with half the flow coming from each production well. A total of 5.6 aquifer pore volumes was produced during leaching. Sporadic problems were encountered with well plugging, frozen lines, and equipment failures. Problems also were encountered in controlling lixiviant distribution to the two ore zones.

Pattern 4 utilized a 15 m (50 ft) radius, five-spot well pattern of four injection wells surrounding a central production well. Leaching was conducted from June to November, 1980. A sodium carbonate/bicarbonate lixiviant was used. Carbon dioxide gas was added to maintain a pH of approximately 7.5. The initial oxidant was about 0.5 g/L H_2O_2; oxygen was used later in the test. Additional information regarding carbonate leaching in Pattern 4 is not available; however, RMEC (1982b) reports that the results from this test were disappointing.

A seven-spot production cell with a 21 m (70 ft) spacing was the pattern proposed for use during commercial-scale production at Nine Mile Lake. The proposed lixiviant was 3 to 5 g/L of H_2SO_4 and 1 g/L of H_2O_2 in water. Approximately 1% to 3% of the total production of ground water was to be bled from the leach circuit prior to injection into the ore zone. The uranium recovery plant was to use ion exchange resin columns to selectively adsorb uranium from the recovered solution; sulfate ions were to be displaced in the process. Vanadium also was to be recovered from the lixiviant. The lixiviant was to be refortified with

H_2SO_4 and H_2O_2, filtered and returned to the well field for reinjection. The research and development tests used a similar uranium recovery process.

A.3.3 Regional Setting

A.3.3.1 Regional Geology

The Nine Mile Lake project site is located on the southwest flank of the Powder River Basin approximately 96 km (68 miles) south of the Irigaray site (Fig. A.5). The axis of the basin is approximately 32 km (20 miles) east of the mine site. The Casper Arch which lies about 24 km (15 miles) to the west of the mine site constitutes a reversal in dip of the strata toward the west into the Wind River Basin. Rocks within the Powder River Basin have been gently folded regionally along axes that parallel the north-northwest basin trend. Dip of the strata in the vicinity of Nine Mile Lake generally is about six degrees east-northeast; however, localized folds also exist in the area. Major faults are known to exist in the interior of the Powder River Basin.

Solution mining at the Nine Mile Lake site occurs in the Teapot Sandstone, the uppermost member of the Mesa Verde Formation, which consists of coarse clastic materials and shales deposited during a transgressive-regressive phase of marine sedimentation during late Cretaceous time (RMEC, 1979).

A.3.3.2 Regional Ground-water Flow Systems

Ground water in the region occurs in both unconsolidated deposits and indurated strata. Unconsolidated materials have not been developed as sources of ground water, except for alluvial deposits in the North Platte River valley. Ground-water movement in indurated strata is controlled largely by the structural and stratigraphic features of the Powder River Basin. Confined conditions occur in most of the aquifers due to the regional dip and alternating sequences of sandstones (aquifers) and shales (aquitards). Recharge in the area occurs primarily by infiltration of direct precipitation and streamflow. Cretaceous strata at the mine area dip about six degrees to the northeast; the general direction of ground-water flow is toward the axis of the Powder River Basin.

Ground water is an important resource in the region and is utilized for livestock watering, domestic purposes, limited irrigation, and industrial supply. A number of high-yield public water supply wells are located in the alluvium along the North Platte River about 8 km (5 mi) south of the site. Within a 5-km (3-mi) radius of the mine site, only two wells (stock watering) are completed in the Teapot Sandstone. No public water supply wells are located within 5 km (3 mi) of the mine property.

A.3.4 Description and Analysis of Data Collected

A.3.4.1 Exploration

Exploration boreholes have been drilled throughout the Nine Mile Lake mine area for evaluation of ore thickness and grade; however, information pertaining to the number and locations of these boreholes is not available. It is not known whether other companies had conducted exploration work in the area prior to RMEC's acquisition of mineral rights at Nine Mile Lake. Therefore, no conclusions can be drawn regarding the existence of preferential pathways for vertical lixiviant migration through abandoned boreholes. The potential for this problem exists at almost every in situ uranium mine site.

Little information is available in NRC docket files concerning geophysical logging at the Nine Mile Lake site. Geophysical logs were recorded for 14 boreholes in order to construct two cross sections; one section was constructed along the north-south trend of the ore body and the other was constructed across (east-west) the ore body. Geophysical records for these 14 boreholes consist of gamma, spontaneous potential, resistivity and neutron logs. Gamma logs were recorded for all of the boreholes to evaluate grade and thickness of mineralization (RMEC, 1979).

The shape and magnitude of log response in the vicinity of a particular rock type were used to identify lithologic units (RMEC, 1979). Unique signatures on the logs correspond to the alternating sequences of sandstone and shale across the site. Only two of the 14 geophysical logs extend below the ore-bearing sandstone. No major sandstone units could be identified by the authors within about 200 m (650 ft) below the ore-bearing sandstone from these two logs. Occasional thin sandstone stringers could be identified throughout the unit overlying the ore zone.

A.3.4.2 Aquifer Tests

Several single-well and multiple-well constant discharge pumping tests were conducted between 1976 and 1978 at the Nine Mile Lake uranium mine. All tests were conducted in the Teapot Sandstone (ore zone). Five multiple-well pumping tests and eleven single-well tests were conducted. Only one of the multiple-well tests utilized an observation well completed in a unit other than the Teapot Sandstone. A summary of the pumping tests conducted at the Nine Mile Lake mine site is presented in Table A.11. Locations of the aquifer tests are shown on Fig. A.23.

Data for aquifer tests 1 and 2 are not available from which to perform an independent analysis. For aquifer tests 3 and 4 which only partially penetrated the ore zone, In-Situ Consulting (1978a) used an unsteady state

Table A.11. Summary of aquifer tests conducted at the Nine Mile Lake mine.

Test No.	Pumping Well	Location	Aquifer[a]	Total Depth of Tested Well m (ft)	Number of Observation Wells	Wells Monitored[a]	Type of Test[b]	Date of Test	Length of Test (hrs)	Discharge Rate L/s (gal/min)
1	P-1A	Pattern 1	T-Sand	?	6	T-Sand	CD	1976	?	?
2	P-15	Pattern 2	T-Sand	?	4	T-Sand	CD	1977	?	?
3	29A	Block E	T-Sand	98 (320)	10	T-Sand	CD	6/26/78	2	2.7 (43)
4	39	Block E	T-Sand	104 (340)	10	T-Sand	CD	6/28/78	7	1.4 (22)
5	29A	Mine Site	T-Sand	98 (320)	1	PB Shale, P-Sand	CD	8/31/78	1.5	3.1 (49)
6	BM-1	Mine Site	T-Sand	155 (510)	0		CD	?	1.8	0.9 (14.2)
7	MM-11	Mine Site	T-Sand	155 (510)	0		CD	?	1.8	3.1 (49.2)
8	BM-2	Mine Site	T-Sand	183 (600)	0		CD	?	1.8	3.2 (50.2)
9	BM-3	Mine Site	T-Sand	165 (540)	0		CD	?	1.8	3.2 (50.2)
10	BM-4	Mine Site	T-Sand	50 (165)	0		CD	?	1.8	2.4 (37.7)
11	BM-5	Mine Site	T-Sand	105 (345)	0		CD	?	1.8	2.8 (44)
12	BM-8	Mine Site	T-Sand	232 (760)	0		CD	?	1.8	3.2 (50.2)
13	BM-9	Mine Site	T-Sand	118 (387)	0		CD	?	1.8	2.7 (42.8)
14	BM-10	Mine Site	T-Sand	219 (720)	0		CD	?	2.0	3.1 (49.2)
15	BM-11	Mine Site	T-Sand	151 (495)	0		CD	?	1.8	3.0 (47.3)
16	Rod Well	Mine Site	T-Sand	?	0		CD	?	1.8	3.1 (49.2)

[a]T-Sand is the uranium-bearing Teapot Sandstone; PB Shale is the Pumpkin Buttes Shale; P-Sand is the Parkman Sandstone.

[b]CD = Constant discharge pumping test.

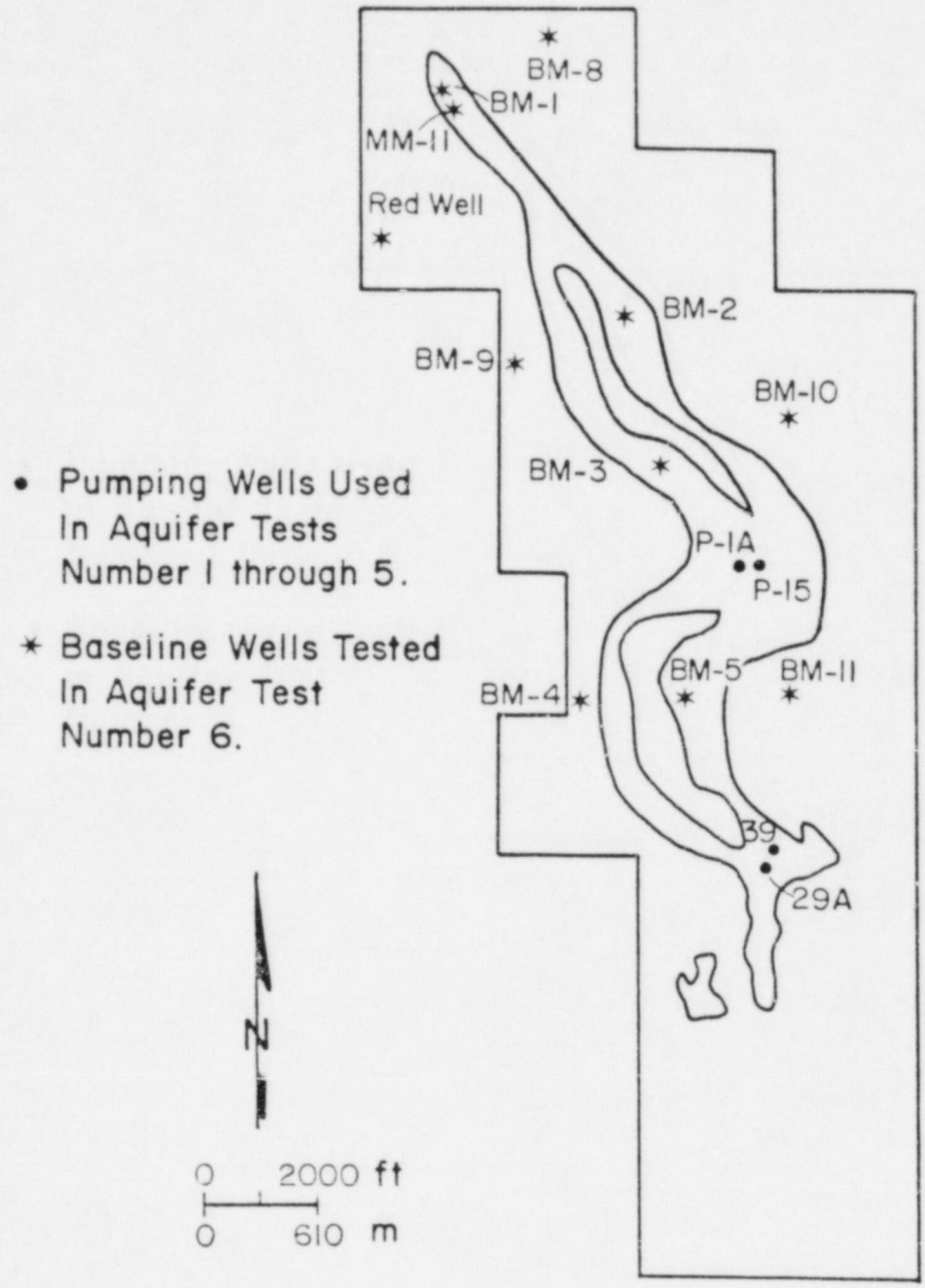


Fig. A.23. Location map of pumping wells for the aquifer tests conducted at the Nine Mile Lake mine site.

type curve method by McKee and Way (not referenced) to analyze individual observation well data within the region of partial penetration effects. The Theis (1935) nonsteady state type curve method was used to analyze data from observation wells 35, 36, 37 and 38 (tests 3 and 4) which are located beyond the region of partial penetration effects (RMEC, 1979). Directional transmissivities were calculated by In-Situ Consulting (1978a) using the Papadopulos (1965) method of analysis.

Drawdown data for aquifer tests 3 and 4 from wells 35, 36, 37, and 38 which were not affected by partial penetration were analyzed by the authors using the Hantush and Jacob (1955) leaky artesian method. Table A.12 presents values of the horizontal hydraulic conductivity and storativity of the Teapot Sandstone and the vertical hydraulic conductivity of the aquitard (Lewis Shale) overlying the Teapot Sandstone estimated by the authors. The values of vertical hydraulic conductivity of the aquitard are somewhat questionable because of the partial penetration of the ore zone and the short pumping periods for tests 3 and 4 (two and seven hours, respectively). The pumping period was not long enough to get an adequate curve from which to match to the leaky type curves. However, most of the curves were beginning to flatten and a match to leaky curves could be estimated with some certainty. The pumping periods for all pumping tests should have been longer in order to better define potential boundary and leakage conditions, or a ratio method test (Neuman and Witherspoon, 1972) should have been designed.

Pumping test number 5 was conducted to detect possible vertical leakage. The pumping well (29A) is completed in the Teapot Sandstone; the observation well (34B) is completed in the underlying Pumpkin Buttes Shale and the Parkman Sandstone. No water level change was observed in well 34B during the one and one-half hour pumping period. However, considering the short duration of the test, no conclusion can be reached regarding hydraulic communication between aquifers.

Aquifer tests 6 through 16 consisted of single-well pumping tests conducted in wells completed in the Teapot Sandstone throughout the Nine Mile Lake mine site. A step drawdown test was conducted prior to each constant discharge test in order to evaluate well loss. Only one of the 11 wells (MM-11) exhibited a relatively low well efficiency (60%). Drawdown data were analyzed by In-Situ Consulting (1978a) using the Jacob straight line method (Cooper and Jacob, 1946), the Papadopulos and Cooper (1967) large diameter well type curve method, and the nonsteady state type curve method (Agarwal et al, 1970). None of these methods facilitate the estimation of vertical saturated hydraulic conductivity of the aquitards. Reasonably consistent values of hydrogeologic parameters for the aquifer were derived using the three methods of analysis, although pumping periods of only about two hours result in a deficiency with respect to the amount of data available for analysis. Hydraulic properties estimated from single-well tests generally are representative of relatively small areas around the pumping well.

The Teapot Sandstone (ore zone) is the only aquifer at the Nine Mile Lake mine site that has been characterized by aquifer testing. The

Table A.12. Summary of aquifer and aquitard characteristics estimated by D'Appolonia Consulting Engineers, Inc., In-Situ Consulting, and the authors from aquifer test data for the Nine Mile Lake mine.

Test No.	Pumping Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1	P-1A	T-Sand	DCE	$1x10^{-3}$ to $3x10^{-3}$ (2.8 to 8.5)	5 to 17 (55 to 180)	$1.5x10^{-4}$	--	Tm,T	CD
2	P-15	T-Sand	ISC	$5x10^{-4}$ to $2x10^{-3}$ (1.4 to 5.7)	4 to 17 (44 to 180)	$1.3x10^{-4}$	--	?	CD
3	29A	T-Sand	ISC	$5x10^{-4}$ to $1x10^{-3}$ (1.4 to 3.3)	12 to 24 (126 to 255)	$2.3x10^{-4}$	--	MW,T,P	CD
			Authors	$1x10^{-4}$ to $8x10^{-4}$ (0.4 to 2.2)	3 to 16 (32 to 173)	$3x10^{-5}$	$3x10^{-6}$ to $4x10^{-6}$ ($9x10^{-3}$ to $1x10^{-2}$)	HJ	CD
4	39	T-Sand	ISC	$6x10^{-4}$ to $1x10^{-3}$ (1.7 to 3.0)	13 to 23 (136 to 243)	$2x10^{-4}$	--	MW,T,P	CD
			Authors	$7x10^{-5}$ to $7x10^{-4}$ (0.2 to 2.1)	1 to 16 (10 to 170)	$2x10^{-5}$	$9x10^{-8}$ to $3x10^{-7}$ ($3x10^{-4}$ to $9x10^{-4}$)	HJ	CD
6	BM-1	T-Sand	ISC	$2x10^{-3}$ to $3x10^{-3}$ (5.9 to 8.1)	44 to 60 (473 to 646)	$9.4x10^{-5}$	--	A,PC,CJ	CD
7	MM-1	T-Sand	ISC	$2x10^{-3}$ to $3x10^{-3}$ (5.0 to 7.5)	37 to 56 (398 to 603)	$6.5x10^{-5}$	--	A,PC,CJ	CD
8	BM-2	T-Sand	ISC	$1x10^{-3}$ to $2x10^{-3}$ (3.0 to 6.2)	34 to 46 (366 to 495)	$8.4x10^{-5}$	--	A,PC,CJ	CD
9	BM-3	T-Sand	ISC	$1x10^{-3}$ to $2x10^{-3}$ (3.0 to 5.8)	32 to 43 (344 to 463)	$9.4x10^{-5}$	--	A,PC,CJ	CD
10	BM-4	T-Sand	ISC	$2x10^{-3}$ to $4x10^{-3}$ (6.2 to 10.2)	46 to 76 (495 to 818)	$8.0x10^{-5}$	--	A,PC,CJ	CD
11	BM-5	T-Sand	ISC	$5x10^{-4}$ to $8x10^{-4}$ (1.4 to 2.2)	10 to 16 (108 to 172)	$6.6x10^{-5}$	--	A,PC,CJ	CD

Table A.12. (Continued)

Test No.	Pumping Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
12	BM-8	T-Sand	ISC	$2x10^{-3}$ to $3x10^{-3}$ (5.7 to 7.0)	42 to 52 (452 to 560)	$9.1x10^{-5}$	--	A,PC,CJ	CD
13	BM-9	T-Sand	ISC	$1x10^{-3}$ to $2x10^{-3}$ (4.4 to 6.5)	33 to 48 (355 to 516)	$9.0x10^{-5}$	--	A,PC,CJ	CD
14	BM-10	T-Sand	ISC	$1x10^{-3}$ to $2x10^{-3}$ (3.1 to 5.5)	23 to 41 (247 to 441)	$9.2x10^{-5}$	--	A,PC,CJ	CD
15	BM-11	T-Sand	ISC	$8x10^{-4}$ to $1x10^{-3}$ (2.3 to 3.2)	17 to 24 (183 to 258)	$7.5x10^{-5}$	--	A,PC,CJ	CD
16	Rod Well	T-Sand	ISC	$2x10^{-3}$ to $3x10^{-3}$ (6.7 to 8.9)	50 to 66 (538 to 710)	$9.3x10^{-5}$	--	A,PC,CJ	CD

[a]T-Sand is the uranium-bearing Teapot Sandstone.

[b]DCE = D'Appolonia Consulting Engineers, Inc.; ISC = In-Situ Consulting

[c]Values are based, in part, on the following assumptions: 1) thickness of the Teapot Sandstone is 80 ft, 2) thickness of the Lewis Shale overlying the Teapot Sandstone is 100 ft, and 3) the aquitard underlying the Teapot Sandstone is impermeable.

[d]Tm = Theim steady state (1906); T = Theis (1935); MW = McKee and Way (?); P = Papadopulos (1965); A = Agarwal et al. (1970); PC = Papadopulos and Cooper (1967); CJ = Cooper and Jacob (1946); NJ = Hantush and Jacob (1955).

[e]CD = Constant discharge pumping test

Source: D'Appolonia Consulting Engineers, Inc., 1976 and In-Situ Consulting, 1978a.

horizontal hydraulic conductivity of this aquifer ranges from about $7x10^{-5}$ to $3x10^{-3}$ cm/s (0.2 to 8.5 ft/d); storativity averages about $8x10^{-5}$. A consistent increase in transmissivity from southeast to northwest is indicated by the results of the pumping tests. A directional variation in transmissivity also is indicated; the major direction is north-northeast. It is attributed by RMEC (1979) to secondary (fracture) hydraulic conductivity in the Teapot Sandstone. An apparent barrier boundary was detected during single-well test number 14, which may reflect a decrease in thickness of the aquifer.

Results from the two multiple-well aquifer tests (3 and 4) for which drawdown data are available indicate that water probably leaks into the Teapot Sandstone under pumping conditions. Data curves analyzed by the authors using the Hantush and Jacob (1955) leaky artesian method suggest that the vertical hydraulic conductivity of the Lewis Shale ranges between about $9x10^{-8}$ and $4x10^{-6}$ cm/s ($2x10^{-4}$ to $1x10^{-2}$ ft/d). These values of vertical hydraulic conductivity are representative only for the southern portion of the mine site area.

The confining bed below the Teapot Sandstone is Pumpkin Buttes Shale which ranges from about 80 to 225 m (260 to 750 ft) in thickness, and it is not likely that water would travel through this thick unit in a period of two to seven hours of pumping. The confining bed above the Teapot Sandstone is Lewis Shale which is about 15 to 168 m (50 to 550 ft) thick at the mine site. A shale unit of this thickness probably would not transmit water during two to seven hours of pumping. In addition, no major aquifers are present above the Lewis Shale although perched ground water does occur locally in the unconsolidated deposits overlying the Lewis Shale (NRC, 1980). Therefore, faults, fractures, abandoned boreholes, and/or damaged wells are the probable pathways for vertical leakage through the confining bed(s) (if leakage is occurring). Additional aquifer tests should be conducted at the Nine Mile Lake mine to better define the hydraulic characteristics of the ore zone, the confining layers, and adjacent aquifers.

A.3.5 Hydrogeology of the Mine Site

Uranium mineralization at the Nine Mile Lake site occurs as a roll-front deposit in the Teapot Sandstone of the Mesa Verde Formation. The uranium ore body extends approximately 6100 m (20,000 ft) in a north-northwest direction; the width ranges between 15 and 900 m (50 and 3000 ft). The Teapot Sandstone is between 15 and 24 m (50 and 80 ft) thick in the mine area, and lies 23 to 180 m (75 to 600 ft) below ground surface (NRC, 1980). The sandstone units at the Nine Mile Lake mine site generally were deposited in shallow marine and transitional environments. The sandstones probably represent deltas and barrier islands; the shales represent deltas and deeper marine shelf deposits (Curry, 1976). The units appear to be relatively continuous across the mine site without numerous facies changes and without lenticular sediment bodies.

The ore body within the Teapot Sandstone is up to 24 m (80 ft) thick, and frequently occurs as two stacked ore zones separated by a thin lignite-shale zone (RMEC, 1982a). Uranium precipitation in the roll-front occurred

at the interface of the oxidation-reduction boundaries in the Teapot Sandstone as a result of the presence of carbonaceous material and pyrite. Uraninite has been identified as the principal uranium mineral at Nine Mile Lake; minor amounts of coffinite occur also.

The Nine Mile Lake mine site is located on the west limb of a small syncline. Strata dip approximately six degrees or less to the east-northeast. Fig. A.24 is a plan view of structural features and Cretaceous strata exposed in the mine site vicinity. An anticline lies east of the mine site. Typical cross sections through the mine site are presented in Fig. A.25. The Teapot Sandstone is exposed to the west of the mine site as a prominent, rocky ridge called the "Reefs". No faults have been reported in the vicinity of the mine. The stratigraphic column for the project area is illustrated in Fig. A.26. All indurated strata of interest at the Nine Mile Lake site are of Cretaceous age.

Confined aquifers occur in the Fox Hills, Teapot, and Parkman Sandstone (Fig. A.26) of the Nine Mile Lake site. Unconsolidated alluvial, eolian, lacustrine, and playa deposits contain ground water under unconfined conditions. These unconsolidated deposits up to 18 m (60 ft) thick are present over most of the mine site. Yields of wells tapping the unconsolidated deposits are low because the material is predominantly fine-grained and the saturated portion is thin (RMEC, 1979).

The Fox Hills Sandstone crops out along the eastern edge of the mine site. A few wells tap this unit and reportedly yield approximately 0.5 to 1.0 L/s (7 to 15 gal/min). The Lewis Shale above the Teapot Sandstone produces small amounts of ground water from isolated, discontinuous sand lenses. The Teapot and Parkman Sandstone members of the Mesa Verde Formation crop out immediately west (updip) of the mine site. These two sandstone members are confined aquifers separated by the Pumpkin Buttes Shale. The Cody Shale underlies the Mesa Verde Formation; it constitutes an extensive aquitard in the region (NRC, 1980).

The hydraulic gradient in the Teapot Sandstone in the vicinity of the mine site is 0.0034 from northwest to southeast. Because water level data for strata other than the Teapot Sandstone are not available, vertical ground-water gradients cannot be evaluated.

A.3.6 Monitoring Well System and Well Construction

Monitoring wells completed in the Teapot Sandstone (ore zone) were placed from 30 to 60 m (100 to 200 ft) beyond the perimeter of the research and development well field. Apparently no monitoring wells were completed in aquifers above or below the Teapot Sandstone during the research and development operations. The Fox Hills Sandstone has been removed by erosion over most of the area of the research and development tests. RMEC originally proposed to locate monitoring wells in the Teapot Sandstone no more than 60 m (200 ft) from any commercial-scale well field. Concomitant with trend wells, monitor wells were proposed at distances no greater than 120 m (400 ft) from the well field. Shallow monitoring wells in the Fox Hills Sandstone were proposed within the well field area at a minimum of

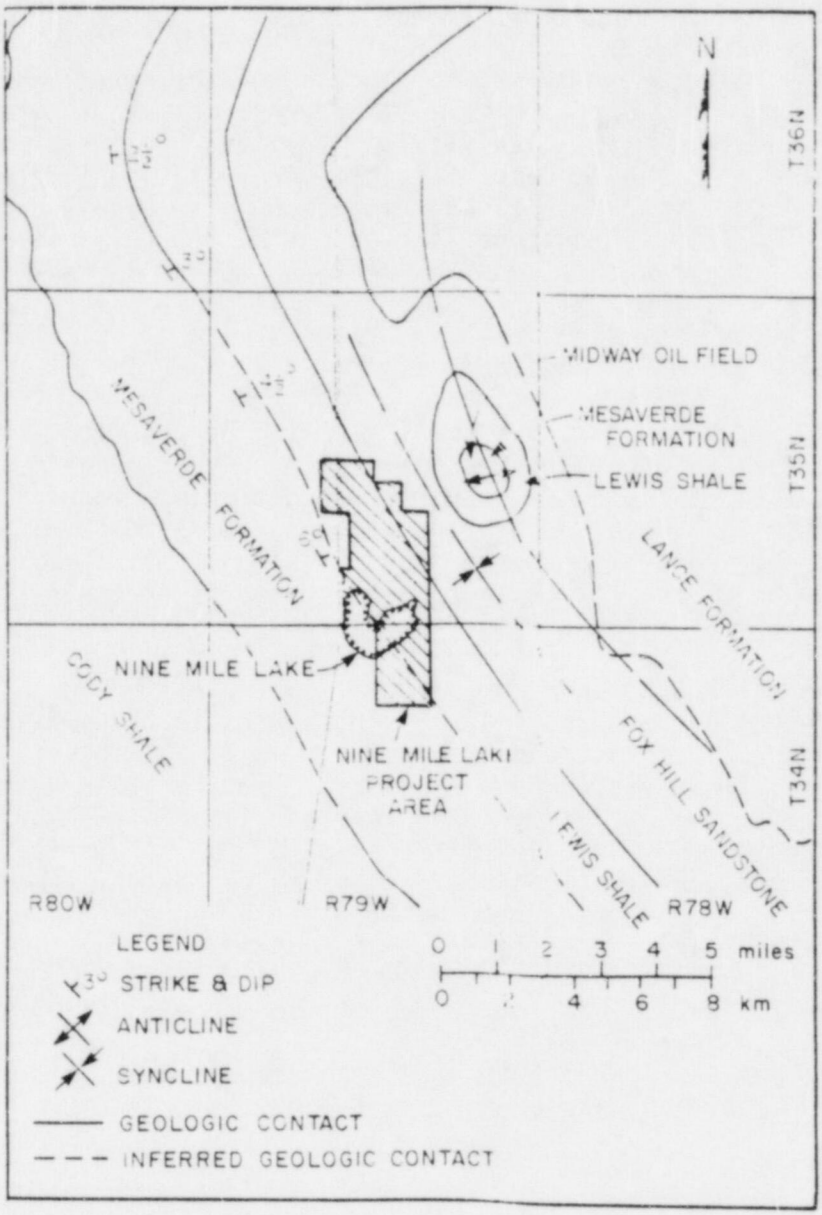


Fig. A.24. Surficial geology and structure map of the Nine Mile Lake project area. *Source:* Modified from NRC, 1980.

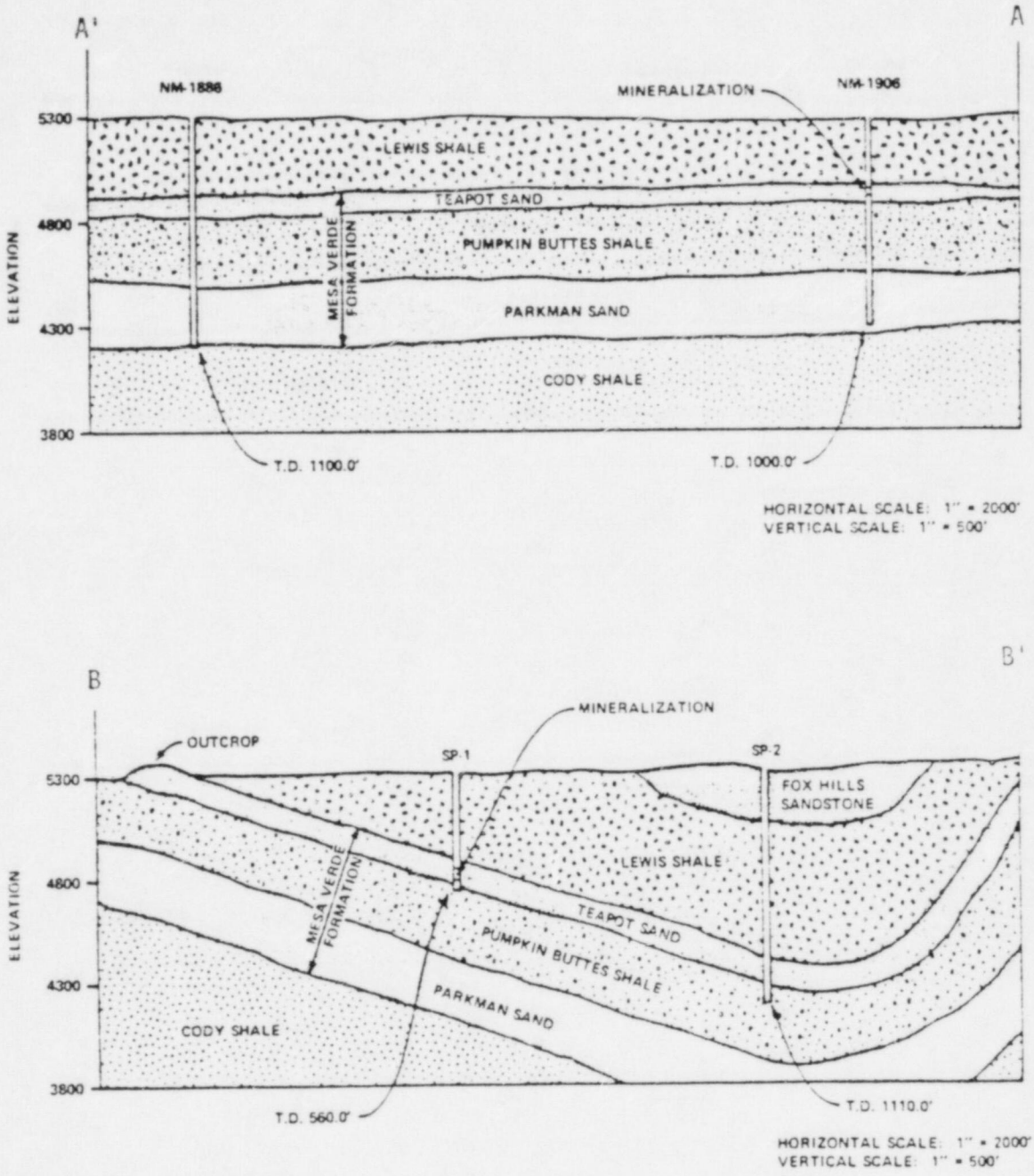


Fig. A.25. Geologic cross sections through the Nine Mile Lake mine site. *Source:* Modified from NRC, 1980.

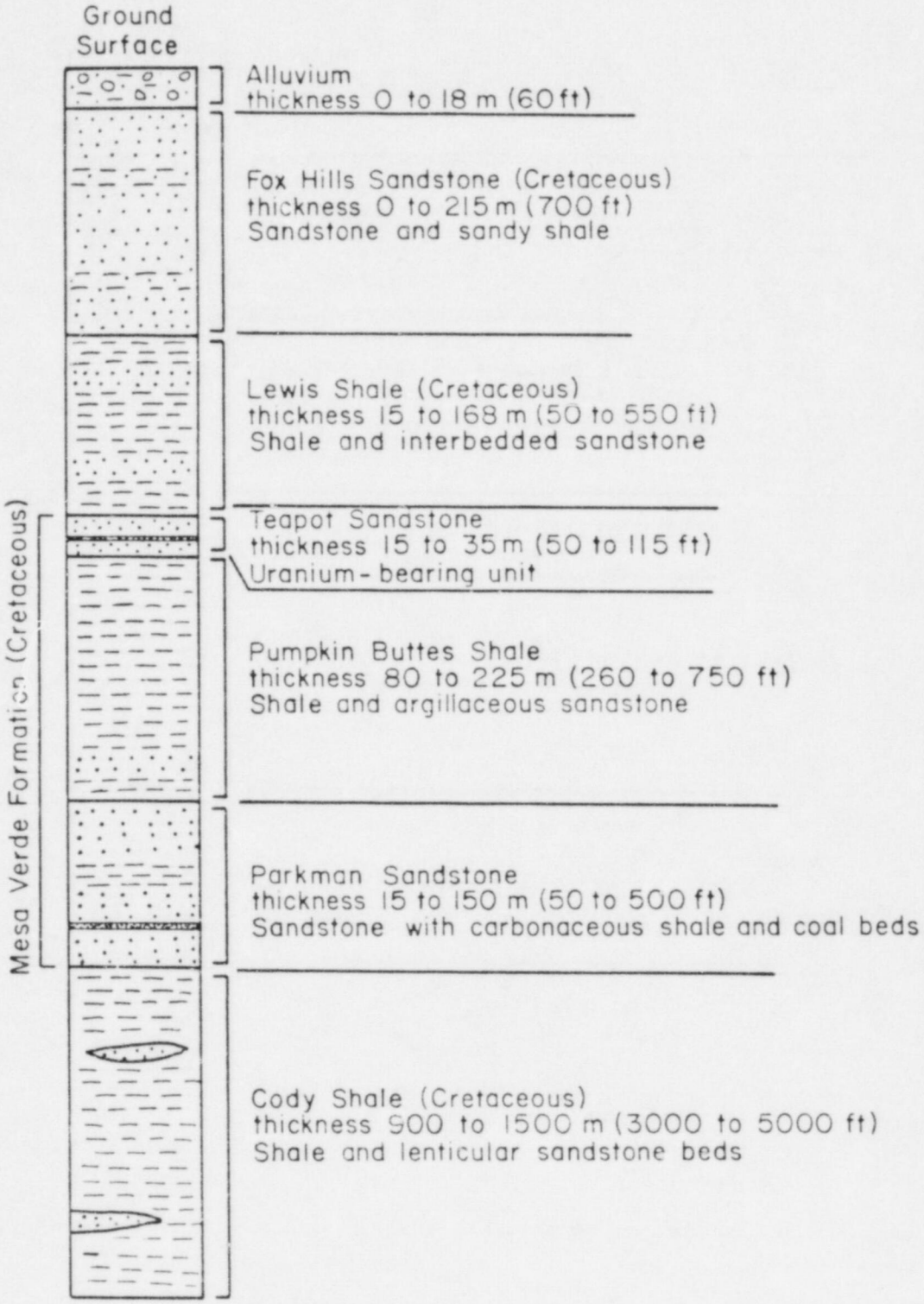


Fig. A.26. Stratigraphic section for the Nine Mile Lake project area. *Source:* Modified from RMEC, 1979.

one well for each 2 ha (5 acres); however, the Fox Hills Sandstone exists only along the very eastern edge of the project area. RMEC proposed no deep monitoring wells because of the absence of an underlying aquifer close to the production zone. Subsequent to these proposals, the decision was made to abandon further plans for commercial development; consequently, the wells were not installed.

Monitoring wells in research and development Patterns 1, 2 and 3 (Fig. A.22) were sampled once each month for the excursion indicators pH, conductivity, sulfate, uranium, and vanadium. These excursion indicators were selected because 1) pH and sulfate identify the presence of the sulfuric acid lixiviant, 2) uranium and vanadium are leaching products, and 3) conductivity is a reasonable index of TDS.

Monitoring wells at research and development Pattern 4 were sampled every two weeks for pH, conductivity, chloride, bicarbonate, uranium, and vanadium. Bicarbonate was a major constituent of the lixiviant. Chloride was a major constituent of the ion exchange process; the pH solution controlled the alkalinity of the process fluids.

UCL's were proposed by RMEC (1979) at the four test patterns by taking the well field average for each indicator, adding two standard deviations from the well field average, and subtracting 10% of this total. A license amendment was requested in 1980 to authorize modified UCL's for uranium and vanadium in Pattern 4 monitoring wells (RMEC, 1980a). The proposed UCL's would be determined by adding 1.0 mg/L to the highest baseline value observed in any of the Pattern 4 monitoring wells during baseline sampling. If a monitoring well sample exceeded the UCL for two indicators, and a second sample taken within 48 hours confirmed two elevated indicators, excursion status for the well would be established.

RMEC (1979) proposed to sample monitoring wells every two weeks for conductivity, pH, uranium, sodium, and sulfate during any future acid leach commercial-scale operation. Reasons for including sodium and deleting vanadium from the list of excursion indicators used during research and development operations are not known. The method for establishing UCL's was not reported. An excursion status would be established in such future commercial operations by a method similar to that used during research and development operations.

Wells at Nine Mile Lake were constructed by drilling a hole initially through the designated sandstone to the underlying confining layer. A gamma-log was recorded to evaluate the grade and thickness of mineralization in holes penetrating the ore-bearing Teapot Sandstone. The entire hole was then cased with PVC plastic pipe and cemented throughout the entire outer annulus. The desired open interval was completed by using a hydraulic jet perforator or a bullet perforator. Well development procedures are not available.

A.3.7 Analysis of Excursions

Five monitoring wells have histories of excursions at the Nine Mile Lake research and development test patterns. Three of the wells (M-40, M-41, M-43) are located in Pattern 3 (Fig. A.22) where sulfuric acid was the tested lixiviant; the remaining two wells (M-56 and M-57) are located in Pattern 4 where leaching utilized a sodium carbonate/bicarbonate lixiviant. All monitoring wells having histories of excursions are completed in the Teapot Sandstone (production ore zone). Table A.13 presents a summary of excursions that have occurred at Nine Mile Lake. The five monitoring wells reported on excursion status are shown on Fig. A.27 No wells have been reported on excursion status since August, 1981. Mining and restoration of all test patterns were completed in November, 1980, and February, 1982, respectively. RMEC has not initiated commercial-scale mining operations at Nine Mile Lake as of June, 1984.

A horizontal excursion was detected initially in monitoring wells M-40 and M-43 located in Pattern 3 on November 15, 1979, based on elevated concentrations of uranium, conductivity, sulfate, and low pH values. Injection in all Pattern 3 wells was curtailed initially while the production rate was increased. All injection and monitoring wells used in Pattern 3 were completed in both ore zones in the Teapot Sandstone. Selective sampling of the upper and lower ore zones in wells M-40 and M-43 indicated that the excursion was confined to the upper ore zone. Injection rates into the upper and lower ore zones became unbalanced because of variations in the hydraulic conductivity of the ore zone, and variations in well efficiencies. The horizontal excursion in the upper ore zone was caused by these conditions (RMEC, 1980b).

From January tnrough March, 1980, injection and production was maintained in the lower ore zone while ground water was pumped from the upper ore zone with no injection. The separate zones were sealed from each other using rubber packers. On March 31, 1980, injection into Pattern 3 was halted, followed by a net production of 0.6 L/s (50 gal/min) from both ore zones until restoration commenced in August, 1981. Well M-40 was taken off excursion status on January 21, 1980. Well M-43 fluctuated on and off excursion status until January, 1981.

Monitoring well M-41 in Pattern 3 exceeded UCL's for conductivity, uranium, and sulfate on July 9, 1981. Consequently, the bleed rate from Pattern 3 was increased from 0.6 to 0.9 L/s (10 to 15 gal/min) with the additional 0.3 L/s (5 gal/min) coming from the upper ore zone. Well M-41 was taken off excursion status in August, 1981, just prior to initiation of restoration.

One of the primary purposes of the Pattern 3 test program was to evaluate the feasibility of dual ore zone production using open injection wells and selectively completed recovery wells. Dual zone mining may be feasible; however, injection/production ratios should be monitored closely for both ore zones. A factor that contributed to the excursion problems in Pattern 3 was the placement of monitoring wells only 30 m (100 ft) from the perimeter of injection wells (RMEC, 1982b). Monitoring wells generally are

Table A.13. Summary of excursions at the Nine Mile Lake mine site during mining operations.

Well	Location	Aquifer	Excursion Parameters[a] Date of Sample[b]	pH	Conductivity (μmhos/cm)	Uranium (mg/L)	Vanadium (mg/L)	Sulfate (mg/L)	Chloride (mg/L)	Bicarb. (mg/L)	Excursion History On	Off	Duration
M-40	Pattern 3	Teapot Sandstone	UCL	6.3	3110	0.055	0.42	2522	--	--	11/15/79	1/21/81	2 months
			11/15/79	--	--	--	--	--	--	--			
			11/21/79	2.6	5500	4.1	--	3558	--	--			
M-43	Pattern 3	Teapot Sandstone	UCL	6.4	3308	0.035	0.28	1266	--	--	11/15/79	1/81	2 months
			11/15/79	--	--	--	--	--	--	--			
			6/09/80	6.9	2500	0.918	0.57	1005	--	--			
M-41	Pattern 3	Teapot Sandstone	UCL	6.3	3404	0.069	0.33	1582	--	--	7/09/81	8/81	1 month
			7/09/81	6.6	3600	0.071	0.065	1798	--	--			
M-56	Pattern 4	Teapot Sandstone	UCL	7.6	3080	0.193	0.37	--	57	373	11/03/80	11/24/80	3 weeks
			11/03/80	6.9	2400	0.303	0.47	--	27	319			
			11/18/80	7.2	2550	0.843	0.96	--	31	283			
M-57	Pattern 4	Teapot Sandstone	UCL	7.8	3218	0.150	0.34	--	61	373	11/03/80	11/16/80	2 weeks
			11/03/80	6.8	2600	0.182	0.38	--	31	311			
			11/15/80	6.9	2800	1.135	1.05	--	--	299			

[a]Excursion indicators for Pattern 3 are pH, conductivity, uranium, vanadium, and sulfate. Excursion indicators for Pattern 4 are pH, conductivity, uranium, vanadium, chloride, and bicarbonate.

[b]UCL = Upper Control Limit; those listed for pH in Pattern 3 are lower control limits; the first sample date is when the well went on excursion status; dates that follow are representative of highest excursion values achieved.

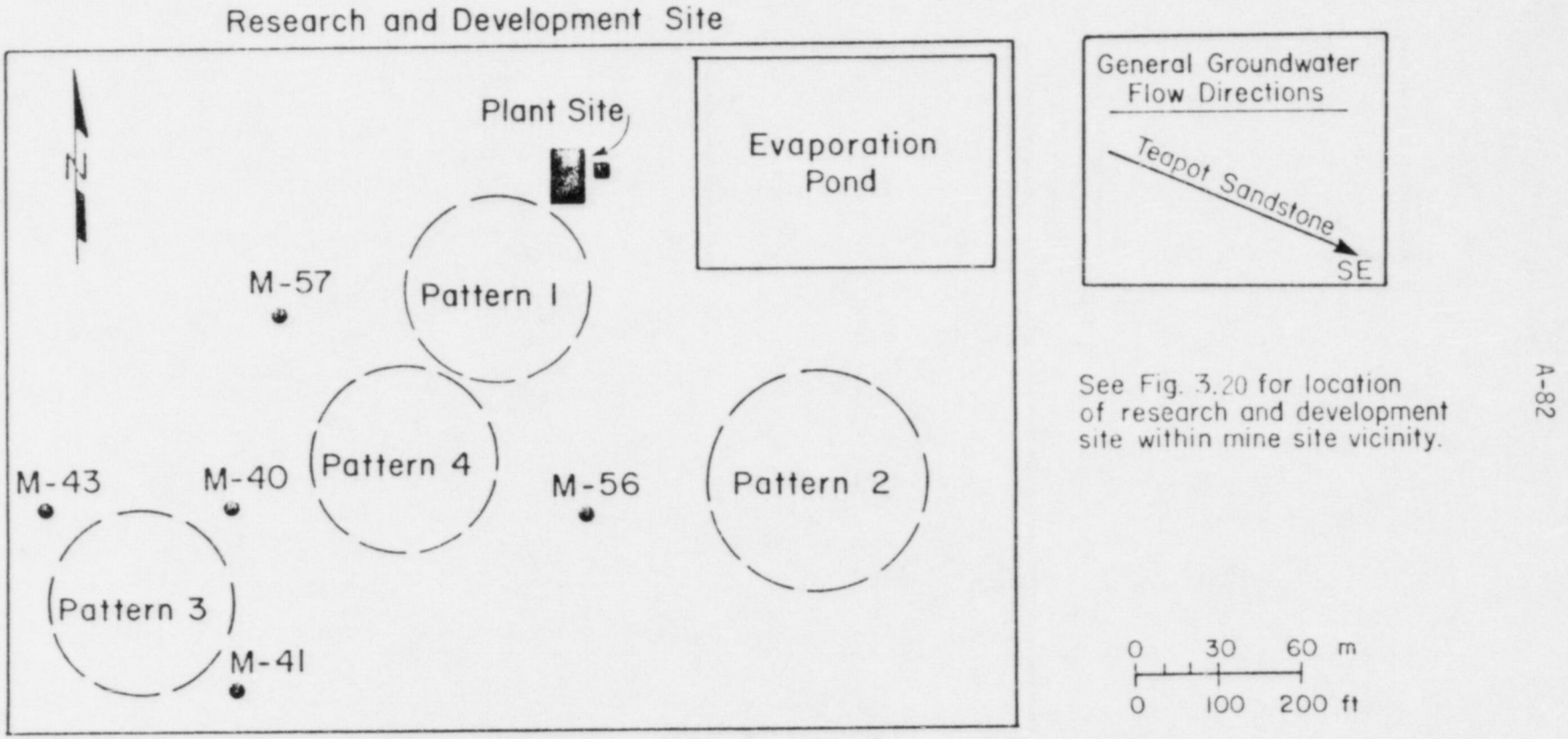


Fig. A.27. Location of monitoring wells reported on excursion status at the Nine Mile Lake research and development site. *Source:* Modified from RMEC, 1982a.

placed at a distance of at least 60 m (200 ft) from the well field, and wells located only 30 m (100 ft) from the well field usually are considered trend wells. RMEC's analysis of horizontal excursions in Pattern 3 appears to be accurate. No overlying or underlying wells were monitored to detect possible vertical lixiviant migration. Consequently, it is impossible to determine whether vertical excursions occurred.

Two monitoring wells (M-56 and M-57) have been on excursion status in Pattern 4 based on elevated concentrations of uranium and vanadium. Excursions in both wells were detected on November 3, 1980, and injection of lixiviant in Pattern 4 was discontinued on November 10, 1980. Corrective action began on November 16, 1980, when overproduction from the pattern was increased from 0.2 to 0.3 L/s (3 to 5 gal/min) while injection of barren production ground water was maintained at 0.9 L/s (15 gal/min). Both wells were taken off excursion status approximately two weeks after the excursion was detected. Apparent excursions in the two Pattern 4 monitoring wells were a result primarily of UCL's for uranium and vanadium that were set too low to account for natural variations. Mining in Patterns 1, 2, and 3 also may have influenced the concentration levels in wells M-56 and M-57 (RMEC, 1980b).

RMEC (1980a) requested a license amendment to modify the UCL's for uranium and vanadium in Pattern 4 monitoring wells. The proposed UCL for uranium and vanadium would be determined by adding 1.0 mg/L to the highest baseline value observed in any of the Pattern 4 monitoring wells during baseline sampling. If the proposed UCL's were in effect during November, 1980, wells M-56 and M-57 would not have been placed on excursion status.

The reported excursions in Pattern 4 may have been the leading edge of a horizontal lixiviant migration. The rapid response to overproduction in the well field tends to support this explanation. However, the absence of elevated indicators other than uranium and vanadium suggests that an excursion may not have occurred. The elevated concentrations of uranium and vanadium in fact may have been due to natural variations only. No overlying or underlying monitoring wells were used in Pattern 4 to detect possible vertical lixiviant excursions.

A.3.8 Restoration

Restoration tests were conducted at all four research and development test patterns at Nine Mile Lake. A complete discussion of restoration attempts at the four test patterns is included in RMEC (1982a). The first restoration test at Nine Mile Lake was performed on Pattern 1, a seven-spot pattern that utilized sulfuric acid for leaching from November, 1976, to August, 1977. Pattern 1 restoration consisted initially of a ground-water sweep which began in September, 1977, and continued intermittently for 13 months. Approximately 12 aquifer pore volumes of affected ground water were withdrawn during this restoration phase and pumped to an evaporation reservoir. At the completion of ground-water sweeping, water quality within the pattern interior had been restored to near baseline ranges except for calcium, sodium, sulfate, chloride, and magnesium. Table A.14

Table A.14. Water quality[a] in wells before and after restoration tests in Pattern 1 at the Nine Mile Lake mine.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Sweep[b] 10/78	Post RO Treatment[c] 8/81	Post Restoration[c] 2/82
Chloride	250	3.3	37	37	93
Sulfate	250	1240	1977	1429	5140
Calcium	--	87	124	80	300
Uranium (U_3O_8)	5.0	0.384	0.063	0.10	0.289
Vanadium	0.1[d]	0.1	0.05	0.09	0.073
TDS	500	2483	3874	2560	7750
Conductivity, (µmhos/cm)	--	3162	--	--	12,000
pH (units)	6.5 - 8.5	6.9	6.3	6.3	6.9

[a]Values are mg/L except as noted.

[b]Values are average from wells P-1A, I-1, I-3, and I-5.

[c]Values are average from wells P-1A, I-1 through I-6.

[d]Agricultural water quality standard.

Note: No values for post-operation/pre-restoration are available.

Source: RMEC, 1982a.

summarizes important parameters at the end of this first restoration phase in Pattern 1.

Pattern 1 remained inactive from the fall of 1978 through the spring of 1981. Occasional sampling during this period indicated substantial water quality deterioration, primarily due to gypsum dissolution within the well field which increased TDS levels (RMEC, 1982a). Reverse osmosis treatment of recirculated ground water was conducted in 1981. Approximately 2.5 aquifer pore volumes were recovered, treated, and reinjected into the aquifer during reverse osmosis treatment. Gradual deterioration of water quality has occurred in the pattern interior since the last restoration effort (Table A.14). Results of the two restoration programs in Pattern 1 have shown temporary improvements in water quality, followed by deterioration and stabilization at a high level of TDS. RMEC (1982a) claims that water quality has stabilized to the degree which does not constitute a degradation of premining water use. However, concentrations of 7750 mg/L for TDS and 5140 mg/L for sulfate indicate that the water is unsuitable for stock purposes (premining water use).

The leaching phase of Pattern 2 using sulfuric acid lasted from December, 1977, through September, 1978. The initial restoration process began in September, 1978, and consisted of a modified ground-water sweep for about four months with reinjection of process water and barren production fluid. Restoration using reverse osmosis treatment then continued for about one month. From May to mid-August, 1979, injection of a high pH, sodium hydroxide solution was maintained to promote ion exchange reactions and speed restoration. Clean water recycle using reverse osmosis treatment continued again for approximately three weeks. A total of 11.5 aquifer pore volumes were produced since restoration began in September, 1978. Nearly all major parameters were restored to premining ranges during this initial restoration program in Pattern 2; however, monitoring during late 1979 and early 1980 detected scattered areas of contaminated ground water around the pattern interior, migrating slowly downdip.

Pumping in Pattern 2 was resumed in August, 1980, in an injection well located southeast of the well pattern, and continued for three months. The pumped ground water was treated using a lime/barium chloride precipitation process and reinjected into the well field. Total production was about 3.5 aquifer pore volumes. Little improvement was indicated within the pattern interior and by May, 1981, some deterioration of water quality was recurring. A new well was drilled southeast of the well field and pumped for about two months to draw affected ground water back toward the Pattern 2 interior. Production was resumed again in November, 1981, in an attempt to collect any remaining unrestored ground water and to prevent interference with Pattern 3 restoration efforts. Restoration of Pattern 2 was terminated on February, 1982, with a total of nearly 18 aquifer pore volumes of ground water produced during the entire restoration phase. Table A.15 presents concentrations of major constituents before, during, and after restoration efforts at Pattern 2 up to February, 1982. Water collected from interior wells after completion of restoration in February, 1982, indicate that at that time all major constituents, except ^{226}Ra, aluminum, vanadium, zinc, iron, silicon, and TDS, has been restored to near

Table A.15. Water quality[a] in wells before, during, and after restoration in Pattern 2 at the Nine Mile Lake mine.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Postleaching/ Prerestoration[b]	During Restoration[b] 9/04/79	During Restoration[b] 2/04/81	Post Restoration[b] 2/82
Chloride	250	46	--	--	--	37
Sulfate	250	1769	5745	1380	2017	1584
Calcium	--	111	128	60	136	102
Uranium (U_3O_8)	5.0	0.239	43	0.6	0.38	0.132
Vanadium	0.1[c]	0.05	430	11.4	3.8	0.986
TDS	500	2852	6750	2360	2960	2390
Conductivity, (µmhos/cm)	--	3339	--	--	--	3080
pH (units)	6.5 - 8.5	6.7	1.5	6.1	5.8	6.9

[a]Values are mg/L except as noted.

[b]Values are average from wells P-15, I-16A, I-17, I-18, and I-19.

[c]Agricultural water quality standard.

Source: RMEC, 1982a.

baseline levels. Vanadium, ^{226}Ra and TDS remained above baseline as of June, 1984.

Pattern 3 tests the feasibility of simultaneous leaching in both upper and lower ore zones in the Teapot Sandstone from September, 1979, to April, 1980. Restoration of the well field was conducted from August, 1981, through January, 1982, by simultaneous injection of clean water in wells located outside the well field and pumping affected ground water from the pattern interior. Production fluid was routed through a reverse osmosis unit, then combined with "clean" formation water, and reinjected into the production zone. Nearly six aquifer pore volumes of ground water were produced from Pattern 3 during restoration. Water quality in Pattern 3 has been restored to pattern baseline ranges or better, except uranium, vanadium, and ^{226}Ra in some interior wells (RMEC, 1982a). A summary of water quality before and after restoration of Pattern 3 is presented in Table A.16.

Pattern 4 tested the use of a sodium carbonate/bicarbonate lixiviant from June to November, 1980. Information concerning the restoration process used from November, 1980, to August, 1981, is not available. RMEC intended to restore Pattern 4 using a clean water recycle method. Contaminated ground water pumped from the well field would pass through a reverse osmosis unit, followed by reinjection into the well field. It is not known whether this process actually was used at Pattern 4. RMEC (1982a) claimed that post-restoration monitoring indicated that all parameters, except vanadium, were within pattern baseline ranges.

Based on water quality data presented by RMEC (1982a), restoration of the well patterns (1, 2, 3) that used a sulfuric acid lixiviant generally was successful in reducing major parameters temporarily to approximately baseline levels. Post-restoration monitoring indicates some deterioration of water quality with high levels of sulfate and TDS. Acidic lixiviants tend to dissolve other trace elements present in the ore such as aluminum, iron, copper, zinc, zirconium, selenium, arsenic, vanadium, and molybdenum along with uranium. Basic lixiviants are preferred for the leaching of moderate to high carbonate ores because such ore zones consume substantial quantities of an acidic lixiviant. Direct comparison of the restoration results from Patterns 1, 2, and 3 (sulfuric acid lixiviants) and Pattern 4 (sodium carbonate lixiviant) is not possible due to the lack of information regarding Pattern 4. However, two major potential problems with the use of acid lixiviants are noteworthy with respect to restoration at the Nine Mile Lake site. These are: 1) precipitation of gypsum ($CaSO_4$) on well screens and within the aquifer during mining plugged the wells and lowered the hydraulic conductivity of the formation, and 2) gradual dissolution of the precipitated gypsum (and release of trace elements incorporated in the gypsum) caused the water quality to deteriorate after restoration.

High sulfate concentration also may indicate leaching fluid that was uncaptured by the ground-water sweeping or water from Nine Mile Lake itself which is naturally high in sulfate. The lake is located approximately 1.6 km (1 mi) from the pilot project. The Teapot Sandstone (Mesa Verde Formation) underlies the eastern portion of the lake (Fig. A.24). Therefore, a

Table A.16. Water quality[a] in wells before and after restoration in Pattern 3 at the Nine Mile Lake mine.

Paramcter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline	Postleaching/ Prerestoration[b] 8/27/81	Post Restoration[c] 2/82
Chloride	250	35	31	26
Sulfate	250	1244	1168	920
Calcium	--	74	66	61
Uranium (U_3O_8)	5.0	0.060	0.585	0.126
Vanadium	0.1[d]	0.18	5.52	0.57
TDS	500	2034	1950	1450
Conductivity, (μmhos/cm)	--	2381	2450	2500
pH (units)	6.5 - 8.5	6.9	4.9	7.1

[a]Values are mg/L except as noted.

[b]Values are average from production wells 50 and 53.

[c]Values are average from wells 50, 53, I-44 through I-49.

[d]Agricultural water quality standard.

Source: RMEC, 1982a.

ground-water sweep may draw lake water high in sulfate into Teapot Sandstone.

A.4 EXXON MINERALS COMPANY - HIGHLAND PROJECT

A.4.1 Description of the Site

The Exxon Minerals Company (EMC) Highland mine site is located in central Converse County, about 40 km (25 mi) north of Douglas, Wyoming (Fig. A.28). The site is in the southern portion of the Powder River Basin. Conventional open-pit and underground uranium mining and milling operations have been conducted at the Highland site since 1972. Two pilot-scale solution mining test facilities also are located at the Highland mine site. The Highland mine was purchased recently by Everest Minerals Company which plans to conduct commercial mining operations in the future.

The topography of the site is characterized by gently rolling upland areas and dissected stream valleys; elevations range from about 1550 to 1600 m (5100 to 5450 ft). The North Fork of Box Creek is the only stream within the Highland site boundaries. It is an ephemeral stream that drains toward the Powder River. The Powder River is located about 116 km (70 mi) east of the site. The average annual precipitation at Bill, Wyoming, located about 20 km (12 mi) from the Highland mine site, is 30.5 cm (12 in).

Research and development testing at the original pilot-scale test site began in March, 1972. Testing was continued at a second test site (herein called the expanded pilot test) beginning in December, 1978 (Fig. A.29). A sodium carbonate/bicarbonate lixiviant was used in both tests. Although EMC has been issued permits to solution mine uranium commercially at the Highland mine site, only pilot tests have taken place.

A.4.2 History of Mining Operations

Pilot operations at the original test well field utilized a lixiviant consisting of sodium carbonate/bicarbonate, with oxygen as the oxidizing agent. Table A.17 presents basic characteristics of the lixiviants used at the site. The original test well field consisted of a single inverted seven-spot well pattern with a central injection well surrounded by six production wells (Fig. A.30). The production wells are located 27 m (90 ft) from the injection well (EMC, 1979a); all wells are completed in the Middle Highland Sandstone. Injection of lixiviant at the original pilot test site began in March, 1972. During the test, approximately 4.35×10^6 L (1.15×10^6 gal) of lixiviant were injected at pressures up to 3.4×10^5 N/m^2 (50 psi). The production wells were pumped at about 0.6 to 0.7 L/s (10 to 12 gal/min) and the uranium-bearing solution was fed through two ion exchange units for removal of uranium. After the uranium was removed, the barren solution (eluate) was refortified with lixiviant and reinjected into the ore zone.

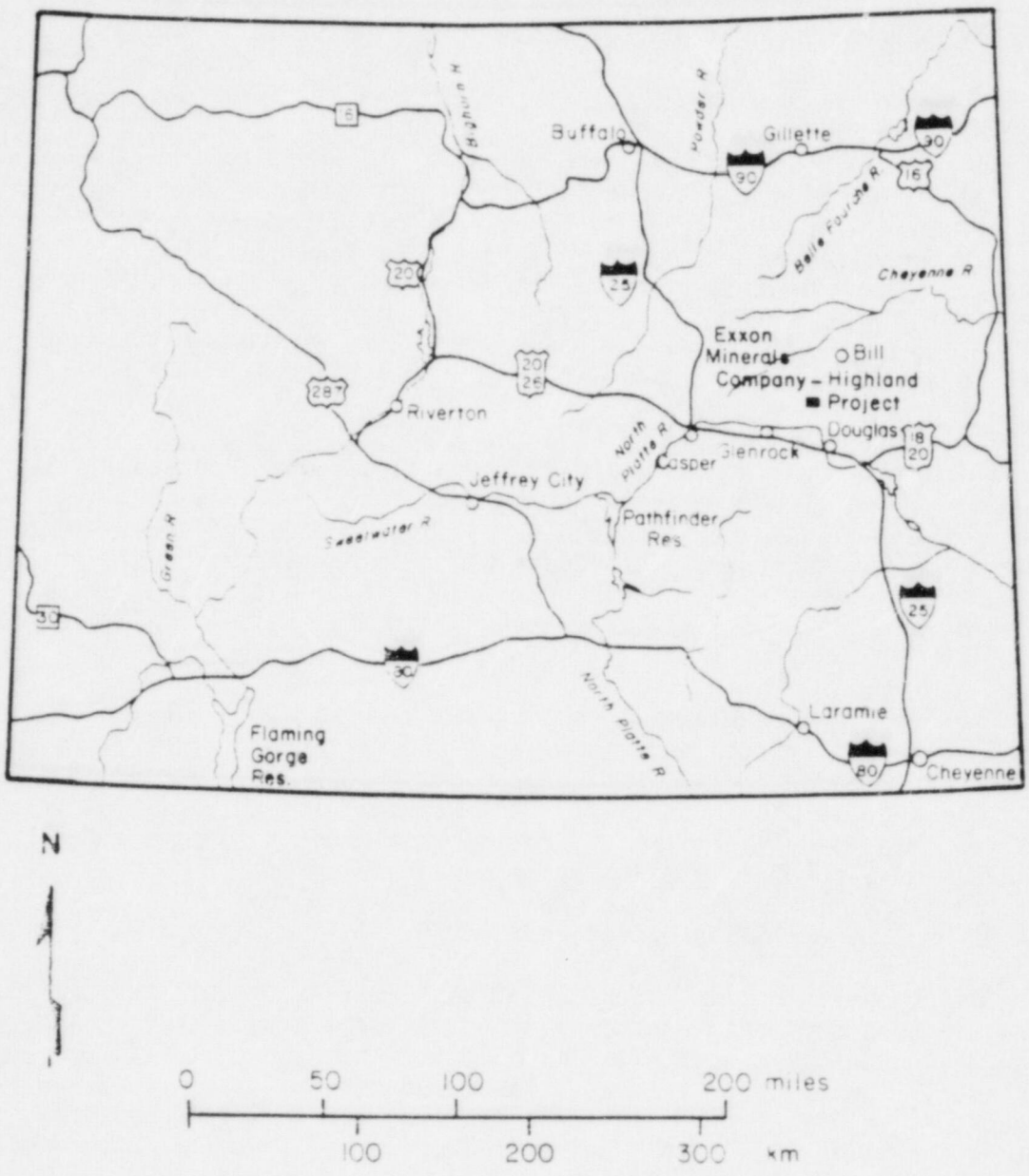


Fig. A.28. Location map of the Exxon Minerals Company Highland mine.

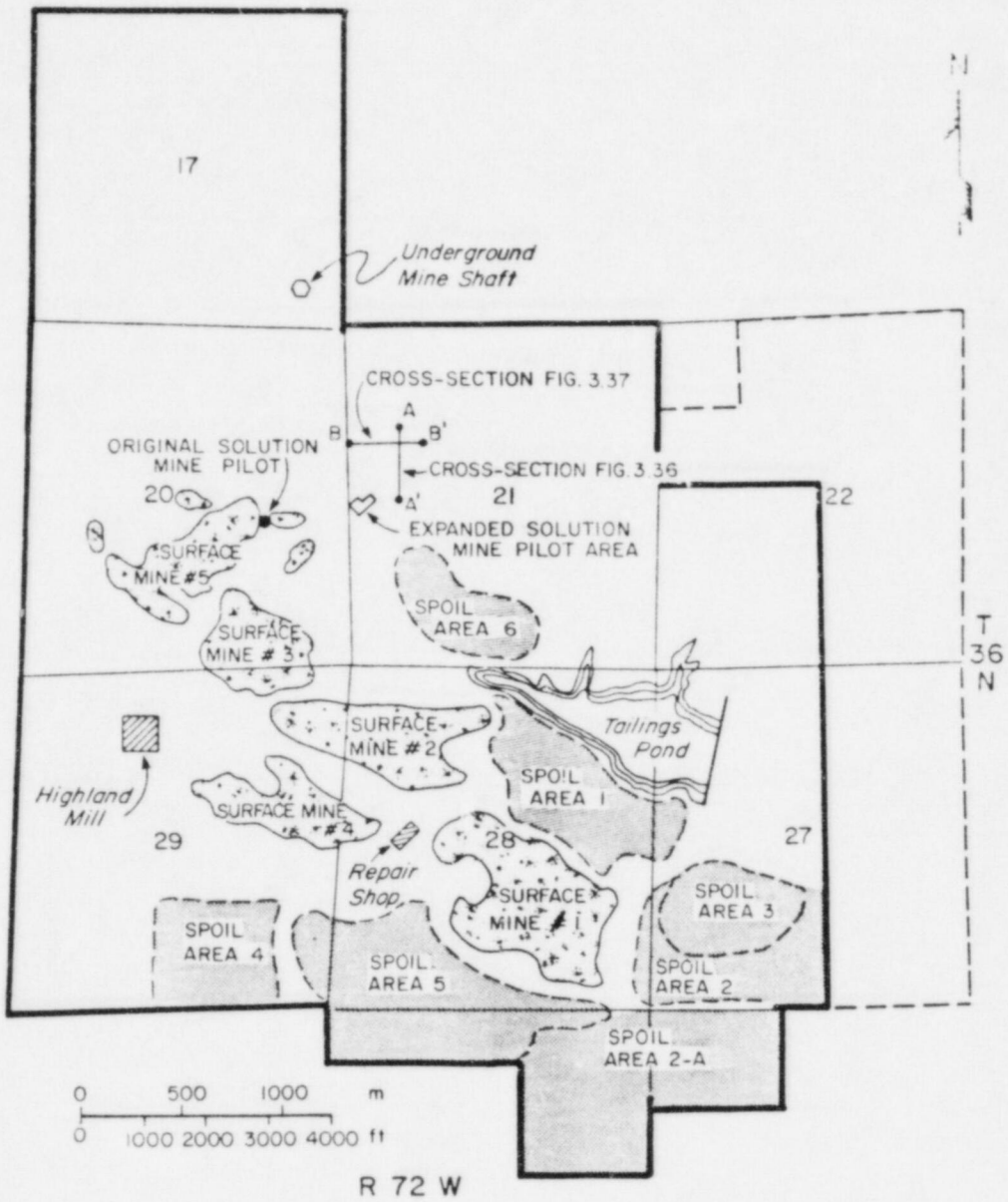


Fig. A.29. Map of the Highland mine operations. *Source:* Modified from EMC, 1980a.

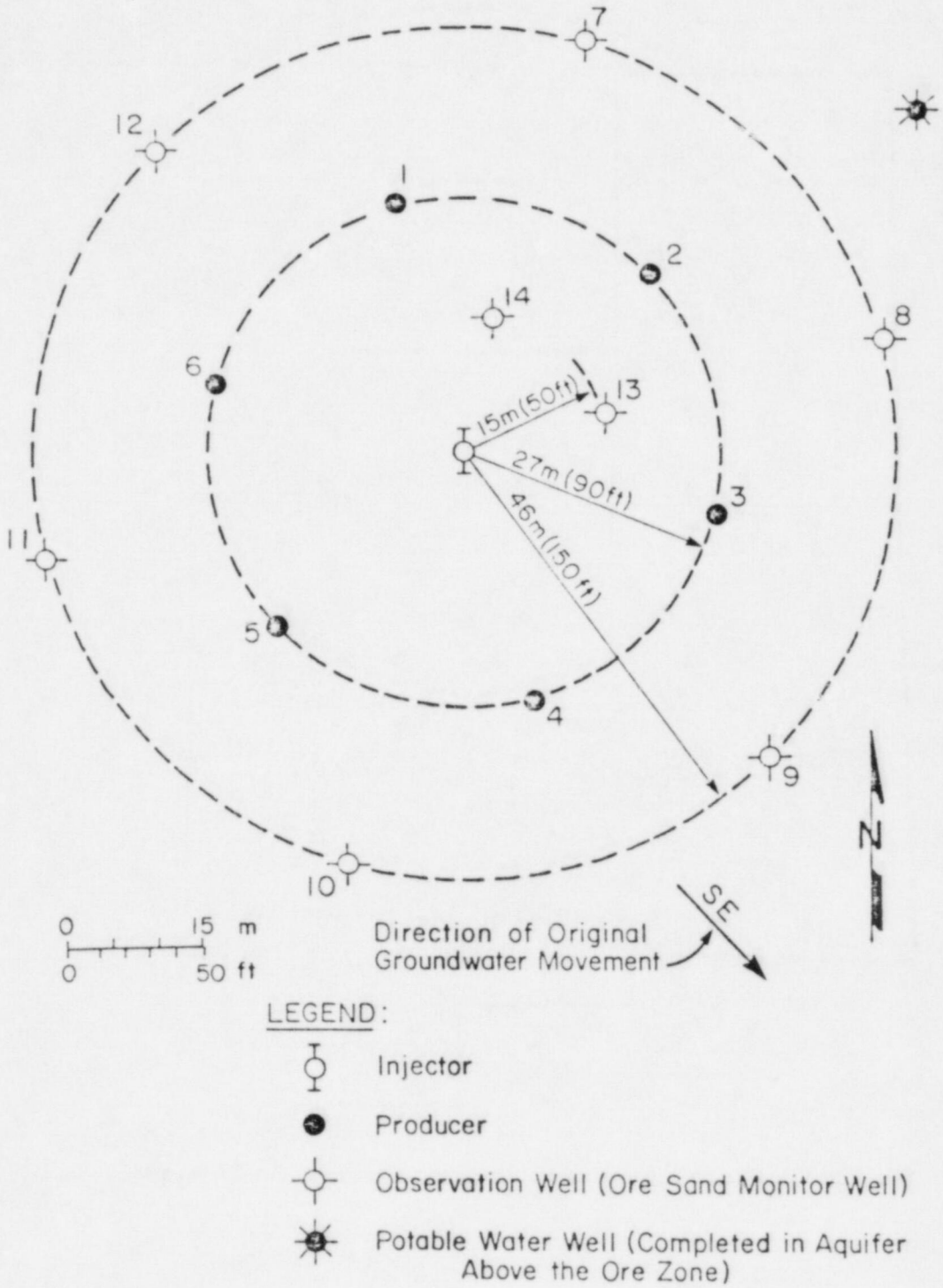


Fig. A.30. Location map of wells at the original pilot solution mine field of the Highland site. *Source:* Modified from EMC, 1979a.

Table A.17. Lixiviant compositions at the Highland Mine.

Constituent mg/L	Original	Expanded
Sulfate	100-300	100-300
Calcium	50-300	50-300
Sodium	400-800	400-800
Chloride	100-800	100-800
Hydrogen Peroxide (g/L)		0.5-1.0
Carbonate + Bicarbonate (g/L)	1.0-25.6	0.5-2.5
pH (units)	6.5-10	6-8

Source: EMC, 1978a.

Uranium was stripped from the ion exchange units by a two step elution process utilizing ammonium nitrate and nitric acid. The solution was then fed to a tank where the addition of ammonia caused the uranium to precipitate. The ammonium diuranate slurry was then transported to the Highland uranium mill for drying and packaging.

Injection of lixiviant was discontinued in November, 1974, although production of uranium-bearing solution from the well field continued until April, 1975. The amount of fluid injected exceeded the amount of fluid produced during the lixiviant injection period of March, 1972, through October, 1974 (NRC, 1982b). This generally is not considered to be an acceptable practice. Maintaining an injection/production ratio of less than 1, thereby producing a hydraulic gradient toward the mining pattern, is a fundamental operating procedure which is necessary to contain mining solutions within the well field. Failure to create such a situation may have been a primary reason for the excursion occurrences detailed in Section A.4.7.

Operations at the expanded pilot test site, located about 460 m (1500 ft) east of the original test site, commenced in November, 1978 (Fig. A.31). The expanded test site consists of four contiguous, inverted five-spot well patterns in an L-shaped configuration. Each five-spot pattern has a central injection well surrounded by four production wells located 21 m (70 ft) from the injection well. These wells are completed in the Lower Highland Sandstone.

Premining aquifer conditioning was performed at the site. This consisted of pumping the production wells, passing the produced water through a water softening system to reduce the calcium content, and reinjecting the water into the ore zone through the injection wells. This procedure was performed to reduce the precipitation of calcite and gypsum within the aquifer during mining, thereby reducing the potential for changes in the hydraulic conductivity of the aquifer. Lixiviant containing up to 2.5 g/L

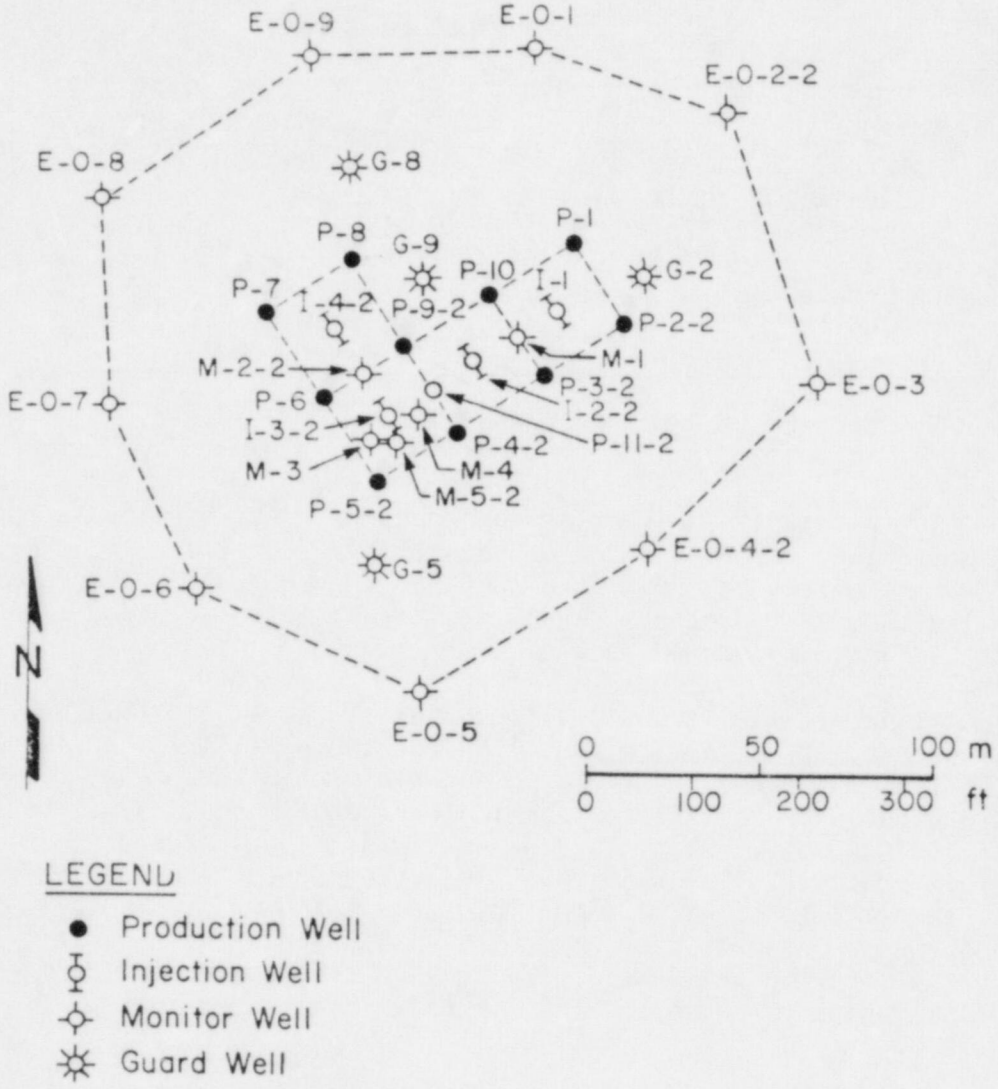


Fig. A.31. Location map of wells at the expanded solution mine well field of the Highland site. *Source:* Modified from EMC, 1982a.

of sodium carbonate and sodium bicarbonate, with up to 1 g/L of hydrogen peroxide as an oxidizing agent, was injected at the expanded pilot test site beginning in December, 1978. Fluid production rates averaged about 3.8 L/s (60 gal/min) during the production phase of the test. Recovery of uranium from the produced solution was by ion exchange in the same fashion as in the original pilot test. Injection of lixiviant was terminated in September, 1981. EMC continued to pump the production wells, recover uranium, and reinject treated water into the ore zone until May, 1983. Since that time, a ground-water sweep restoration program has been conducted with no reinjection.

A.4.3 Regional Setting

A.4.3.1 Regional Geology

The Highland site is located on the eastern flank and near the southern tip of the Powder River Basin approximately 117 km (73 mi) south of the Irigaray site (Fig. A.5). The Eocene Wasatch Formation crops out in the area; the Paleocene Fort Union Formation comprises the subsurface strata to a depth of at least 1000 m. The Lance Formation of Cretaceous age underlies the Fort Union Formation. These formations dip gently to the southwest in the vicinity of the mine site. The Wasatch Formation consists of cyclic sequences of sandstones, siltstones, claystones, and thin coal seams. The Fort Union Formation consists of up to 1040 m (3400 ft) of fine- to coarse-grained channel sandstones, siltstones, claystones, and coal beds (NRC, 1978b). The channel sandstones vary from loose friable sands to highly cemented (concretionary) sandstones. Outcrops of the Lance Formation in the region are thin bedded, brown to gray sandstones and shales (Humble Oil and Refining Company, 1970). The reader is referred to Sharp and Gibbon (1954) for a detailed description of the geology of the southern Powder River Basin.

A.4.3.2 Regional Ground-water Flow Systems

Hodson et al. (1973) present a general description of the hydrostratigraphic units in the Powder River Basin. Much of the following discussion is derived from this reference and a site investigation report by Exxon Production Research Company (1982). Water bearing units in the vicinity of the Highland site include parts of the Wasatch Formation, Fort Union Formation, Quaternary alluvium, and upper Cretaceous rock located more than 1000 m below land surface.

Regional ground-water flow in the upper Cretaceous Parkman Sandstone is to the northeast in the Highland area. Exxon Production Research Company (1982) reported that ground-water flow in overlying formations probably would be similar. Ground-water recharge occurs from runoff and snowmelt that infiltrate into the highly transmissive Quaternary deposits underlying many stream channels in the area, and into permeable strata exposed in areas of higher relief. The Highland uranium mill tailings pond is located hydraulically downgradient from the in situ uranium mine test sites.

A.4.4 Description and Analysis of Data Collected

A.4.4.1 Exploration

Over 17,000 exploration boreholes reportedly were drilled in the Highland project area to delineate the extent of the ore zone. Humble Oil and Refining Company (1970) reported that electric geophysical logs were recorded in wells constructed to obtain stratigraphic data. No information is available in NRC docket files concerning well logging, plugging, and abandonment procedures at the Highland mine site.

A.4.4.2 Aquifer Tests

EMC conducted three aquifer tests in the pilot well fields (Fig. A.29) at the Highland site. Well injection tests (test 1) were conducted in wells at the original pilot project (Fig. A.30) and two multiple-well pumping tests (tests 2 and 3) were conducted at the expanded pilot project (Fig. A.31). In addition, a multiple-well pumping test (test 4) was conducted in Section 24, T36N, R73W at a site where commercial solution mining was being evaluated. These tests were conducted to demonstrate 1) hydraulic connection between injection and production wells in the ore zone, 2) hydraulic connection between the production area (well fields) and monitor wells, and 3) hydraulic confinement of the ore zone.

Core testing and well injection testing were conducted at the original pilot well field to verify that hydraulic communication existed between the injection well and the production wells (Fig. A.30). Well injection testing consisted of pumping fresh water into the ore zone through the injection well while the pressure response over time and the total magnitude of the pressure response were measured at each of the production wells. No data for test 1 are available for review.

The core testing and well injection testing established that hydraulic communication existed between the wells, and that fluids could be displaced between wells in the test pattern. The results of these tests are presented in Table A.18. EMC concluded from these test data that the average sandstone thickness of the Highland units is 7 m (23 ft); average hydraulic conductivity was estimated to be 1×10^{-3} cm/s (2.8 ft/d). Average porosity was estimated to be 29%. Hydraulic conductivity of the underlying and overlying claystone and siltstone units was assumed by EMC to be less than 1×10^{-7} cm/s (2.8×10^{-4} ft/d).

Static water levels were measured in each of the observation and monitor wells at the expanded pilot site (Fig. A.31) in September, 1978. Figs A.32 and A.33 are maps of the potentiometric surface in the Lower Highland Sandstone. A hydraulic gradient of about 0.025 to the northwest (the direction of the underground mine) is indicated by the September 1, 1978, data. The September 7, 1978, data (Fig. A.33) indicate water levels had dropped 0.6 to 1.2 m (2 to 4 ft) in the Lower Highland Sandstone since the earlier measurements were taken. Water levels in well M-1 completed in the Middle Highland Sandstone were about 10.7 m (35 ft) higher than water levels in the Lower Highland Sandstone; water levels in well M-2 completed

Table A.18. Results of core and injection tests conducted at the Highland mine site.

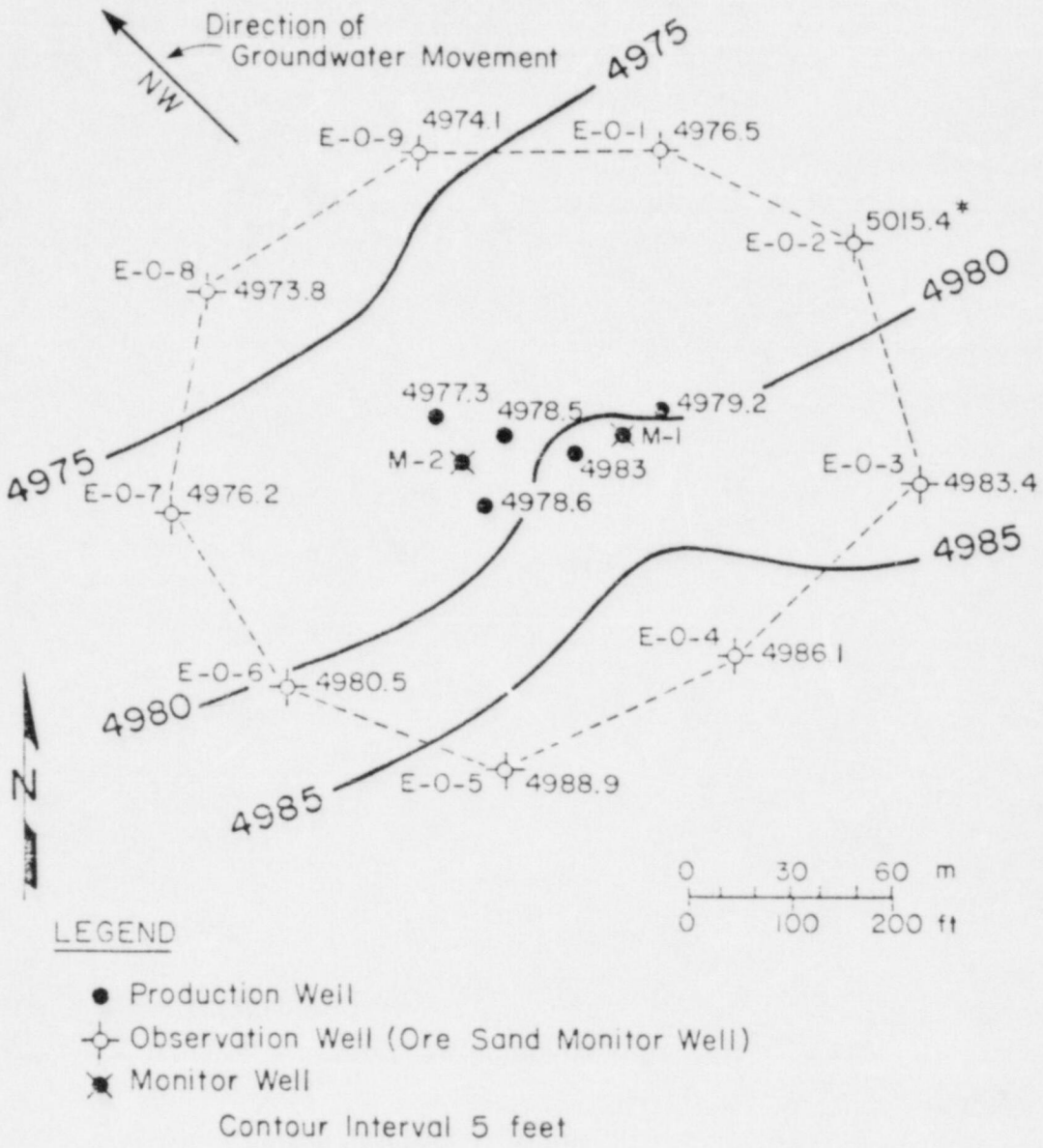
Well Designation	Net Sandstone Thickness m (ft)	Hydraulic Conductivity cm/s	Porosity
I[a]	7 (23)	1.2×10^{-3}	.28
#1	6.7 (22)	1.0×10^{-3}	.30
#2	7 (23	7.8×10^{-4}	--
#3	9.4 (31)	1.3×10^{-3}	.30
#4	7.6 (25)	7.8×10^{-4}	--
#5	4.6 (15)	9.7×10^{-4}	.29
#6	7.3 (24)	1.2×10^{-3}	--
X[b]	39 (129)	5.9×10^{-4}	--
Y[b]	31 (103)	4.4×10^{-4}	--
Z[b]	40 (130)	4.0×10^{-4}	--
Average[b]	7 (23)	1.0×10^{-3}	.29

[a]Injection well, no number designation.

[b]Unidentified.

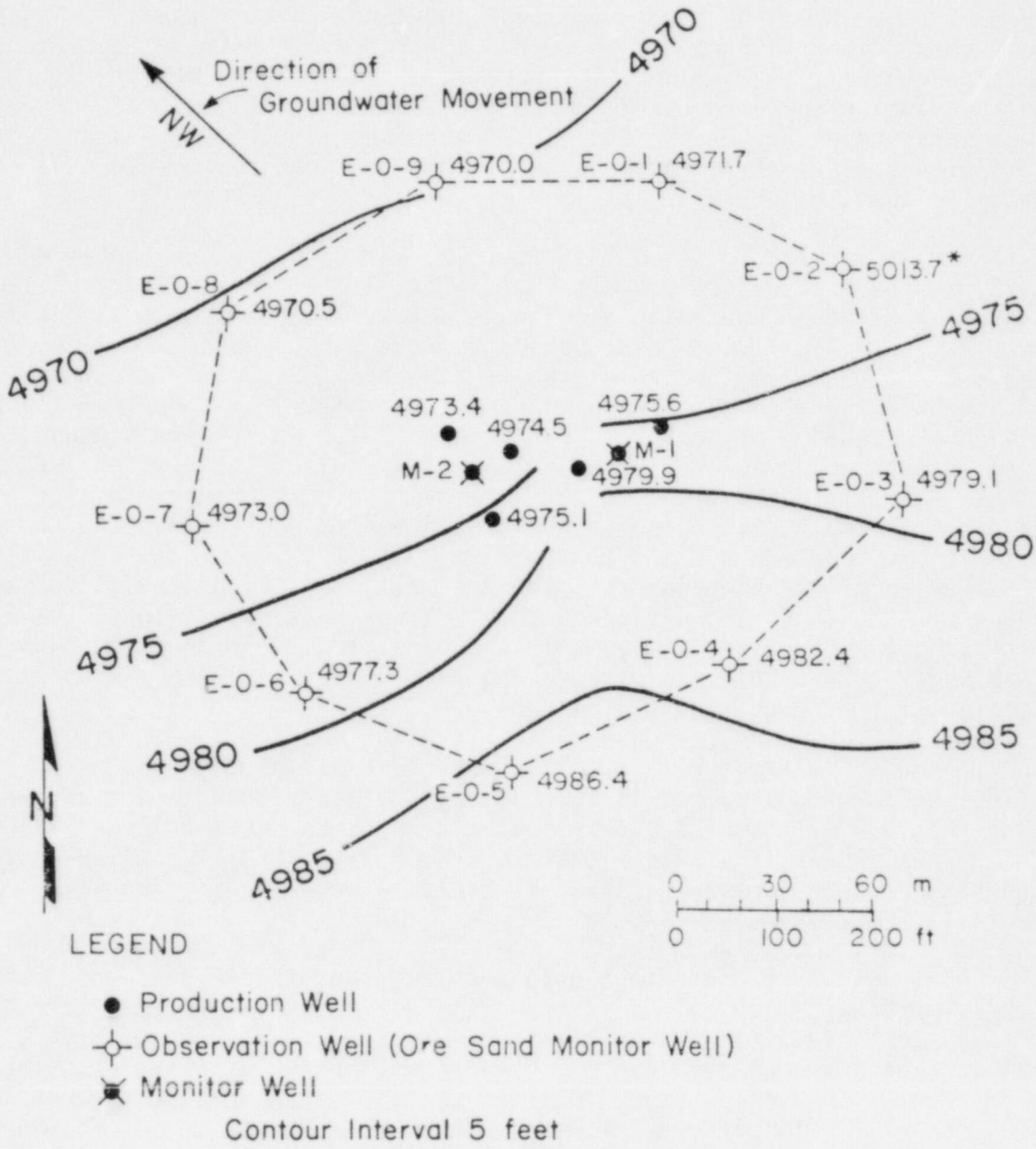
[c]Not including wells X, Y, and Z.

Source: EMC, 1978a.



*Well O-2 completion was apparently in communication with the overlying zone. The well was reworked; therefore, it is not included in the contours.

Fig. A.32. Water level contour map for September 1, 1978, at the Highland mine. *Source:* Modified from EMC, 1978b.



*Well O-2 completion was apparently in communication with the overlying zone. The well was reworked; therefore, it is not included in the contours.

Fig. A.33. Water level contour map for September 7, 1978, at the Highland mine. *Source:* Modified from EMC, 1978b.

in a sand underlying the Lower Highland Sandstone were about 1.2 m (4 ft) lower than that of the Lower Highland Sandstone. Water levels fluctuated less than 0.15 m (0.5 ft) in these two monitor wells between September 1 and September 7, 1978. The difference in hydraulic head measured in each aquifer was interpreted by EMC to suggest isolation of the ore zone aquifer from surrounding aquifers. The fact that potentiometric levels are lower in aquifers at greater depth indicates that any vertical movement of ground water is downward at the site.

A multiple-well pumping test (test 2) was conducted on September 7, 1978, at the expanded pilot well field (Fig. A.31). Well P-9 completed in the ore zone (the Lower Highland Sandstone) was pumped at a constant rate of 0.3 L/s (4.8 gal/min) for a period of five days. Water levels were measured in 15 wells throughout the test period. Drawdowns of 1.0 to 1.7 m (3.4 to 5.7 ft) were observed after about two days in all observation wells with the exception of wells E-0-2, M-1, and M-2. Water levels began to rise in each observation well after about two days. These water level rises were attributed to water level recovery due to changes in mine dewatering pumping at adjacent underground and open-pit mines.

Review of the pumping test data and well completion details for well E-0-2 by EMC led to the discovery that the well was open in both the overlying aquifer (Middle Highland Sandstone) and the production aquifer. This well subsequently was plugged and redrilled to isolate the open section in the production aquifer only.

A second multiple-well pumping test (test 3) was conducted in October, 1978. Well P-9 was pumped at 1.3 L/s (20 gal/min), producing drawdowns of more than 3.7 m (12.0 ft) after about six hours in observation wells E-0-1, E-0-2, and E-0-3. The water level rose 0.03 m (0.1 ft) in well M-1 over the same period. Drawdown data for this test were insufficient for analysis (EMC, 1980a).

Analysis of the pump test data for test 2 by the Cooper and Jacob (1946) method yielded values of transmissivity ranging from 3.2 to 6.4 m^2/d (34.6 to 69.0 ft^2/d) for the Lower Highland Sandstone aquifer. The lack of drawdown data early in the test together with the fact that the condition of the wells (i.e., fully or partially penetrating) is not known prevents evaluation of leakage using either the Hantush and Jacob (1955) leaky artesian method or the Hantush (1960) modified leaky artesian method.

In addition to the lack of early drawdown data, several potentially significant factors were overlooked in the analysis of the pumping tests conducted at the expanded pilot well field. Antecedent water level measurements and barometric pressure measurements were either not taken or not reported. EMC (1978b) indicated that the water level rises observed in the observation wells during the second half of the test period probably were due to changes in mine dewatering operation.

Test 4 was conducted in May, 1979, in Section 24 of the Highland site; this area has been identified for potential commercial in situ uranium

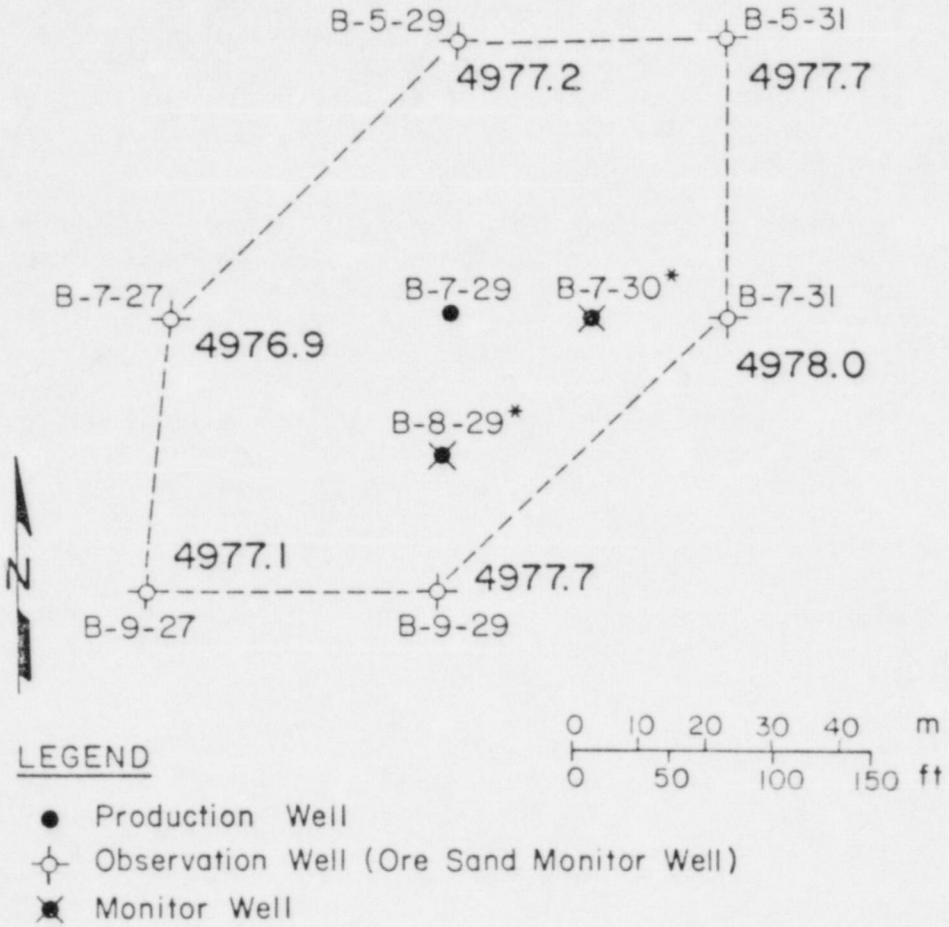
mining (Fig. A.34). In this area, the ore zone occurs in the Lower and Middle Highland Sandstones which are connected due to channel scouring of the interbedded claystone unit. The testing began on May 24, 1979, with the pumping of well B-7-29 (completed in the ore zone) at approximately 0.35 L/s (5.6 gal/min). Water levels were measured in six monitor wells completed in the production aquifer and in wells B-7-30 and B-8-29 which are completed in overlying and underlying aquifers, respectively. The test was conducted for 48 hours.

EMC (1980b) analyzed the pump test data by the Jacob straight line method (Cooper and Jacob, 1946) and estimated aquifer characteristics to be as follows: transmissivity ranges from 3.2 to 4.3 m^2/d (35 to 46 ft^2/d) and storativity ranges from $6.6x10^{-5}$ to $1.9x10^{-4}$. Water levels in the overlying aquifer, a 1.5 to 3 m (5 to 10 ft) thick sandstone separated from the ore zone by about 6 m (20 ft) of shale, rose steadily during and after the test. According to EMC (1980b), water levels had never really recovered from drawdown experienced during well development. The difference in hydrostatic heads between the ore zone and overlying aquifers led EMC to conclude that the ore zone is isolated from the aquifer above it. Water levels in the underlying aquifer declined about 0.15 m (0.5 ft) by the end of the pumping test, and continued to decline even after the water level in the pumped well essentially recovered to pretest conditions. EMC explained this behavior as indicative of fluctuations in nearby open-pit mine dewatering activities.

The methods of analysis [Theis (1935) non-equilibrium method and Cooper and Jacob (1946) straight line plot] utilized by EMC to analyze the multiple-well pumping test data do not allow analysis of the hydraulic properties of the confining units surrounding the production aquifer.

Analysis of the test data by the authors using the Hantush (1960) modified leaky artesian method suggests that water was supplied to the aquifer by compression of surrounding aquitards during the pump test. A lack of early drawdown data, however, prevents a unique match of the pump test data to the Hantush (1960) type curves. This prevents the determination of unique values for transmissivity and storativity. Values of B (leakage function) were found generally to range between 1.0 and zero. The values of B for the pump test data suggest that leakage into the aquifer does occur under pumping conditions. However, the Hantush (1960) modified leaky artesian method of analysis does not allow direct calculation of vertical hydraulic conductivity of the confining units nor does it divide leakage into separate components from the overlying unit and the underlying unit. The Hantush (1960) modified leaky artesian method was used by the authors to calculate the hydrogeologic characteristics of the aquifer and upper aquitard by assuming that the wells are fully penetrating and that the lower aquitard is impermeable. Table A.19 presents a summary of the aquifer and aquitard characteristics.

Pump test analyses by EMC were hampered by the lack of early drawdown data and by limitations prescribed by the available methods of analysis. The limitations of the Hantush and Jacob (1955) leaky artesian and Hantush (1960) modified leaky artesian methods of analysis could be overcome by



*The fluid level data shown above was taken at 9 am on May 24, 1979, prior to starting the hydraulic pump test. The overlying aquifer (Well B-7-30) and underlying aquifer (Well B-8-29) fluid levels were 5032.5 and 4986.3, respectively.

Fig. A.34. Water level elevations prior to the pump test in the Section 24 mine area. *Source:* Modified from EMC, 1980a.

Table A.19. Summary of aquifer and aquitard characteristics estimated by Exxon Minerals Company and the authors from aquifer test data for the Highland mine.

Test No.	Injection Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1	--	H-Sand	EMC	4.0×10^{-4} to 1.3×10^{-3} (0.9 to 2.8)	--	--	--	--	I
2	P-9	Lower H-Sand	EMC	--	3.2 to 6.4 (34.6 to 69)	--	--	CJ	CD
3	P-9	Drawdown data are insufficient for analysis.							
4	B-7-29	Lower and Middle H-Sand	EMC	--	3.2 to 4.3 (35 to 46)	6.6×10^{-5} to 1.9×10^{-4}	--	CJ	CD
		Lower and Middle H-Sand	Authors	3.6×10^{-5} to 2.0×10^{-4} (0.08 to 0.44)	0.55 to 3.1 (6 to 33)	1.9×10^{-5} to 1.6×10^{-4}	2.6×10^{-8} to 2.6×10^{-6} (7.4×10^{-5} to 7.4×10^{-3})	H	

[a] H-Sand is the uranium-bearing Highland Sandstone unit.

[b] EMC = Exxon Minerals Company

[c] Values are based, in part, on the following assumptions: 1) the aquitard underlying the tested aquifer is impermeable, and 2) the specific storage of the overlying aquitard is two orders of magnitude larger than that calculated for the tested aquifer.

[d] CJ = Cooper and Jacob (1946); H = Hantush (1960).

[e] I = Injection; CD = Constant Discharge.

Source: EMC, 1980a.

using the ratio method developed by Neuman and Witherspoon (1972). The ratio method allows estimation of the vertical hydraulic conductivity for the confining units directly, but specific storage of the aquitards must be determined independently. However, drawdown data for additional wells in the overlying and underlying confining units are needed to use the ratio method, and these data were not collected during the EMC Highland pumping tests. Therefore, more accurate values of vertical hydraulic conductivity of the confining units cannot be determined from the ratio method.

A.4.5 Hydrogeology of the Mine Site

A generalized stratigraphic column for the Highland site is shown in Fig. A.35. The strata adjacent to and underlying the Highland mine site have been folded by a series of northwest plunging anticlines and synclines (Fig. A.36) with amplitudes of 3 to 6 m (10 to 20 ft). No faults have been detected by surface reconnaissance and drilling activities at the site (Langen and Kidwell, 1974); however, minor faults would be difficult to detect in this stratigraphic setting.

The lower Fort Union Formation, which contains the uranium mineralization at the site, is characterized by rapid lateral facies changes from coarse-grained channel sandstones to fine-grained siltstone and claystone floodplain deposits. The uranium ore is present mainly in three arkosic, fluvial sandstones designated by EMC as the Upper, Middle, and Lower Highland Sandstones. These host sandstones vary in thickness from 0 to 15 m (0 to 50 ft); they are separated by 3 to 6 m (10 to 20 ft) of siltstones and claystones. Depth to the Highland Sandstones ranges from 0 to 220 m (0 to 720 ft) across the site (Figs. A.37 and A.38).

Figs. A.37 and A.38 may be over-simplified. It is possible that scour and fill deposits have breached the overlying and underlying confining units in places, thus creating pathways for vertical excursions. Overlying confining units at both the WMC Irigaray site and the EMC Highland site are known to be thin and may actually be absent in localized areas that could be missed by drilling. The ore zone is approximately 7.0 m (23 ft) thick and lies about 97 m (320 ft) below ground surface at the original pilot test site. The ore zone mined at the expanded pilot test site has an average thickness of 8.0 m (27 ft) and lies at a depth of approximately 120 m (400 ft) (EMC, 1978a)

The uranium minerals in the Highland Sandstones are uraninite and coffinite (Langen and Kidwell, 1974). These minerals occur as coatings on sand grains in roll-front, tabular and elliptically shaped deposits, as well as thin sooty layers and irregular masses usually less than 10 microns thick. Anomalous quantities of vanadium and selenium also are associated with the uranium mineralization in the Highland Sandstones. Langen and Kidwell (1974) reported a distinct relationship between long, linear areas of mineralization and the flanks of the northwest plunging anticlines.

Ground water occurs under both confined and unconfined conditions at the Highland mine site. The Highland Sandstones crop out along the southern project boundary (Fig. A.29). The sandstones dip below the site

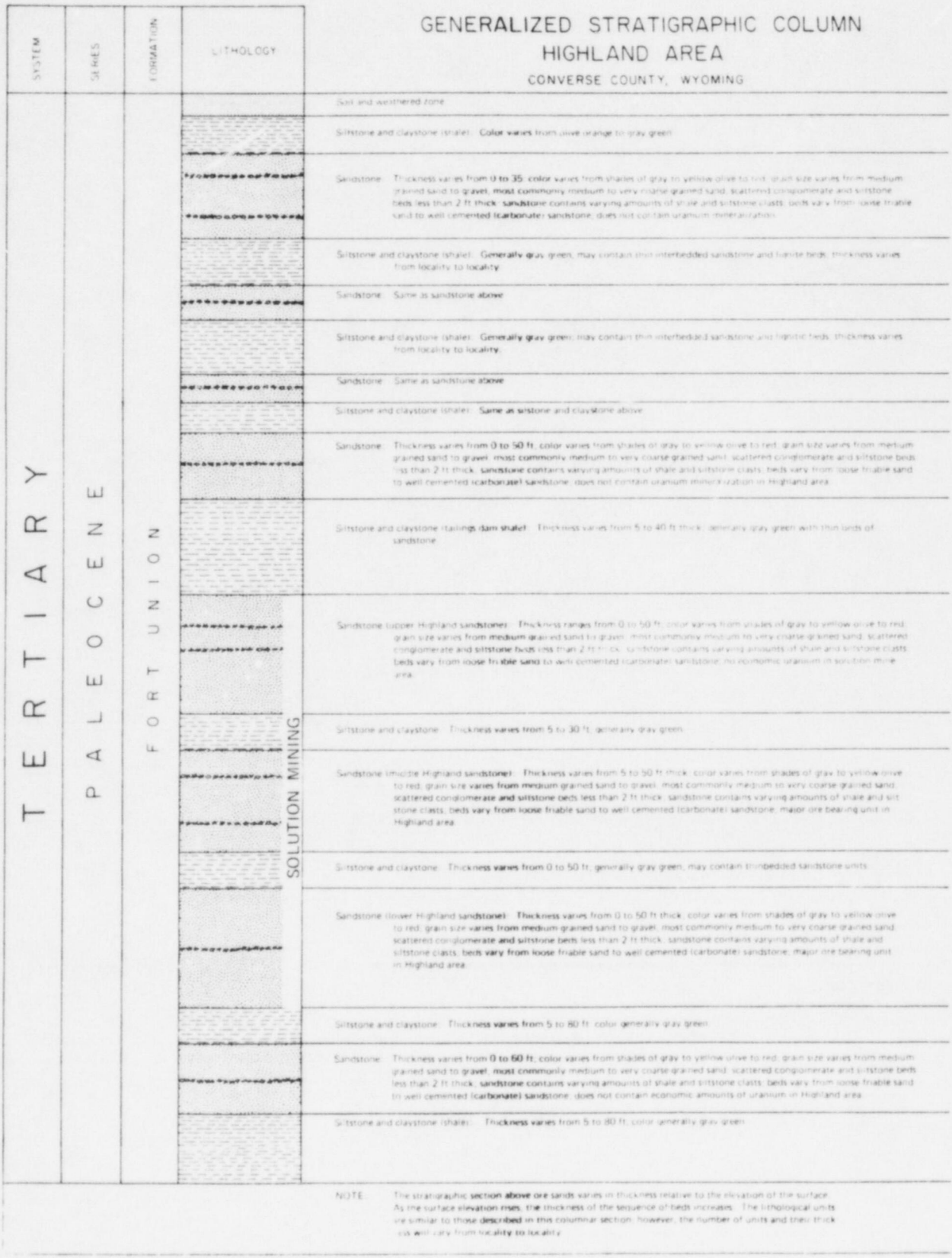


Fig. A.35. Generalized stratigraphic column of the Highland mine area. *Source:* NRC, 1978b.

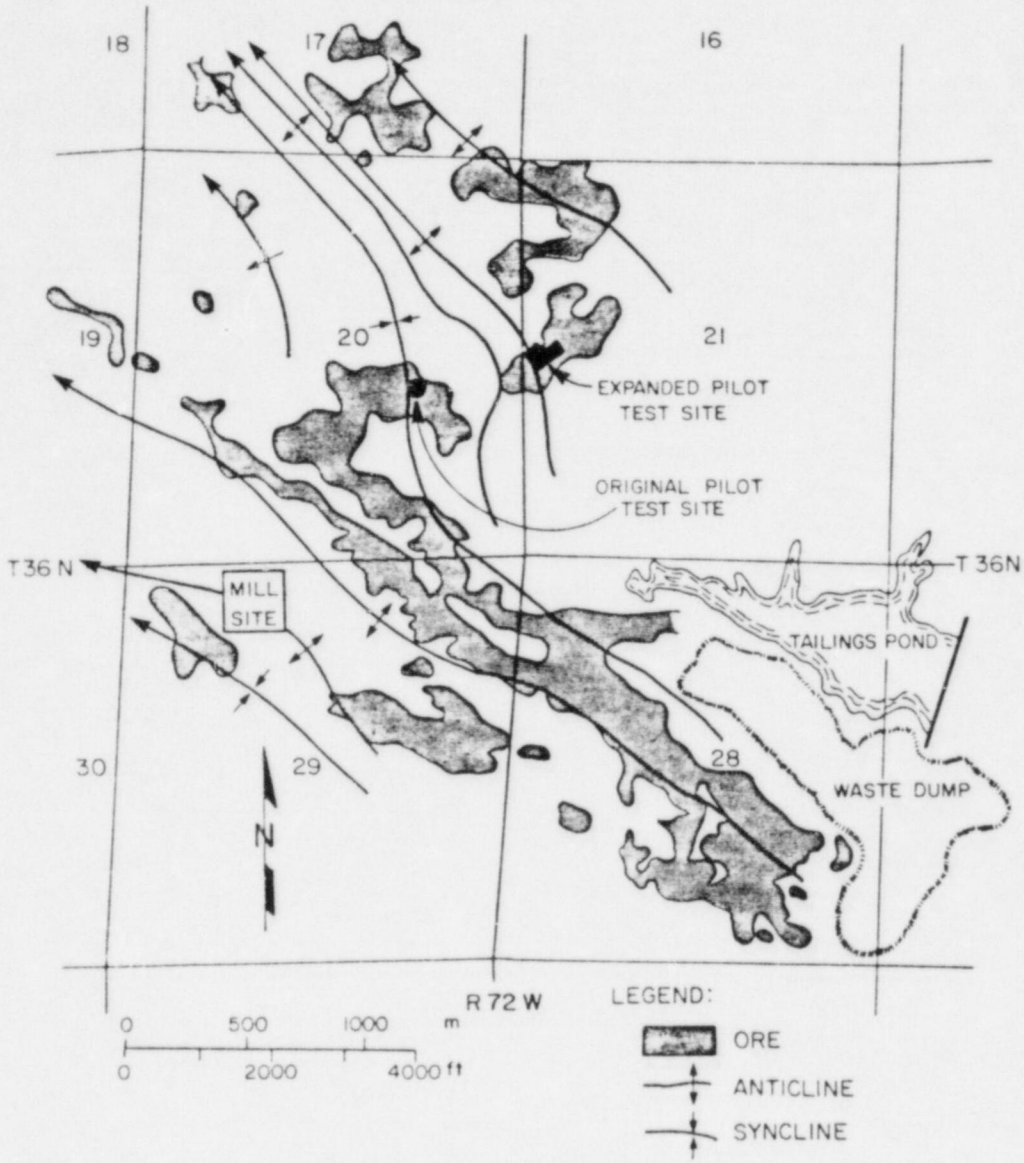


Fig. A.36. Highland ore deposit, showing relationship to local structures. *Source:* Modified from Langen and Kidwell, 1974.

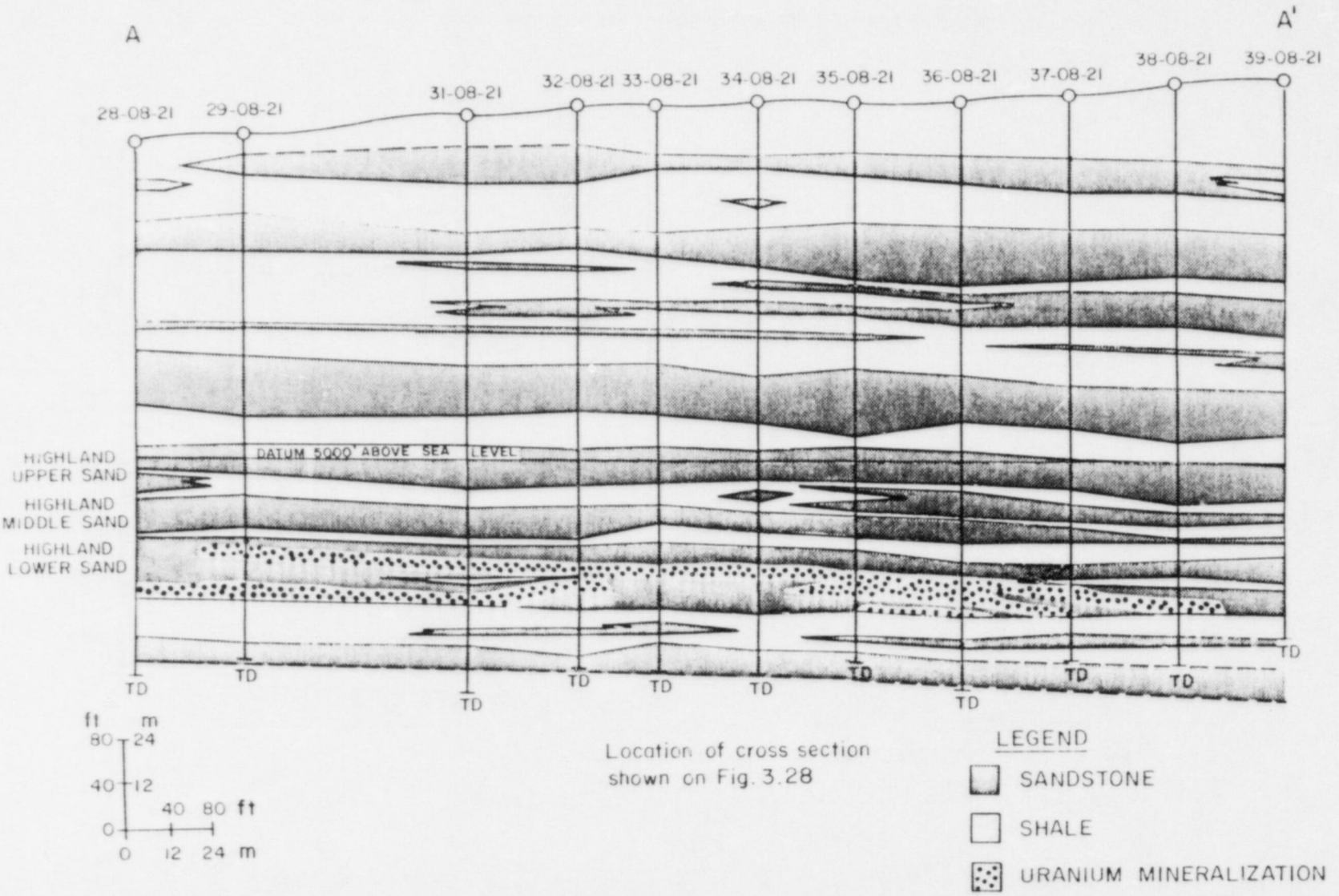


Fig. A.37. Geologic cross section A–A′ in the vicinity of the pilot mine well fields. *Source:* EMC, 1980a.

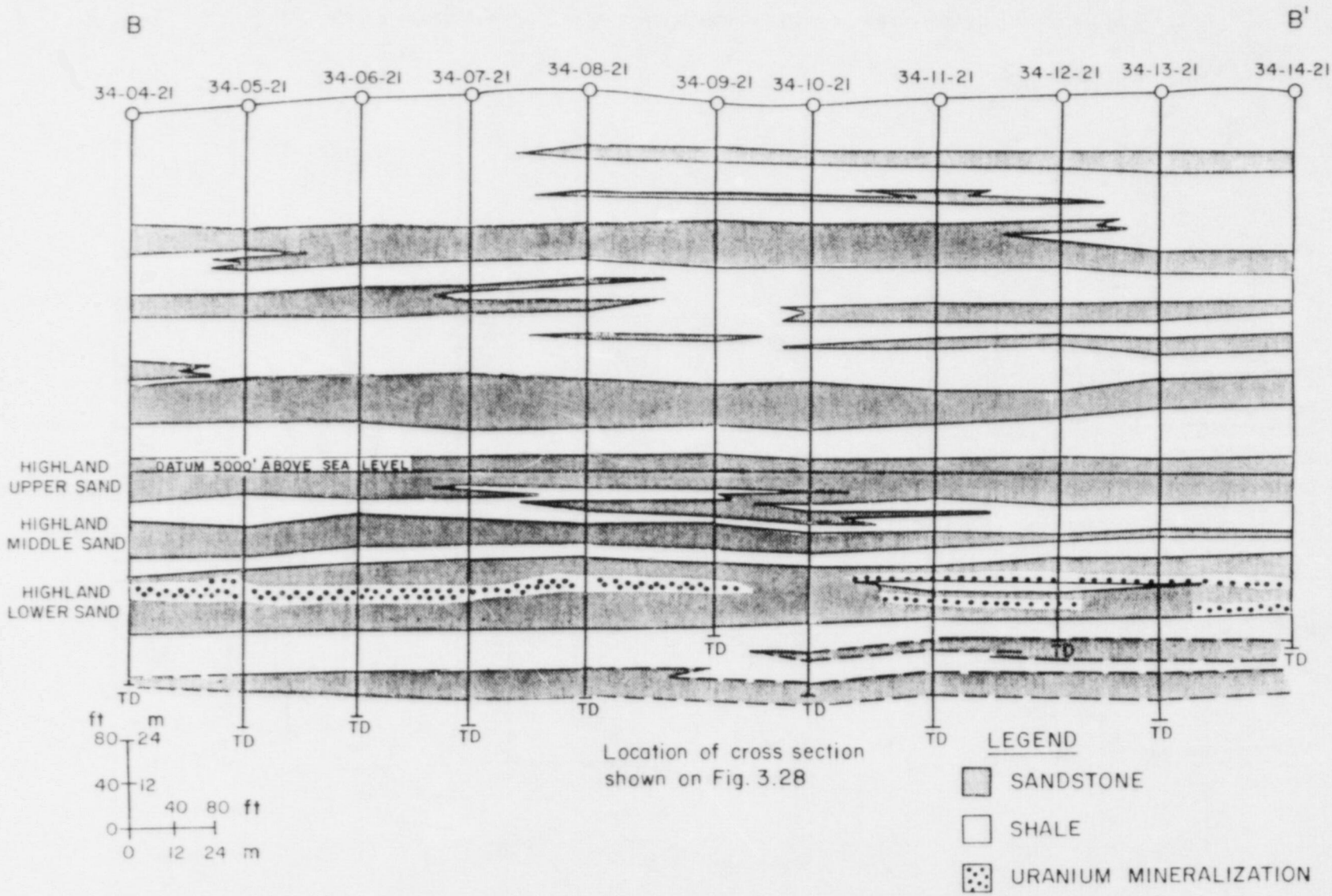


Fig. A.38. Geologic cross section B–B′ in the vicinity of the pilot mine well fields. *Source:* EMC, 1980a.

to depths of up to 220 m (720 ft); these sandstones form confined aquifers beneath the site.

Water levels in the Fort Union Formation at the Highland site have been affected significantly by dewatering of open pit and underground mines in the area. EMC (1978a) reported that the hydraulic gradient was toward the Highland project site because more water was being pumped from their mines than from other surrounding mines. Total withdrawal of ground water from the Highland Sandstone aquifers by EMC and Kerr-McGee is estimated to be about 5.5×10^6 m^3/year (4,500 acre-ft/year) (NRC,1978b).

EMC (1978a) reports that the hydraulic gradient in the Highland Sandstones has changed direction since the original pilot test was conducted. Water level data for the Middle Highland Sandstone for a period of six months in 1969 and 1970 indicate that the direction of ground-water flow was southeast. Water level data for the Lower Highland Sandstone collected in 1978 indicate that the direction of ground-water flow was northwest. EMC explains this apparent change in gradient as a result of mine dewatering in an area northwest of the pilot well field. However, it is questionable whether the gradient measured in the middle unit at the original pilot site was representative of the gradient in the lower unit at the expanded pilot test site in 1969 and 1970.

A.4.6 Monitoring Well System and Well Construction

Well field monitoring at the original pilot test site consisted of six observation wells (ore sand monitor wells) completed in the Middle Highland Sandstone; each observation well was placed about 46 m (150 ft) from the injection well. The original pilot test site is shown in Figs. A.29 and A.30. No wells were completed in overlying or underlying strata within the well field; however, a potable water supply well completed in the Upper Highland Sandstone was located about 61 m (200 ft) northeast of the injection well. No set sampling schedule and no excursion parameters or UCL's were ever established for these wells. The six observation wells were sampled periodically from February 1973, to August 1976. These procedures were established prior to the creation of the Uranium Recovery Licensing Branch of the NRC.

Well field monitoring at the expanded pilot test site (Figs. A.29 and A.31) consisted of 11 monitor wells completed in the Lower Highland Sandstone (ore zone), four "guard" (trend) wells completed in the ore zone, and seven monitor wells completed in either: 1) aquifers above the ore zone, 2) aquifers below the ore zone, or 3) shale intervals above and below the ore zone (Table A.20). Figs. A.37 and 3.38 show the stratigraphy in this area. Monitor wells in the Lower Highland Sandstone encircle the production well field. They are spaced 76 to 107 m (250 to 350 ft) apart and are approximately 61 m (200 ft) from the outer edge of the production well field. Monitor wells in other units are located within the production well field perimeter. Guard wells in the Lower Highland Sandstone are located outside the production well field perimeter but inside the monitor well ring.

Table A.20. Well completion information for the Highland mine.

Well	Completion Date	Completion Zone[a]
	Originl Pilot	
Injection		
Injector	3/10/71	MHS
Production		
1	10/27/69	MHS
2	5/06/70	MHS
3	11/10/69	MHS
4	5/01/70	MHS
5	11/22/69	MHS
6	5/04/70	MHS
Observation		
7	6/16/70	MHS
8	5/25/70	MHS
9	6/05/70	MHS
10	6/02/70	MHS
11	5/28/70	MHS
12	6/09/70	MHS
13	9/01/76	MHS
14	3/15/79	MHS
	Expanded R&D Pilot	
Injection		
E-I-1	11/30/77	LHS
E-I-2[b]	1/11/78	LHS
E-I-2-2	1/23/80	LHS
E-I-3[b]	12/02/77	LHS
E-I-3-2	1/11/80	LHS
E-I-4[b]	12/28/77	LHS
E-I-4-2[c]	1/16/80	LHS
Production		
E-P-1	1/03/78	LHS
E-P-2[b]	12/20/77	LHS
E-P-2-2	8/17/78	LHS
E-P-3[b]	11/18/77	LHS
E-P-3-2	7/17/79	LHS
E-P-4[b]	12/15/77	LHS
E-P-4-2	1/02/80	LHS
E-P-5[b]	11/29/77	LHS
E-P-5-2	8/17/78	LHS
E-P-6	12/22/77	LHS

Table A.20. (Continued)

Well	Completion Date	Completion Zone[a]
	Expanded R&D Pilot (Continued)	
E-P-7	11/18/77	LHS
E-P-8	12/29/77	LHS
E-P-9[b]	12/15/77	LHS
E-P-9-2	7/18/79	LHS
E-P-10	12/22/77	LHS
Monitor		
E-0-1	1/04/78	LHS
E-0-2[b]	11/14/77	LHS
E-0-2-2	9/26/78	LHS
E-0-3	12/21/77	LHS
E-0-4[b]	1/04/78	LHS
E-0-4-2	7/26/78	LHS
E-0-5	11/23/77	LHS
E-0-6	12/03/77	LHS
E-0-7	12/12/77	LHS
E-0-8	12/12/77	LHS
E-0-9	12/19/77	LHS
E-M-1	12/19/77	MHS
E-M-2[b]	12/21/77	(-)LHS
E-M-2-2	2/10/78	(-)LHS
E-M-3	12/18/79	UHS
E-M-4	12/22/79	MHS
E-M-5[b]	12/20/79	TDS
E-M-5-2	12/24/79	TDS
Guard		
E-G-2	11/09/78	LHS
E-G-5	11/17/78	LHS
E-G-8	3/08/79	LHS
E-G-9	11/02/78	LHS

[a]MHS = Middle Highland Sandstone; LHS = Lower Highland Sandstone; UHS = Upper Highland Sandstone; (-)LHS = Sandstone layer below Lower Highland Sandstone; TDS = Tailings Dam Sandstone.

[b]Well not in service.

[c]Converted to a production well in October, 1980; E-P-7 and E-P-8 were removed from service at the same time; Exxon Minerals Corporation decided that the associated well spot was not needed for research purposes, but wanted to produce E-I-4-2 since it was the best well in the well spot.

Source: EMC, 1982b.

Injection and monitor wells at the Highland mine site were constructed according to the following procedure (Humble Oil and Refining Company, 1970). A borehole was drilled beyond the production sandstone into the underlying siltstone/claystone interval. A 14 or 11 cm (5.5 or 4.5 in) steel casing was cemented in place, with the cement filling the annulus from the base of the hole to the ground surface. Cement was later drilled from the inside of the casing to a point where a 4.6 m (15 ft) plug of cement was left at the base of the casing. The casing was perforated at the mineralized sand interval(s).

Production wells were drilled to the top of the mineralized sand interval. An 18 cm (7 in) steel casing was placed into the hole and cemented in place. Cement was drilled from inside the casing and the underlying mineralized sand formation was cored. A slotted 11 cm (4.5 in) steel well screen was set into the sand formation and the space between the screen and the open hole was filled with pea gravel. The wells were developed by pumping to remove silt from the sand formation and gravel pack.

A.4.7 Analysis of Excursions

Excursion indicators were not established for the original pilot test in 1972, probably because of the lack of specific regulations. EMC sampled the production aquifer prior to mining to evaluate the baseline water quality at the test site. Three water samples were taken from production wells in May, 1970, and analyzed for selected parameters. Table A.21 lists the average values for these samples. Because baseline uranium concentrations were found to be low (about 0.2 mg/L), uranium was considered to be a good excursion indicator.

EMC detected increased (above baseline) uranium concentrations in water samples taken from monitor wells surrounding the original test site by the second year of the test. The monitor wells were completed in the production aquifer. In mid-1973, uranium concentrations up to 101 mg/L were detected in wells 7 and 9 (Fig. A.30). Initial reports by EMC indicated that by altering production and injection rates, the leach solutions were drawn back into the production area. In March, 1974, uranium concentrations in wells 7 and 9 were below baseline (EMC, 1979a). Production rates remained lower than injection rates throughout this "excursion" period (NRC, 1982b).

Uranium concentrations up to 92 mg/L were detected in monitor wells 7 and 9 during the second and third quarters of 1974. This occurrence subsequently was explained to be a result of restricted production well pumping from May to August due to equipment problems (EMC, 1979a). After this excursion, production and injection rates were altered to draw mining solutions back into the production well field. Uranium concentrations were reduced below baseline in the monitor wells by June, 1975. Soon thereafter, following what EMC describes as "vigorous air lift treatment" which produced about 3900 L (1296 gal) of fluid from each monitor well, uranium concentrations up to 313 mg/L were detected in monitor wells 7, 8, and 10. Within one year, the uranium concentrations decreased to levels ranging

Table A.21. Baseline water quality at the Highland original pilot test site.[a]

Parameter	Concentration[b]
Sodium	161
Calcium	77
Magnesium	13
Chloride	27
Sulfate	119
Bicarbonate	237
Selenium	0.5
Uranium	0.212
Radium-226 (pCi/L)	120
Thorium-230 (pCi/L)	86

[a]Average of three samples taken from three production wells in original pilot area during May, 1970.

[b]Units are mg/L except as noted.

Source: EMC, 1978a.

between 10 and 40 mg/L (NRC, 1982b). This decline in uranium concentrations probably was due to the fact that injection of lixiviant was discontinued in November, 1974, while production well pumping continued. The unexpected occurrence of uranium in the monitor wells was explained by EMC as the result of leach solutions migrating rapidly along more permeable zones within the ore zone.

Several notable occurrences of elevated concentrations of excursion indicators were reported by EMC during production operations at the expanded pilot test site. UCL's for various excursion indicators were modified during the expanded pilot test. Preoperational ground-water sampling and analysis were conducted at the expanded pilot test to establish baseline water quality. Fourteen wells were sampled over the first half of 1978. Most constituents analyzed were present at concentrations below Wyoming Drinking Water Standards except ^{226}Ra, iron, manganese, pH, and TDS.

UCL's for each well were set originally for the following excursion indicators: bicarbonate, carbonate, arsenic, selenium, chloride, uranium, TDS and/or conductivity, and pH. UCL's were set 20% above mean baseline for all parameters in each well except uranium and pH. The limits for uranium and pH were set at 1 mg/L and 2 units above mean baseline, repectively. The UCL's have been changed several times; arsenic, selenium, TDS and/or conductivity, and pH have been deleted. Sulfate has been added as an excursion indicator. The present UCL's are: 1) bicarbonate plus carbonate - 20 mg/L above the highest value recorded for each well during the baseline sampling program, 2) chloride - 10 mg/L above the highest value recorded for each well during the baseline sampling program, 3) uranium - 5 mg/L above the highest value recorded for each well during the baseline sampling program, and 4) sulfate - 20% above the highest value recorded for each well during the baseline sampling program.

Originally, an excursion was defined by two indicators having exceeded their UCL's at any well. An excursion now is defined when two indicators exceed their UCL's or if any one indicator, except sulfate, exceeds its UCL by 20% or more. Monitor wells presently are sampled on a biweekly basis and are analyzed only for the current excursion indicators. Sampling of the guard wells is sporadic and appears to be dependent on excursion occurrence. Excursion events are summarized in Table A.22.

UCL's for bicarbonate, chloride and conductivity were exceeded in ore zone monitor well E-0-9 (Fig. A.30) from December 26, 1978, to February 6, 1979. The injection/production ratio was decreased in the nearest well pattern as corrective action for the excursion, and an additional monitor well was installed between the well pattern and well E-0-9. After sampling the new well, EMC explained the excursion in well E-0-9 as follows: elevated bicarbonate was the result of the conversion of carbonate to bicarbonate in the well; elevated chloride was due to residual salt from cementing activities (salt had been added to the cement to promote hardening); and elevated conductivity was attributed to the higher bicarbonate and chloride concentrations. Water samples from the newly installed monitor well indicated levels of chloride, bicarbonate, and conductivity to be

Table A.22. Summary of excursion events at the Highland mine site.

Well Designation	Period of Occurrence	Parameter	Sample Concentration	Upper Control Limit[a]
Original Test Site				
7 and 9	6/73 to 3/74	Uranium	≤101	---
7 and 9	5/74 to 6/75	Uranium	≤94	---
7, 8 and 10	8/75 to 9/76	Uranium	≤313	---
Expanded Test Site				
E-0-9	12/26/78 to 2/6/79	Bicarbonate	100-448	178
		Chloride	62-190	60
		Conductivity, µmhos/cm	610-1500	588
E-0-3	1/25/79 to 3/13/79	Bicarbonate	158-211	146
E-0-8	1/25/79 to 3/13/79	Conductivity, µmhos/cm	480-580	480
I-3	12/12/79 to 3/31/80	(casing leak into upper Highland sandstone)		
E-0-7	3/16/81 to 8/12/81	Bicarbonate	223-262	240
		Chloride	92-110	90
		Sulfate	115-147	112
E-0-8	3/16/81 to 8/12/81	Sulfate	86-122	101
		Chloride	25-40	20

[a]Units are mg/L expect as noted.

about 24 mg/L, 200 mg/L, and 588 umhos/cm, respectively. These values are typical of premining conditions. EMC therefore concluded that excursion of leach solutions was not indicated by conditions in well E-0-9 (EMC, 1979b). This conclusion was accepted by the NRC as excursion parameter concentrations remained below UCL's.

The UCL for bicarbonate (146 mg/L) was exceeded in well E-0-3 (Fig. A.31) from January 25 to March 13, 1979. Well E-0-3 monitors the production aquifer. Concentrations up to 211 mg/L were reported during this time. EMC reported that neither of the adjacent monitor wells experienced similar bicarbonate fluctuations during the same period; the elevated bicarbonate in E-0-3 was attributed again to the conversion of carbonate to bicarbonate. Spatial variation in water quality, natural ground-water movement and abnormally low baseline values were mentioned also as possible contributing factors (EMC, 1979c).

Elevated conductivity values were also measured in wells E-0-8 and M-1 during the first quarter of 1979. Well M-1 completed in the Middle Highland Sandstone (Fig. A.31) is designed to detect vertical migration of mining solutions from the Lower Highland Sandstone. Subsequently, EMC requested that conductivity be deleted from the list of excursion indicators on the basis of fluctuations in conductivity at various wells during the history of the expanded pilot test. This request was granted by the NRC. However, conductivity values have in fact generally remained below the UCL's originally set for wells E-0-8 and M-1.

A casing leak in injection well I-3 (Fig. A.31) was discovered on December 12, 1979. The leak occurred about 76 to 84 m (250 to 275 ft) below ground surface and flowed at about 0.008 L/s (0.12 gal/min) into the Upper Highland Sandstone. A well was constructed within 3 m (10 ft) of well I-3 and pumped for over three months at about 0.63 L/s (10 gal/min) to withdraw the lixiviant. Analyses of water samples from this new well and two other wells [M-4, located 15.2 m (50 ft) east of I-3-2 and completed in the Middle Highland Sandstone, and M-5-2, located 15.2 m (50 ft) south of I-3-2 and completed in a clayey sand overlying the Upper Highland Sandstone] indicated water quality typical of premining conditions. EMC concluded on the basis of these data that no vertical excursion of mining solutions through the confining strata had occurred.

Chloride and sulfate concentrations exceeded their UCL's in well E-0-8 in December, 1980. Chloride concentrations had been increasing over the previous six months; therefore, EMC suggested the trend could have been due to a change in the natural ground water. Chloride concentrations remained above the UCL until April, 1981. However, chloride concentrations have declined since the beginning of 1981, and ranged from 6 to 12 mg/L in 1983. Sulfate concentrations, which exceeded the UCL on December 9, 1980, dropped to or below the UCL by December 30, 1980.

Excursions were confirmed in wells E-0-7 and E-0-8 (Fig. A.31) in March, 1981. Bicarbonate, chloride and sulfate levels exceeded UCL's in well E-0-7, and chloride and sulfate concentrations exceeded UCL's in well E-0-8. The injection/production ratio was lowered in an adjacent well

pattern as corrective action for these excursions, and concentrations of chloride and bicarbonate had dropped below their respective UCL's by August 1981. EMC attributed these excursions to a testing procedure whereby solution was produced by pumping from an injection well. This was conducted to see if lixiviant could be circulated more effectively in the region between injection wells.

The fluctuating sulfate levels during the expanded pilot test led EMC to request that sulfate be deleted as an excursion indicator. Poor reliability of sulfate analyses and very low UCL's for sulfate were given as the basis for this request. EMC reasoned that sulfate was not useful as an excursion indicator because no sulfuric acid was used in the Highland test and oxidation of sulfide materials to sulfate in the ore zone appeared to be minimal (EMC, 1980b). The method of establishing UCL's for sulfate subsequently was relaxed; however, sulfate levels exceeded the newly established, higher UCL in the first few months of 1982 (NRC,1982b).

In summary, excursion events at the Highland test well fields seem to be a result of not maintaining proper hydraulic gradients in the mining areas. The injection/production ratios were not maintained at sufficiently low values to sustain hydraulic gradients toward the well fields. One vertical excursion of mining solutions has been verified. This excursion was due to a casing leak which resulted in the introduction of mining solutions into an overlying aquifer.

The aquifer test data suggest that water may be released from storage in the confining layers under pumping conditions. The aquifer test data do not indicate the presence of any major discontinuities in the confining layers. The limited aquifer test data (supported by the nonoccurrence of naturally caused vertical excursions) suggest that the confining strata are continuous and of relatively low hydraulic conductivity in the vicinity of the expanded pilot test. Scour and fill deposits in the confining layers potentially exist in other areas of the Highland site; these deposits could provide vertical pathways for lixiviant migration out of the ore zone.

A.4.8 Restoration

Restoration of the original pilot well field has been in progress since November, 1974, when lixiviant injection was terminated. The lixiviant used at this well field was sodium carbonate/bicarbonate. The restoration technique utilized has been a combination of ground-water sweeping and clean water recycle. The recovered water is treated by reverse osmosis. From November, 1974, to April, 1975, solution produced from the production wells was stripped of uranium and reinjected into the production aquifer through the injection well. Once the uranium concentration in the withdrawn solution became too low for economic recovery, the solution was fed through a reverse osmosis circuit to reduce concentrations of contaminants. The effluent from the reverse osmosis circuit was discharged into the Highland uranium mill tailings pond and the clean water was reinjected into the production aquifer through the injection well (Fig. A.30). By June, 1975, uranium concentrations in the ground water in the monitor wells had dropped below the limits of detection (1 mg/L).

Beginning in November, 1980, fresh water was injected through the production wells while the injection well and three monitor wells were pumped at a combined rate of 0.31 to 0.76 L/s (5 to 12 gal/min) in an attempt to reduce further the concentrations of arsenic, selenium and uranium which remained high in the portion of the well field northeast of the injection well. Approximately 8.8×10^7 L (2.2×10^7 gal) of ground water, or about 6.6 pore volumes, were produced during the restoration phase of the original pilot test from November, 1974, to July, 1982 (EMC, 1982a,b). Table A.23 presents water quality data for 1983. This table indicates that uranium concentrations exceeded baseline as of December, 1983, although reported values for all wells except #13 and #14 are below the 5 mg/L drinking water standard. In August, 1983, arsenic concentrations were found to be <0.001 mg/L and 0.002 mg/L, respectively. The original pilot well field remained in a post-restoration stability monitoring mode as of June, 1984.

Restoration of the expanded pilot well field began in October, 1981. Restoration techniques utilized at this well field included ground-water sweeping and clean water recycle with ion exchange and reverse osmosis used for treatment. Fresh water was injected into the ore zone through the injection wells and ground water was withdrawn at the production wells. The ground water was passed through an ion exchange circuit to remove uranium and then through a reverse osmosis circuit to reduce contaminants. The effluent from the reverse osmosis circuit was discharged to the mill tailings pond; the treated and fresh water was injected into the ore zone through the injection wells (NRC, 1982b). Use of the reverse osmosis circuit was discontinued in May, 1983. The ground-water sweep procedure produced approximately 2.5×10^6 L (6.5×10^5 gal) of ground water per month from the ore zone throughout the fourth quarter of 1983. This process reduced the TDS content and uranium concentrations.

Restoration targets proposed originally by EMC (1981) for both pilot well fields were: 1) to reduce radionuclides to levels below the maximum permissible concentrations (MPC) for unrestricted areas (NRC, 1982b) or to concentrations less than or equal to 1.2 times natural background concentrations where the natural background is greater than MPC, and 2) to reduce nonradioactive constituents to concentrations consistent with Wyoming Department of Environmental Quality livestock criteria or to concentrations less than or equal to 1.2 times natural background, whichever is greater. However, the NRC declared these unacceptable (NRC, 1982b).

Subsequently, EMC submitted documentation which they believe demonstrates maximum restoration of the original pilot well field. This information currently is under review by the NRC and the Wyoming Department of Environmental Quality. EMC has restored water quality parameters at the expanded pilot well field to levels acceptable to the NRC as of June, 1984.

Table A.23. Post-restoration water quality data (mg/L) for the original pilot well field in 1983.

	Carbonate	Bicarbonate	Uranium	Chloride	Sulfate
Baseline	--	237	0.212	27	119
Injection Well					
June			0.9		
Well #1					
June			--		
Well #2					
June			3.4		
Well #3					
June			2.9		
Well #4					
June			0.9		
Well #5					
June			0.9		
Well #6					
June			0.9		
Well #7					
January	26	244	8.5	15	173
February	37	236	12	10	141
March	31	211	10	15	129
June	--	--	4.2	--	--
December	12	134	1.7	21	124
Well #8					
January	1	234	6	10	160
February	1	217	2.6	5	123
March	1	207	2.6	7	156
June	--	--	0.9	--	--
December	<1	170	<0.9	7	83
Well #9					
January	1	199	1.7	12	180
February	1	200	1.7	10	151
March	1	145	1	10	207
June	--	--	0.9	--	--
December	[illegible]	104	<0.9	8	113
Well #10					
January	1	228	6	12	224
February	1	224	4.3	12	188
March	1	233	2.6	12	115
June	--	--	3.4	--	--
December	<1	143	<0.9	8	156
Well #11					
January	1	179	3.4	12	193
February	1	166	2.6	10	155
March	1	[illegible]8	1	13	177
June	--	--	0.9	--	--
December	<1	112	<0.9	8	153
Well #12					
January	1	173	1	12	160
February	1	193	1	7	137
March	1	195	1	10	171
June	--	--	1.7	--	--
December	5	143	0.9	21	129
Well #13					
June			23.7		
August			8.5		
Well #14					
June			2.5		
August			6.8		

Source: NRC, 1983a and EMC, 1984.

A.5 TETON EXPLORATION DRILLING COMPANY - LEUENBERGER PROJECT

A.5.1 Description of the Site

Teton Exploration Drilling Company (TEDC) operates the Leuenberger in situ uranium mine in Converse County, Wyoming, about 12 km (7.5 mi) northeast of Glenrock (Fig. A.39). The site is located in the southern portion of the Powder River Basin. Elevations at the site range from about 1570 to 1640 m (5160 to 5380 ft). Gentle slopes exist throughout the mine area, and the east end of the site is located on the crest of a broad, low ridge. Two streams which flow only in response to snowmelt and precipitation events drain the mine site. The North Platte River, approximately 6.7 km (4 mi) south of the Leuenberger mine, is the nearest perennial stream. The average annual precipitation in the area is about 30 cm (12 in).

Solution mining research and development operations (Fig. A.40) began on January 22, 1980, within two well fields. The N and M ore zones were mined separately in each of the two well fields. The N and M ore bodies are located in Section 14, T34N, R74W. All mining, processing, and restoration were concluded at the Leuenberger mine as of January, 1982. TEDC has no plans for commercial-scale operations or additional research and development testing in the near future. Post-restoration monitoring continued through 1982.

A.5.2 History of Mining Operations

Research and development testing began January 2, 1980, in two separate five-spot well patterns. One pattern was designated for each of the N and M ore zones (Fig. A.41). Each test pattern consisted of four injection wells placed approximately 15 m (50 ft) apart surrounding a central pumping well. Two additional five-spot patterns contiguous to the original M zone pattern were drilled. An additional pattern was completed in the N zone well field, but was never utilized for mining.

The pilot testing operation utilized a sodium carbonate/bicarbonate lixiviant at a concentration of about 2 g/L, with a hydrogen peroxide oxidant at a concentration of about 0.5 g/L until August 26, 1980, when oxygen (250-400 ppm) was tested. The production rate was approximately 2% to 4% above the injection rate. Leaching was terminated in the N well field area on June 1, 1980, after about 47 aquifer pore volumes of lixiviant were circulated through the test pattern. Approximately 29 aquifer pore volumes of lixiviant were circulated through the three M zone test patterns by the completion of mining on February 17, 1981.

TEDC proposed to utilize a five-spot well pattern comprised of four injection wells surrounding a central recovery well during commercial-scale operation. Production rates would exceed injection rates by approximately 0.5% to reduce the possibility of excursion. All wells were to have the capability for both injection and production to enable well field flow reversal to facilitate ground-water restoration. Pattern dimensions were proposed to be approximately 15 by 30 m (50 by 100 ft) in the N ore zone. The commercial project would use a lixiviant consisting of sodium

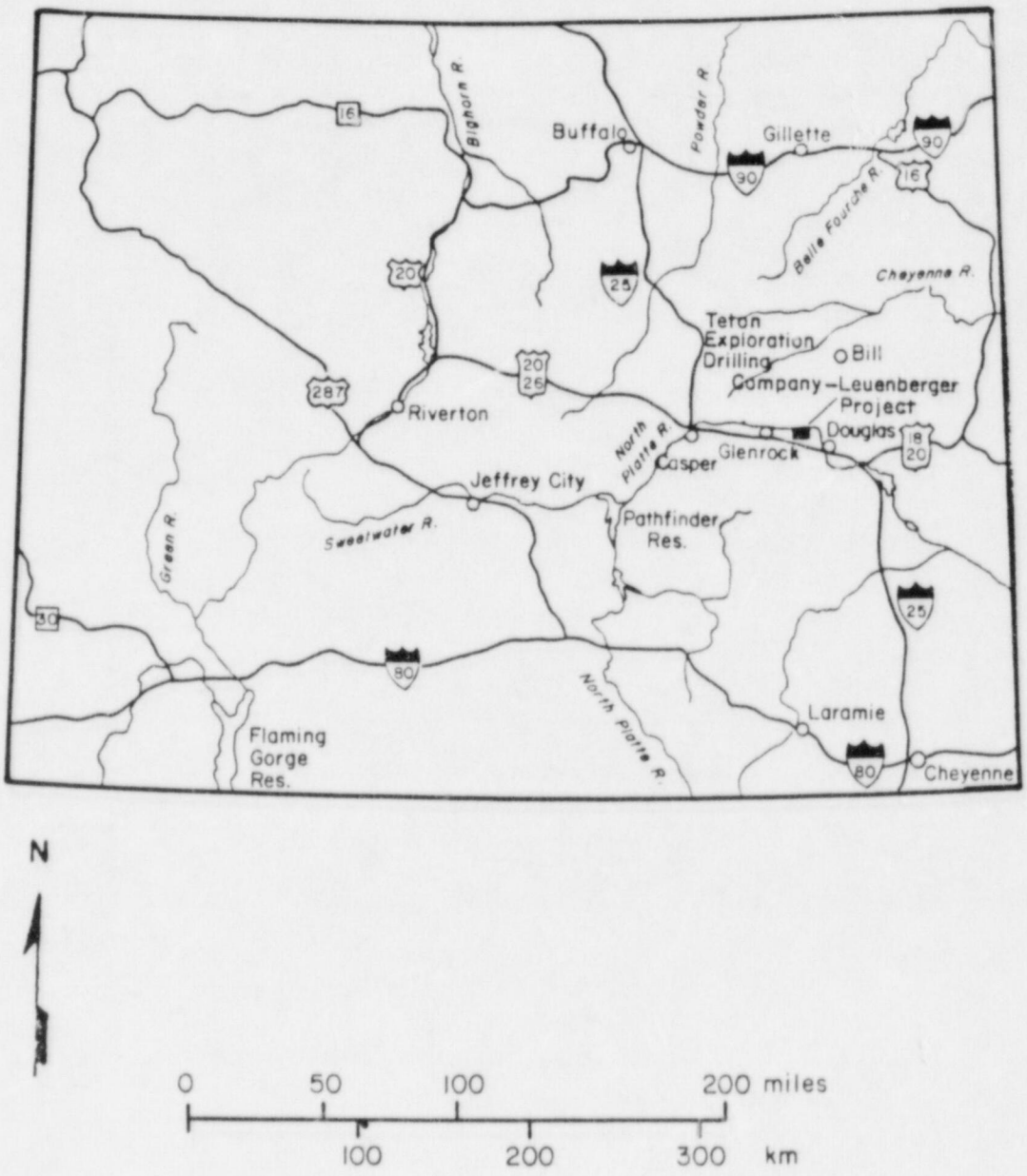


Fig. A.39. Location map of the Teton Exploration Drilling Company Leuenberger mine.

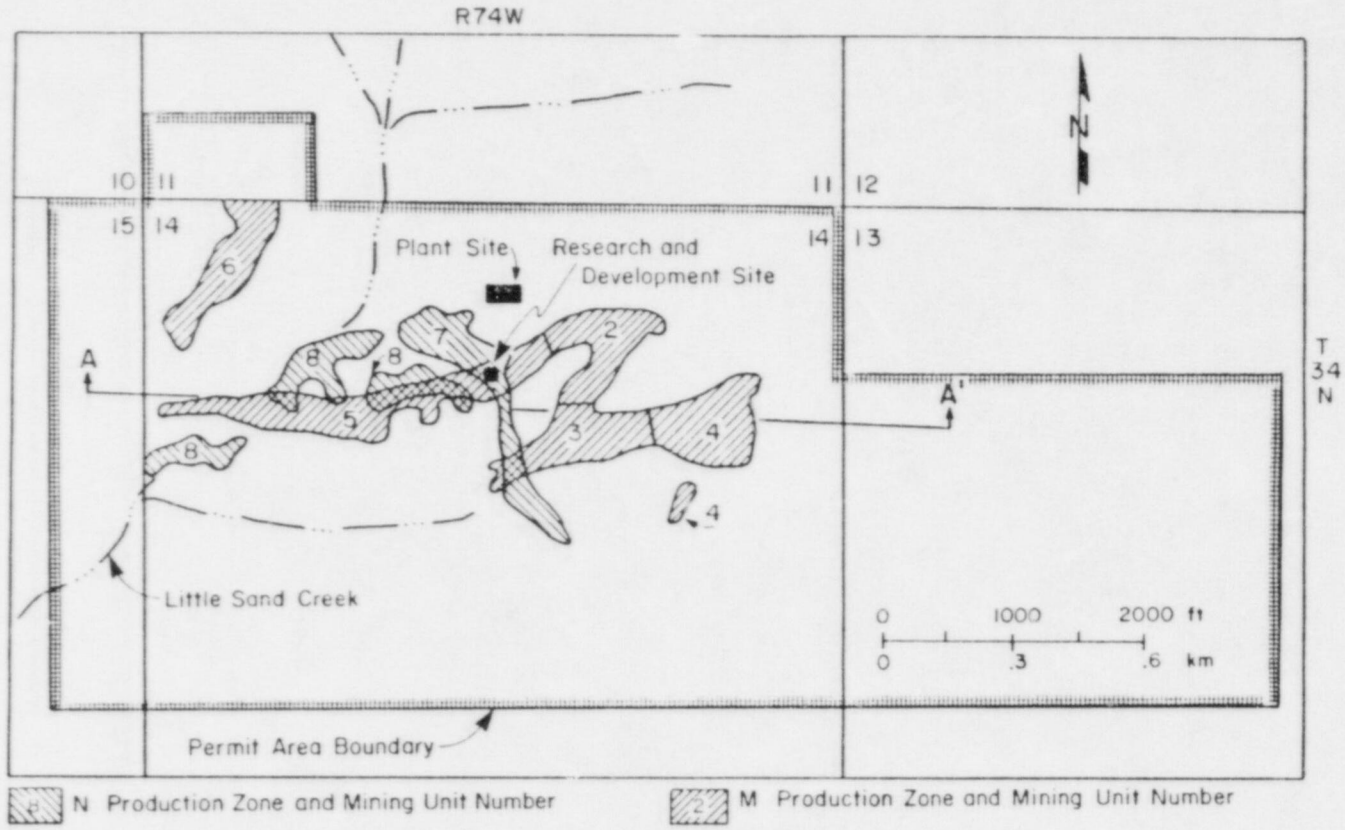


Fig. A.40. Leuenberger uranium project site and immediate vicinity.

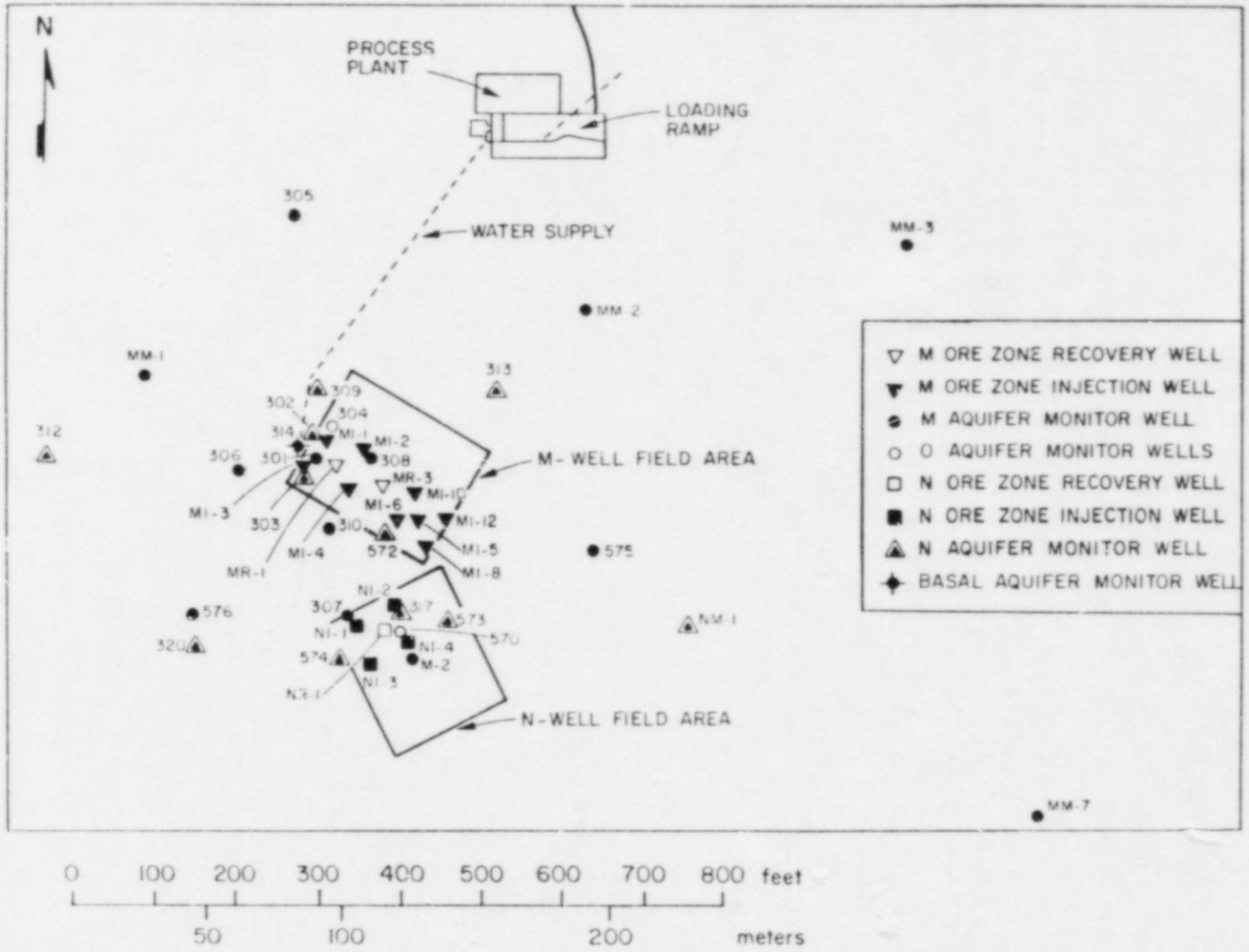


Fig. A.41. Research and development well field at the Leuenberger site. *Source:* Modified from NRC, 1982c.

carbonate/bicarbonate and either oxygen or hydrogen peroxide in water. Injection pressures were expected to be 6.9×10^5 N/m^2 (100 psi) or less. The commercial well patterns at Leuenberger would be distributed over six mining units in the M zone and two mining units in the N zone (Fig. A.40).

A.5.3 Regional Setting

A.5.3.1 Regional Geology

The Leuenberger project site is on the southwest flank of the Powder River Basin approximately 80 km (50 mi) south of the Irigaray site (Fig. A.5) and about 10 km (6 mi) from the basin axis. Solution mining of uranium at the Leuenberger mine site is limited to the Lebo Member (N and M sandstones) of the Paleocene Fort Union Formation. Sedimentary units of the Fort Union Formation were deposited in a swampy, forested lowland threaded by sluggish rivers (NRC, 1982c). The Laramie Range, Casper Arch, and Hartville Uplift probably are the sources of the materials comprising the Fort Union Formation.

Sedimentary units in the Powder River Basin dip at a low angle along axes that parallel the north-northwest basin trend. No major faults are known to exist in the Powder River Basin. The dip of strata in the Leuenberger project area averages 3 to 5 degrees to the northeast.

A.5.3.2 Regional Ground-water Flow Systems

The thick sequence of sedimentary rocks in the Powder River Basin contains numerous water-bearing zones. Ground water occurs under confined conditions in most of these zones due to the regional dip and alternating sequences of sandstone, siltstone, shale, and coal. Ground-water movement generally is controlled by the topography, structure, and stratigraphy. Precipitation supplies most of the recharge water to the Cenozoic rocks of the basin.

Ground water is utilized in the region for livestock watering, domestic purposes, and for limited irrigation. The Fort Union and Wasatch formations have potential for significant ground-water yields in the vicinity of the Leuenberger site. There are 41 wells supplying water for stock use and 20 wells supplying water for domestic purposes within 5 km (3 mi) of the project site. No public water supply wells are recorded within 5 km (3 mi) of the mine property.

A.5.4 Description and Analysis of Data Collected

A.5.4.1 Exploration

Numerous exploration boreholes have been drilled at the Leuenberger mine site to evaluate the location and grade of roll-front mineralization. TEDC (1980) has identified approximately 1200 abandoned wells and boreholes in the vicinity of the Leuenberger mine site from their records and public records. The holes include boreholes drilled prior to TEDC's activities; however, the exact number of these boreholes is not reported. All known

wells and boreholes listed at the Leuenberger site were drilled between 1969 and 1980.

The holes drilled by TEDC (1980) were plugged and abandoned (method unknown) in compliance with the regulations of the State of Wyoming. TEDC inspected aerial photographs of the mine vicinity in an attempt to locate and identify borehole locations. Approximately 250 "surface disturbances were located from these aerial photographs. TEDC seems to have made a good effort to locate all boreholes in the mine vicinity; however, undetected abandoned boreholes may exist at the Leuenberger mine site. In addition, the quality of borehole plugging throughout the mine site cannot be ascertained.

Geophysical gamma logs were recorded for the exploration boreholes and wells at the Leuenberger site; resistivity and spontaneous potential logs were recorded for some of the holes. Geophysical logs are available for about 60 boreholes. Complete geophysical suites were not recorded for all exploration boreholes at the mine site.

Geophysical logs were used by TEDC (1980) to construct seven cross sections both along and across the major trend of the ore zones. Identification of lithologic units by TEDC was based on the shape and magnitude of resistivity, gamma, and spontaneous potential responses. The ore-bearing sandstones exhibit significant magnitudes of response on all three logs; the other sandstone and claystone units are readily distinguishable. Thin, discontinuous sandstone stringers usually are detectable in the claystone units.

A.5.4.2 Aquifer Tests

Seven aquifer tests were conducted by TEDC throughout the Leuenberger site between 1979 and 1981. Three multiple-well pumping tests were conducted in the M Sandstone and one multiple-well pumping test was conducted in the N Sandstone. Two single-well recovery tests were conducted in the Basal Sandstone. One single-well recovery test was conducted in the O Sandstone. Table A.24 summarizes the aquifer tests conducted at the Leuenberger site. Locations of these aquifer tests are shown on Fig. A.42. All wells fully penetrate their respective aquifers.

The aquifer tests were either constant discharge pumping tests or recovery tests. However, discharge data submitted for aquifer test number 1 show that the pumping rate varied significantly during the test from 2.5 to 4.2 L/s (40 to 66 gal/min), and the pump was turned off for eight minutes after 139 minutes of pumping had occurred. Because these discharge variations do not adhere to the constant discharge criterion for these pumping tests, the results of test number 1 are considered questionable. Aquifer and aquitard characteristics estimated by TEDC and by the authors are presented on Table A.25. Aquifer tests 3, 4, and 7 (Table A.25) were single-well tests, and only recovery data are reported for these tests. TEDC (1980) analyzed these recovery data using the Jacob straight line method (Cooper and Jacob, 1946). Results reported for the single-well recovery tests represent rough approximations of aquifer transmissivity. The results cannot be used to evaluate the degree of aquifer confinement.

Table A.24. Summary of aquifer tests conducted at the Leuenberger mine.

Test No.	Pumping Well	Mining Unit	Aquifer[a]	Total Depth of Tested Well m (ft)	Number of Observations Wells	Units Monitored[a]	Type of Test[b]	Date of Test	Length of Test (hrs)	Discharge Rate L/s (gal/min)
1	301	1	M-Sand	119 (390)	6	M-Sand, N-Sand, Basal	CD	2/21/79	48	2.5 to 4.2 (40 to 66)
							R	2/23/79	32	
2	317	7	N-Sand	75 (245)	8	M-Sand, N-Sand, O-Sand	CD	6/26/79	36.5	2.7 (43)
							R	6/28/79	35	
3	BM-2	6	Basal	153 (503)	0		R	7/14/80	1.7	0.9 (15)
4	BM-1	4	Basal	171 (561)	0		R	7/18/80	1.8	1.8 (29)
5	MM-6	6	M-Sand	117 (384)	4	M-Sand, N-Sand, Basal	CD	7/21/80	96	1.9 (30)
							R	7/25/80	96	
6	MM-10	4	M-Sand	128 (421)	5	M-Sand, N-Sand, Basal	CD	7/29/80	96	1.7 (26)
							R	8/02/80	96	
7	LAP-1	-	O-Sand	40 (131)	0		R	8/27/81	1.7	1.2 (19)

[a]M-Sand and N-Sand are the uranium-bearing sandstone units; Basal is the sandstone below the M-Sand; O-Sand is the sandstone unit overlying the N-Sand.

[b]CD = Constant discharge pumping test; R - Recovery test.

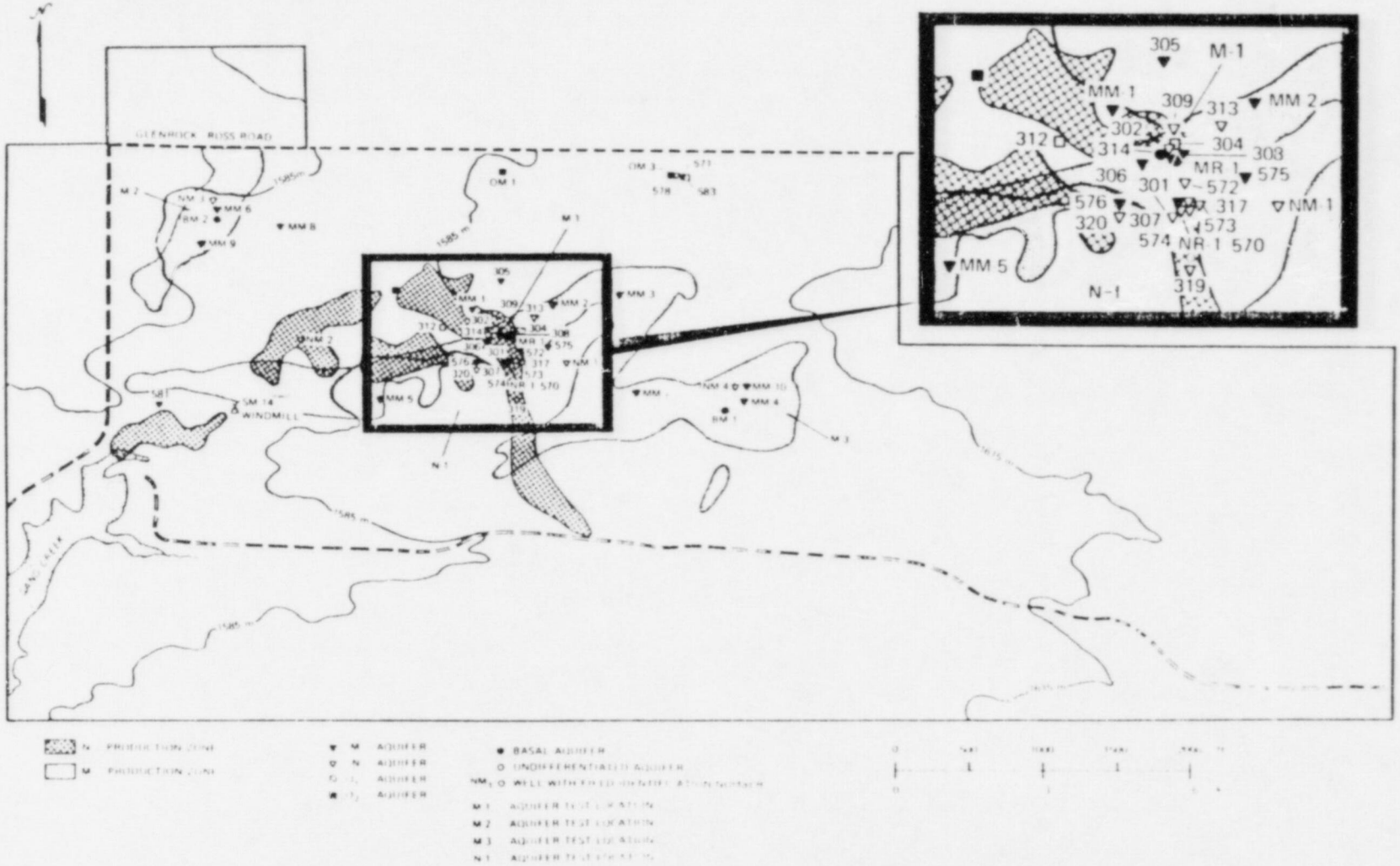


Fig. A.42. Location map of the four ore zone pumping tests (pumping wells 301, 317, MM-6, MM-10) and two Basal recovery tests (wells BM-1 and BM-2) at the Leuenberger site. *Source:* NRC, 1982c.

Table A.25. Summary of aquifer and aquitard characteristics estimated by Teton Exploration Drilling Company and the authors from aquifer test data for the Leuenberger mine.

Test No.	Pumping Well	Aquifer[a]	Analysis Conducted By[b]	Tested Aquifer[c]			Aquitard Overlying Tested Aquifer[c]	Method of Analysis[d]	Type of Test[e]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1	301	M-Sand	TEDC	$3x10^{-4}$ to $4x10^{-4}$[f] (0.8 to 1.2)	4 to 7[f] (47 to 74)	$2.6x10^{-4}$[f]	--	T,CJ,H,Hh	CD
			TEDC	$2x10^{-4}$ to $4x10^{-4}$[f] (0.7 to 1.0)	4 to 5[f] (43 to 58)	--	--	CJ	R
2	317	N-Sand	TEDC	$6x10^{-4}$ to $9x10^{-4}$[f] (1.6 to 2.5)	7 to 12[f] (79 to 127)	$8.3x10^{-5}$[f]	--	T,CJ,H,Hh	CD
			Authors	$2x10^{-4}$ to $4x10^{-4}$[f] (0.5 to 1.0)	2 to 4 (23 to 43)	$3x10^{-5}$	$4x10^{-7}$ to $3x10^{-5}$ ($1x10^{-3}$ to $7x10^{-2}$)	H	CD
			TEDC	$5x10^{-4}$ to $8x10^{-4}$[f] (1.6 to 2.4)	7 to 12[f] (80 to 120)	--	--	CJ	R
3	BM-2	Basal	TEDC	$2x10^{-4}$ (0.5)	1.5 (16)	--	--	CJ	R
4	BM-1	Basal	TEDC	$4x10^{-4}$ (1.0)	7 (72)	--	--	CJ	R
	MM-6	M-Sand	TEDC	$2x10^{-4}$ to $3x10^{-4}$ (0.5 to 0.7)	3 to 4 (34 to 47)	$6.5x10^{-5}$	--	T,CJ,H	CD
			Authors	$1.4x10^{-4}$ (0.4)	2 to 3 (25 to 29)	$5x10^{-5}$	$1x10^{-8}$ to $3x10^{-8}$ ($4x10^{-5}$ to $8x10^{-5}$)	H	CD
			TEDC	$2x10^{-4}$ to $3x10^{-4}$ (0.5 to 0.7)	3 to 4 (35 to 42)	--	--	CJ	R
6	MM-10	M-Sand	TEDC	$2x10^{-4}$ to $3x10^{-4}$ (0.5 to 0.8)	3 to 4 (31 to 46)	$2.6x10^{-4}$	--	T,CJ,H	CD
			Authors	$1x10^{-4}$ to $2x10^{-4}$ (0.4 to 0.6)	2 to 4 (23 to 38)	$2x10^{-4}$	$1x10^{-9}$ to $2x10^{-8}$ ($3x10^{-6}$ to $5x10^{-5}$)	H	CD
			TEDC	$2x10^{-4}$ to $3x10^{-4}$ (0.5 to 0.6)	3 to 4 (31 to 36)	--	--	CJ	R
7	LAP-1	O-Sand	TEDC	$3x10^{-4}$ (0.8)	8 (83)	--	--	CJ	R

[a] M-Sand and N-Sand are the uranium-bearing sandstone units; Basal is the Basal Sandstone; O-Sand is the O Sandstone.

[b] TEDC = Teton Exploration Drilling Company.

[c] Values are based, in part, on the following assumptions: 1) the aquitard underlying the tested aquifer is impermeable, and 2) the specific storage of the overlying aquitard is two orders of magnitude larger than that calculated for the tested aquifer.

[d] T = Theis (1935); CJ = Cooper and Jacob (1946); H - Hantush (1960); Hh - Hantush (1966).

[e] CD = Constant discharge pumping test; R - Recovery test.

[f] Results are questionable or not valid because the test was conducted improperly or method(s) of analysis were used improperly.

Source: TEDC, 1979 and 1980.

TEDC (1980) compared all drawdown data curves for pumping tests 1, 2, 5, and 6 (Table A.25) to the modified leaky artesian type curves presented by Hantush (1960). Aquifer transmissivity and storativity were estimated by TEDC using the Theis (1935) and Cooper and Jacob (1946) methods of analysis. TEDC concluded that no vertical connections exist between aquifers at the project site. However, the authors found that the drawdown curves for the observation wells for aquifer tests 2, 5, and 6 matched the type curves of the Hantush (1960) solution reasonably well. The curve matches suggest that water leaked into the pumped aquifers (N and M sandstones) during the pumping tests. Table A.25 presents values of horizontal hydraulic conductivity for the N and M sandstones and values of vertical hydraulic conductivity for the overlying aquitard estimated by the authors using the Hantush (1960) modified leaky artesian method. Drawdown data from aquifer tests 5 and 6 (M Sandstone) match the type curves of the Hantush (1960) solution with much less certainty than the drawdown data from test 2 (N Sandstone).

Water level responses to pumping in respective overlying and underlying aquifers during aquifer tests 2 and 5 were limited to fluctuations of less than 0.2 m (0.6 ft). These fluctuations were similar to recorded changes in barometric pressure. A maximum water level decline of 0.5 m (1.5 ft) occurred in the aquifer underlying the M Sandstone during pumping test number 6. This decline may be a result of leakage because little barometric change occurred; however, it should not be used as conclusive evidence that the aquitards are leaky.

All of the aquifers of interest at the Leuenberger site appear to have similar values of hydraulic conductivity (Table A.25). The horizontal hydraulic conductivity of these aquifers (O, N, M, and Basal sandstones) ranges from about $2x10^{-4}$ to $4x10^{-4}$ cm/s (0.5 to 1.0 ft/d). The storativity of the N and M sandstones averages approximately $1x10^{-4}$. The direction of major transmissivity in the N and M sandstones was evaluated from aquifer tests 1 and 2, using the Hantush (1966) method of analysis, to be north-northwest (TEDC, 1980). Apparent barrier boundaries were detected during one of the M Sandstone pumping tests (number 6) and during the N Sandstone pumping test (number 2). These apparent boundaries appear to correspond with thinning of these units at the eastern edge of the ore deposit (TEDC, 1980).

Drawdown data for the three multiple-well pumping tests (2, 5, and 6) indicate that water probably leaks into the N Sandstone. Water may leak into the M Sandstone but it probably is much less significant. Drawdown data curves matched by the authors to the Hantush (1960) modified leaky artesian type curves indicate that the vertical hydraulic conductivity of the aquitard overlying the N Sandstone ranges from about $4x10^{-7}$ to $3x10^{-5}$ cm/s ($1x10^{-3}$ to $7x10^{-2}$ ft/d) (Table A.25). The aquitard overlying the M Sandstone appears to have a much smaller range of vertical hydraulic conductivity between approximately $1x10^{-9}$ and $3x10^{-8}$ cm/s ($3x10^{-6}$ to $8x10^{-5}$ ft/d).

Type curve matches to the Hantush (1960) solution suggest that water leaking into the aquifer may be derived from storage in the aquitards.

This phenomenon is to be expected in this hydrostratigraphic environment because the particular aquitards are from 15 to 52 m (50 to 170 ft) thick. Additional aquifer tests are needed at the mine site to better define leakage potential and aquitard characteristics. The ratio method developed by Neuman and Witherspoon (1972) is the only field method that can quantify the vertical hydraulic conductivity for each confining layer separately.

A.5.5 Hydrogeology of the Mine Site

The Lebo Member of the Fort Union Formation contains the uranium mineralization at the Leuenberger site. Two separate roll-front uranium deposits, one in each N and M sandstone units, exist at the project site. The N Sandstone lies above the M Sandstone at a depth of 67 and 82 m (220 to 270 ft) below land surface. The N ore zone ranges in thickness from 1.5 to 6 m (5 to 20 ft) within a total N Sandstone thickness of approximately 15 m (50 ft). The M Sandstone unit lies at a depth of 98 to 120 m (320 to 390 ft) beneath land surface. The thickness of the M Sandstone ranges from 15 to 20 m (50 to 65 ft); the M ore zone ranges in thickness from about 1.5 to 6 m (5 to 20 ft).

Uranium accumulation in the N and M sandstones occurred as a result of uranium migration downdip in Paleocene fluvial channel deposits until reducing conditions were encountered and the uranium was precipitated (TEDC, 1980). Sandstones of the Fort Union Formation at the Leuenberger mine site are fine- to coarse-grained with frequent intervals of conglomerate; the coarse-grained deposits represent high-energy systems. This environment may include alluvial fans or braided streams. The claystone units separating the sandstones are relatively thick and apparently continuous across the mine site, indicating lower-energy depositional systems; however, scour and fill deposits may cut through the units in places. The N and M sandstones pitch out to the east of the mine site.

The M and N sandstone units contain 74% and 26% of the minable uranium ore at the Leuenberger site, respectively. The small clay fraction within the M and N sandstones is montmorillonite and kaolinite (TEDC, 1980). Clay swelling during research and development testing did not cause significant reduction of the hydraulic conductivity of the sandstones. There is no apparent calcite, gypsum, or silica cementation of the N and M sandstones (TEDC, 1980).

A geologic column of the Lebo Member at the Leuenberger mine site is presented in Fig. A.43. The Lebo Member is composed of interbedded fine- to coarse-grained sandstone, siltstone, claystone, and coal. The Lebo Member crops out at the Leuenberger mine site and dips to the northeast about 2° to 3°. A simplified east-west cross section of the mine site is shown in Fig. A.44. No folds or faults have been reported at the Leuenberger site nor are any evident from geophysical log interpretations (TEDC, 1980). A surface geologic map of the region surrounding the Leuenberger site is shown on Fig. A.45.

The four principal confined aquifers at the Leuenberger site are the O_1, N, M, and Basal sandstone units (Fig. A.43). These sandstones are

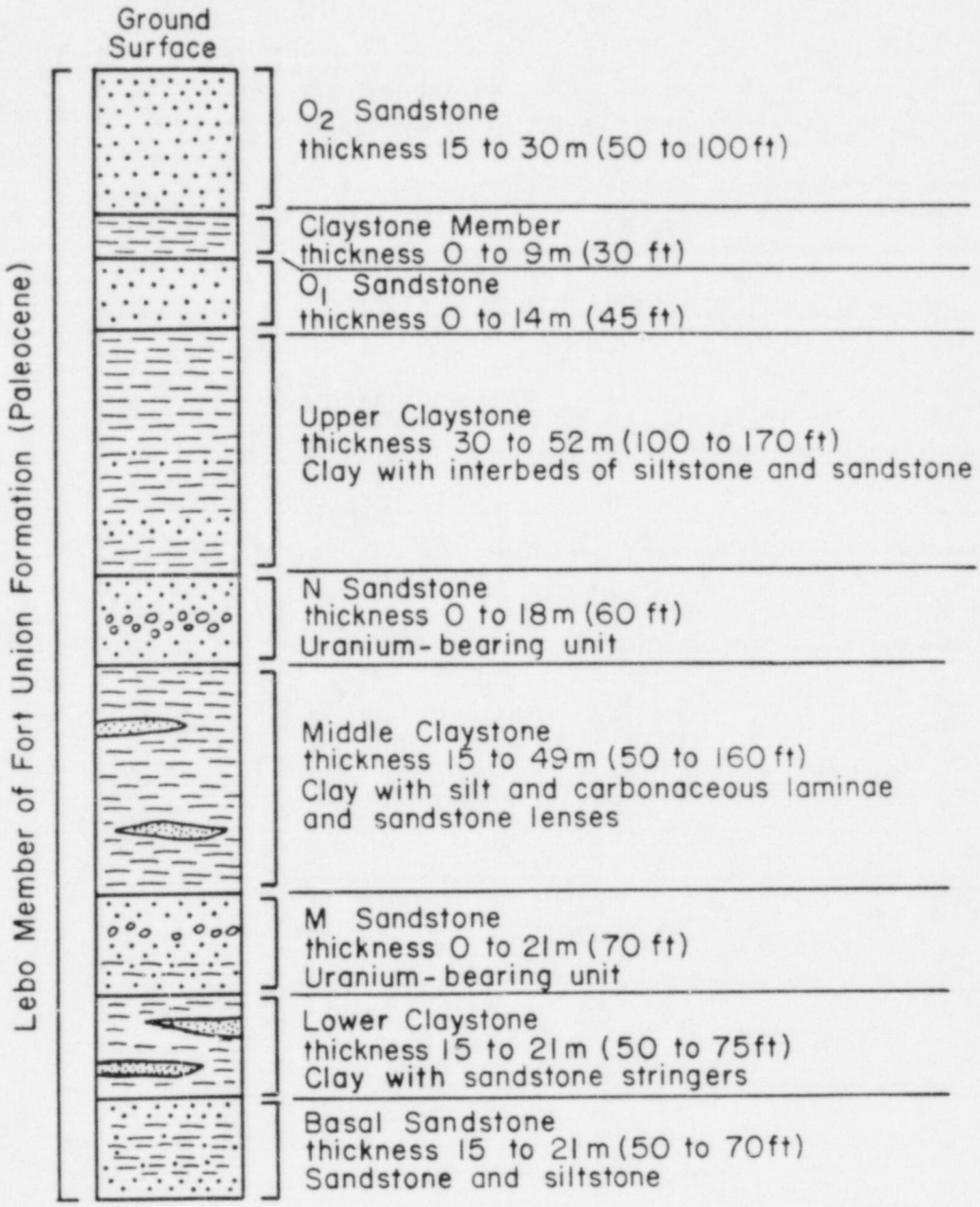


Fig. A.43. Stratigraphic section for the Leuenberger project area. *Source:* Modified from TEDC, 1979.

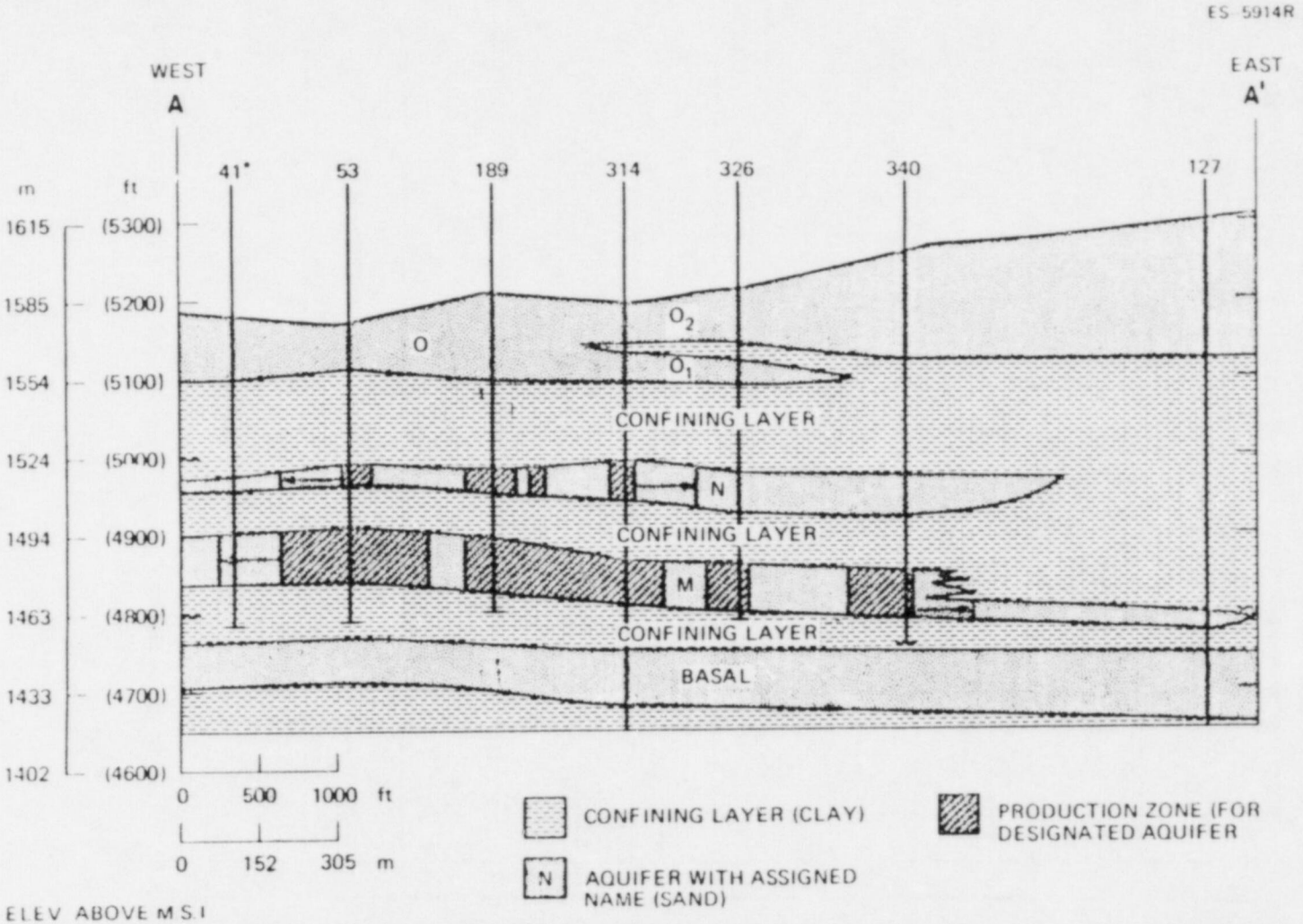


Fig. A.44. Simplified geologic cross section through the Leuenberger mine site. *Source:* NRC, 1982c.

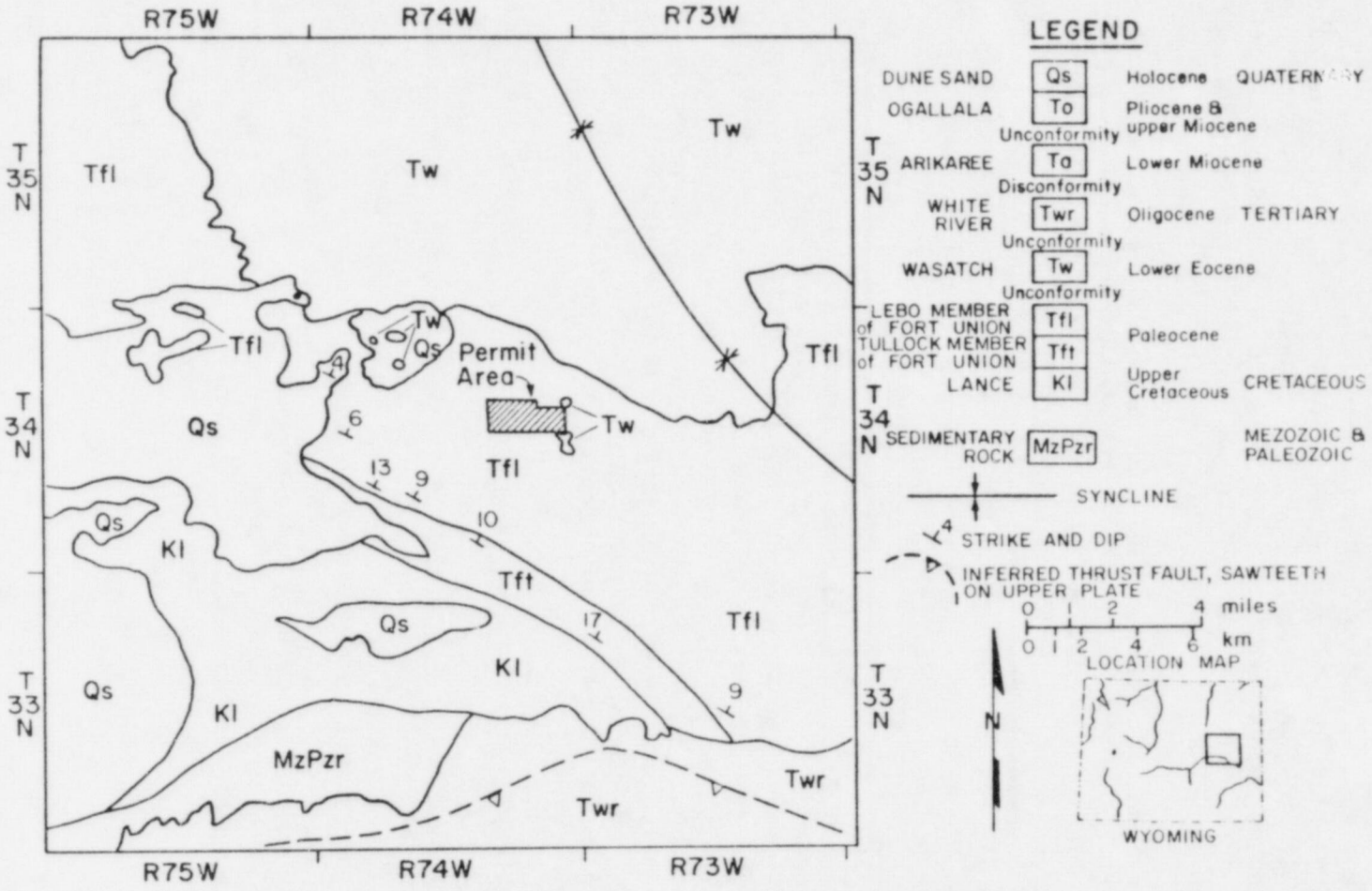


Fig. A.45. Surficial geology and structure map of the region surrounding the Leuenberger site. *Source:* Modified from TEDC, 1980.

interbedded among claystones of the upper portions of the Lebo Member. The O Sandstone is separated into the upper O_2 and lower O_1 members by a 0 to 9 m (0 to 30 ft) thick claystone. Depth to the water table in the O Sandstone varies from 0 to greater than 40 m (0 to 130 ft); the water table intersects the land surface within the permit area along Little Sand Creek. Ground-water flow in the O Sandstone is to the southwest as the water table conforms with the topography. The Upper Claystone divides the O Sandstone into an unconfined O_2 unit above the clay and a confined O_1 unit below the clay in the northern and eastern portions of the mine area. Ground-water flow in the N and M sandstones is northeasterly.

Potentiometric levels of the major aquifers prior to mining operations at the Leuenberger site indicated a downward direction of ground-water movement. The static head in the O Sandstone was approximately 11 m (35 ft) and 23 m (75 ft) higher than the static heads in the ore zone sands and the Basal Sandstone, respectively. The static head in the N Sandstone was about 0.9 m (3 ft) higher than that in the underlying M Sandstone. Water in the two ore zone sands moves horizontally down the formational dip toward the northeast (NRC, 1982c).

Baseline water quality of the M ore zone was evaluated by sampling wells 301, 306, 307, 308, 575, and 576 (Fig. A.41), in the M ore zone within Mining Unit I between January 10, 1979, and April 19, 1981 (NRC, 1983a). Research and development data indicate that ground-water quality in the N ore zone is similar to the M ore zone. Levels of ^{226}Ra in the ground water in the M and N ore zones in Mining Unit I exceed the drinking water standard of 5 pCi/L (Table A.26); however, baseline averages of other tested constituents are within or very close to meeting drinking water standards (NRC, 1983a). The position of the NRC is that the water quality must be restored to the degree necessary to conform with the highest potential premining use.

A.5.6 Monitoring Well System and Well Construction

Monitoring wells were placed in the N and M sandstones (ore zones) and in the aquifers above and below both ore zones (O and Basal sandstones, respectively). Monitoring wells utilized during the research and development operations were sampled every two weeks for excursion indicators chloride, alkalinity, sodium, sulfate, uranium, and conductivity. Except for sodium and sulfate, these indicators were proposed by TEDC (1980) for monitoring a commercial-scale operation at Leuenberger. These indicators were selected because 1) a sodium carbonate/bicarbonate lixiviant was used, 2) chloride was the elution ion (sulfate may be substituted for choride in the elution circuit), 3) conductivity is a measure of TDS, and 4) the uranium concentration of the formation water increases greatly during mining.

UCL's were established by TEDC (1979) for the excursion indicators during research and development operations for each monitor zone. These values were determined by multiplying a 90% t-distribution factor by the standard deviation of the indicators, and adding the baseline mean value (this procedure was not illustrated or explained by TEDC). If any two

Table A.26. Data used to establish preliminary mean baseline values[a] for mining unit I at the Leuenberger mine site.

Parameter	Well Number						Mean Baseline
	301[b]	306[c]	307[d]	308[e]	575[f]	576[g]	
Bicarbonate	214.7	234	234	142	217	224	211
Carbonate	0	0	0	0	0	0	0
Chloride	2.3	6.0	6.7	8	8.8	8	6.6
Sulfate	106.3	94.7	323	246	103	100	162
Anion Equivalent	5.81	5.98	10.76	7.67	5.96	5.98	7.02
Calcium	61.3	58.7	112	95	62.5	66	75.9
Magnesium	16	17.4	43.4	21	16.5	15.3	21.6
Sodium	26.7	29	26.3	22	27.5	28.6	26.7
Potassium	7	8.3	10.2	10	9.4	8.5	8.9
Cation Equivalent	5.74	5.86	10.62	7.68	5.95	6.04	6.98
-/+ balance	101.22	102.05	101.32	99.9	100.17	99.01	100.57
ΣTDS	434.3	448.1	755.6	544	444.7	450.4	512.7
TDS	330	341	700	197	350	350	378
Uranium	<.005	0.15	<.10	0.072	<.10	<.10	0.10
Aluminum	<.10	0.36	<.05	0.04	<.05	<.05	0.11
Ammonium	1.18	0.12	0.15	<.10	0.12	<.10	0.30
Arsenic	<.005	0.007	<.005	<.005	0.006	<.005	0.006
Barium	<.50	<.40	0.12	0.05	<.50	<.50	<.50
Boron	0.10	0.08	<.25	0.01	<.25	<.25	<.25
Cadmium	<.005	<.02	<.05	0.002	<.01	<.01	<.05
Chromium	<.02	<.10	<.05	0.02	<.05	<.05	<.10
Copper	<.01	<.09	<.05	<.01	<.05	<.05	<.09
Fluoride	0.51	0.48	0.43	0.51	0.49	0.50	0.49
Iron	0.26	1.31	0.11	0.10	0.19	0.12	0.35
Lead	<.02	0.06	<.05	0.02	0.017	<.05	<.05
Manganese	0.05	0.08	0.053	0.04	<.20	0.055	0.08
Mercury	<.001	<.001	0.003	<.0005	<.001	<.001	0.0013
Molybdenum	<.02	<.50	<.10	<.05	<.10	<.10	<.50
Nickel	<.01	<.10	<.05	<.02	<.05	<.05	<.10
Nitrate/Nitrite	0.013	0.45	0.10	<.05	0.16	0.11	0.15
Selenium	<.005	<.01	0.019	<.005	<.005	<.005	<.01
Vanadium	<1.0	<.05	<.10	<.05	<.10	<.10	<.10
Zinc	0.026	<.05	<.05	0.03	<.05	<.05	<.05
Radium-226 (pCi/L)	420	562	6.67	102	7.41	20.96	186.5
Thorium-230 (pCi/L)	NA	3.6	1.67	2.4	2.97	1.25	2.35
Gross A (pCi/L)	NA	NA	NA	NA	NA	NA	NA
Gross B (pCi/L)	NA	NA	NA	NA	NA	NA	NA
pH (unit)	NA	7.85	8.04	8.04	7.5	7.63	7.8
Conductivity (mhos/cm)	NA[h]	463	812	715	476	476	588

[a]Values are mg/L except as noted.
[b]Mean of 1/10/79, 1/10/79, and 2/16/79.
[c]Mean of 6/11/79, 6/11/79 and 5/13/80.
[d]Mean of 6/1/79, 10/10/80, and 1/14/81.
[e]5/31/79, one sampling.
[f]Mean of 2/29/79, 5/13/80, 7/16/80, 10/6/80, 1/13/81, and 4/6/81.
[g]Mean of 6/29/79, 5/13/80, 7/16/80, 1/4/81, and 4/6/81.
[h]Not Available.

Source: NRC, 1983a.

constituents exceeded the UCL's in a sample from a given monitoring well, and a second sample taken within 48 hours also exceeded the limits, excursion status was established.

TEDC (1980) proposed to set UCL's for the excursion indicators during commercial-scale operations as follows: 1) 1.2 times the highest representative baseline value for alkalinity and conductivity, 2) 10 mg/L above the highest representative baseline value for chloride, and 3) either 1.0 mg/L or baseline plus 1.0 mg/L if baseline for uranium is above 1.0 mg/L. If two indicators exceed their respective UCL's as confirmed by a second sample taken within 48 hours, an excursion status would be established. The NRC (1983a) modified the proposed UCL's for uranium in all ore zone monitor wells, and for chloride in all monitor wells. Uranium UCL's were modified to 20% greater than the highest representative baseline value measured from all ore zone monitoring wells. Chloride limits were modified to 20% greater than the highest representative baseline value for each monitoring well. For each vertical excursion monitor well (overlying and underlying aquifers) the UCL's for uranium would be determined by adding 20% to the highest value from three representative baseline samples from each well (NRC, 1983a). Because chloride is relatively mobile and stable in solution, it would be the leading excursion indicator. If the UCL for chloride is exceeded, an excursion status would be established. If the UCL of any single indicator is exceeded by more than 20% or if the UCL's are exceeded by any two indicators after routine and confirmation sampling, excursion status would be established. These UCL's were never implemented, however, since commercial production has not taken place.

Monitoring wells at the research and development test site were completed in the N and M sandstones at a distance of about 60 m (200 ft) from the well field boundary (Fig. A.41). One Basal Sandstone monitoring well and two O Sandstone monitoring wells were used at the test site. No trend wells were used.

TEDC (1980) proposed placing an individual ring of not less than six ore zone excursion monitoring wells approximately 180 m (600 ft) apart around each commercial mining unit and no more than 90 m (300 ft) from the mining unit perimeter. Figure A.46 shows the monitoring system for Mining Unit 1. Wells also would be completed in the O and Basal sandstones at a uniform density of one well per 2 ha (5 acres) of well field, or no less than one well in each aquifer per mining unit. In addition, the ore zone not being mined would be monitored while mining is occurring in the other ore zone, with wells completed in the N and M sandstones at a uniform density of one well per 0.8 ha (2 acres) of well field.

Wells at the Leuenberger site were drilled initially through the target zone using a standard rotary drill rig. The hole was logged geophysically, reamed to a larger diameter and cased with fiberglass or PVC pipe to the top of the selected interval. The outer annulus was cemented via weep holes at the base of the casing using a predetermined volume of cement slurry. After the cement hardened, the well was drilled out past the bottom of the casing and underreamed through the target sandstone using a "nonclogging" drilling mud. A well screen was lowered through the casing

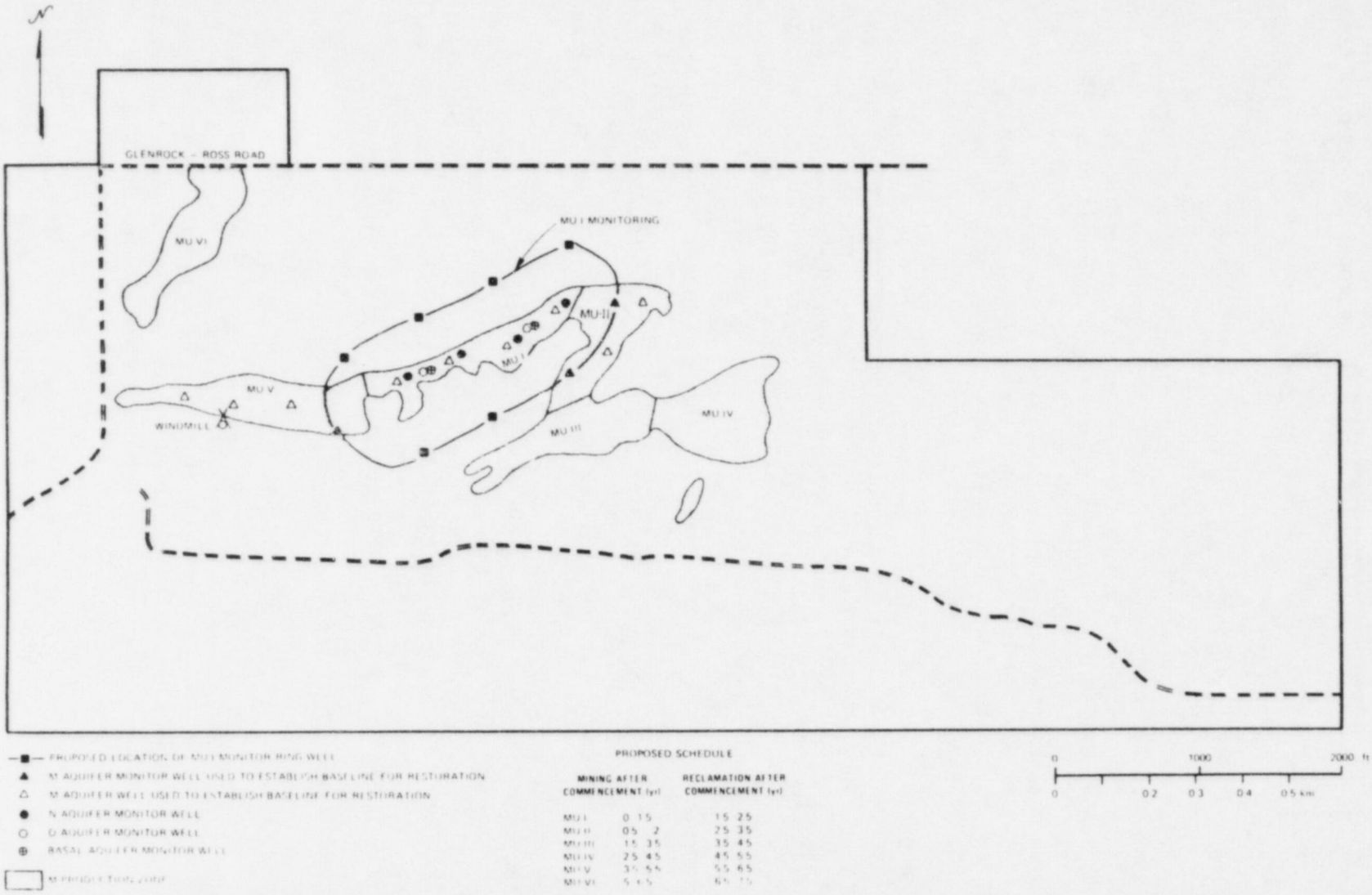


Fig. A.46. Map of the Leuenberger project area showing locations of mining unit I restoration baseline and excursion monitor wells. *Source:* NRC, 1983a.

and set with the use of unslotted (blank) casing and packers (Fig. A.47). Gravel packs were not installed. The well was developed by water jetting in combination with washing and air-bubbling cycles. High-pressure injection tests between packers were used to check the quality of well completion.

A.5.7 Analysis of Excursions

No legal excursions were reported during TEDC's 13 month research and development operations from January, 1980, through February, 1981. Several monitoring wells exceeded UCL's for two or more excursion indicators; however, on subsequent sampling dates the parameters in question declined below their UCL's. As a result, TEDC was never required to proceed with corrective measures to control possible lixiviant migrations. Only one monitoring well was completed in the aquifer underlying both ore zones (Basal Sandstone); two monitoring wells were completed in the aquifer overlying the ore zones (O Sandstone).

Monitoring well 309 which is completed in the N Sandstone (ore zone) slightly exceeded its UCL's for conductivity, sulfate, and alkalinity on February 6, 1980. Subsequent samples from this well indicated that the three excursion indicators declined below their UCL's and remained there. Well 309 is located about 76 m (250 ft) north of the N well field boundary and within 7.6 m (25 ft) of the northwest boundary of the M well field (Fig. A.41).

Basal Sandstone monitoring well 314 and M Sandstone (ore zone) monitoring well MM-2 exceeded UCL's on one or more sampling dates in April and May, 1980. Well 314 exceeded UCL's for chloride, sulfate, and/or alkalinity for three nonconsecutive sampling dates. Well MM-2 exceeded UCL's for sulfate and sodium on one sampling date only. TEDC failed to realize that three indicators exceeded UCL's in well 314 at the time of occurrence; by the time the water quality data were evaluated, concentrations of the excursion indicators had already dropped below their UCL'S (TEDC, 1980). No corrective actions were initiated for these apparent excursions. Monitoring well MM-2 is located down-gradient from the well field with respect to natural ground-water flow in the M aquifer; this suggests that the well may have detected some lixiviant migration. Additional reasons for the elevated concentrations cannot be evaluated due to the lack of data.

A.5.8 Restoration

Restoration tests were conducted at the two research and development areas (M and N well fields) between June, 1980, and December, 1981, at the Leuenberger site (Fig. A.41). A complete discussion of restoration attempts at the two well fields is included in NRC (1983a) and TEDC (1982). On June 1, 1980, TEDC initiated a five-step ground-water restoration program at the N well field. Table A.27 presents restoration data at the end of each step.

Step 1 was conducted from June 1 to July 12, 1980, and consisted of circulating approximately 14 aquifer pore volumes of contaminated ground

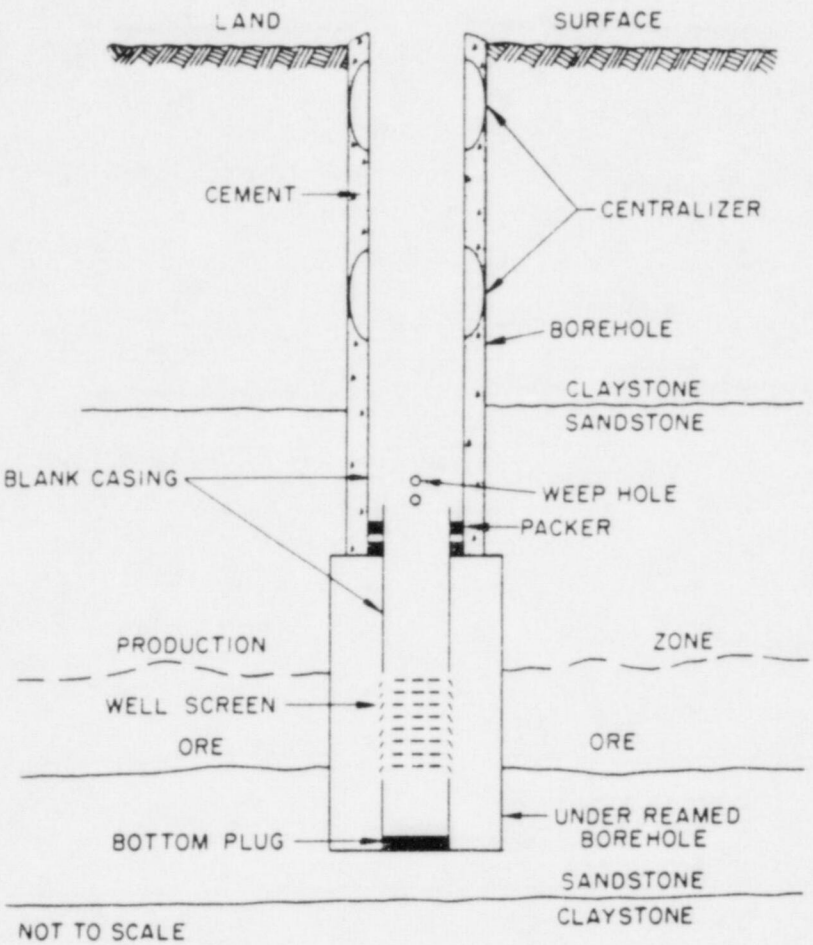


Fig. A.47. Schematic of a typical injection and recovery well completion at the Leuenberger project site. *Source:* NRC, 1983a.

Table A.27. Water quality[a] in wells before, during, and after N-zone restoration at the Leuenberger mine site.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Leaching/ Prerestoration[c] 6/01/80	End of Step 1[c] 8/09/80	End of Step 2[c] 8/18/80	End of Step 3[c] 9/19/80	End of Step 4[c] 10/21/80	After Step 5[c] 11/06/80[d]	Post Restoration[c] 10/05/81
Chloride	250	24	52	44	26	15	16	10	6
Bicarbonate	500	173	1342	936	672	444	432	312	318
Sulfate	250	246	461	391	316	266	286	292	293
Uranium (U_3O_8)	5.0	0.54	28.4	6.8	3.9	3.2	3.0	2.9	1.4
TDS	500	530	2500	--	1066	--	--	585	720
Conductivity (µmhos/cm)	--	--	2010	1480	1135	875	1040	885	1014

[a]Values are mg/L except as noted.

[b]Average values from all N Well Field wells.

[c]Values represent samples from unknown wells.

[d]Step 5 ended November 1, 1980.

Source: NRC, 1983a.

water through the process plant to remove residual uranium. Step 2 occurred from August 10 through 18, 1980, and consisted of injecting about 4.5 aquifer pore volumes of clean ground water withdrawn from the M well field in an effort to dilute concentrations of N zone chemicals.

Ground-water sweeping was utilized in Steps 3, 4, and 5. Step 3, from August 21 to September 17, 1980, consisted of pumping production well NR-1 (Fig. A.41) at 2.5 L/s (40 gal/min) and discharging a total of 11.5 aquifer pore volumes directly to evaporation ponds. A total of 6.6 aquifer pore volumes of ground water was pumped from injection well NI-3 in Step 4 in one month ending October 21, 1980. About 5.3 aquifer pore volumes were pumped from injection wells NI-1 and NI-3 in Step 5 between October 21 and November 1, 1980. Table A.27 shows that most of the water quality improvement occurred in Steps 1 and 2 with little or no improvement after Step 3.

Major ionic constituents exceeding baseline or drinking water standards at the end of restoration (November, 1980) in the N well field were bicarbonate, sodium, and sulfate (Table A.28). Radium-226 exceeded applicable standards before mining and after restoration. Table A.28 shows that concentrations of bicarbonate, calcium, sodium, ^{226}Ra, and TDS in the ground water have increased since restoration was completed. However, the N well field is considered to have been restored successfully to the highest potential premining use (NRC, 1983a). Approximately 24 aquifer pore volumes of ground water were removed during N well field restoration. All of the water was discharged to evaporation ponds. No water treatment units were used throughout the N well field restoration period.

Four-phase restoration of the M well field was conducted from February through December, 1981. Table A.29 presents water quality data for the M well field restoration. Results between phases are not directly comparable because different sets of wells were sampled to establish: 1) baseline, 2) prerestoration, 3) restoration progress, and 4) post-restoration conditions. Although conclusions based on these data are only tentative with respect to baseline conditions, it is clear that each successive restoration phase improved the ground-water quality over the preceeding phase. Phase 1 of M zone restoration was initiated February 25, 1981, with routine well field operation without chemical addition, and with partial removal of recovered constituents via ion exchange and electrodialysis. According to TEDC (1982), Phase 1 was designed to reduce chemical parameters within the well field, remove uranium and other constituents, perform operational tests on the electrodialysis unit and develop a comprehensive restoration plan. Approximately 6.6 aquifer pore volumes of ground water were circulated until the end of Phase 1 on April 20, 1981. Rapid improvement of ground-water quality in the well field was noted during Phase 1.

Phases 2, 3, and 4 utilized a clean water recirculation method in which ground water was pumped from various well combinations, treated using ion exchange and electrodialysis, and reinjected into the well field. TEDC utilized a directional sweep approach which was designed to remove solution external to the well patterns. Pumping began in the southeast part of the well field during Phase 2 and progressed to the northwest during Phases 3 and 4. Water was reinjected "behind" the affected areas as restoration

Table A.28. Ground-water restoration-test water quality data[a] in the N ore zone for the Leuenberger mine site.

Parameter	Baseline Water Quality	Livestock Criteria[b]	Domestic Criteria[b]	Post Leaching/ Prerestoration June 1980	Post Leaching Water Quality		
					End Restoration[c]	Post Restoration	
					November 1980	January 1981	January 1982
Ammonia (NH_3 as N)	0.27±0.41[d]				0.11±0.03	0.17±0.08	0.11±.13
Total Nitrate/Nitrite	0.46±0.76	1/10	1/10		0.18±0.03	<0.1±0	0.25±.20
Bicarbonate	172.5±34.2			1342	207.8±30.7	213.8±33.5	237±32
Carbonate	1.7±3.8				0±0	0±0	0
Calcium	93±28				93±4.7	94±1.7	103±5.7
Chloride	23.9±28.2	2000	250	52	6±2.12	6±1.2	4±2.3
Boron	0.02±0.02	500	0.75		<0.25±0	<0.25±0	<0.01
Fluoride	0.46±0.11		1.4-2.4		0.37±0.08	0.31±0.13	0.42±0.07
Magnesium	18.6±8.7				23.4±5.03	21.9±1.6	21.0±3.0
Potassium	30.9±36.4				13.1±1.7	12±2.1	12±1.1
Sodium	33.6±6.6				43.8±9.6	43.2±13.9	50±9.4
Sulfate	246.4±33.9	3000	250	461	266.2±8.3	264.2±8.3	259±13.2
Aluminum	0.07±0.04	5.0			<0.05±0.01	<0.05±0	<.05
Arsenic	0.01±0	0.2	0.05		<0.005±0	0.006±0.002	<.005
Barium	0.05±0.02		1.0		<0.1±0	0.13±0.05	<.03
Cadmium	0.01±0.02	0.05	0.01		<0.01±0	<0.01±0	<.002
Chromium	0.03±0.02	0.05	0.05		<0.05±0	<0.05±0	<.01
Copper	0.02±0.02	0.50	1.0		<0.05±0	<0.05±0	<.01
Iron	0.20±0.23		0.30		0.62±0.57[e]	0.48±0.36[e]	<.01
Lead	0.02±0.02	0.10	0.05		0.05±0.01	0.05±0.004	<.01
Manganese	0.04±0.02		0.05		0.06±0.01	0.21±0.33	0.03±.02
Mercury	0.01±0.02	0.00005	0.002		<0.001±0	<0.001±0	<.0005
Molybdenum	0.07±0.02				<0.1±0	<0.1±0	<.05
Nickel	0.03±0.01				<0.05±0	<0.05±0	<.02
Selenium	0.01±0	0.05	0.01		0.008±0.006	<0.005±0	<.005
Uranium	0.54±0.76	5.0	5.0	28.4	0.64±0.53	0.52±0.39	0.50±39
Vanadium	0.14±0.20	0.10			<0.1±0	<0.1±0	<.05
Zinc	0.02±0.02	25	5		<0.05±0	<0.05±0	<.05
TDS	530±39			2500	577±36	582±34	608±23.6
Radium-226 (pCi/L)	185±242	5.0	5.0		252.42±318	346±140	368±77
pH (units)	8.0±0.38	6.5-8.5	6.5-8.5		7.9±0.1	7.4±0.4	7.3±0.1

[a]Values are mg/L except as noted.

[b]Criteria are based on water quality standards given in the Wyoming Water Quality Rules and Regulations, Chapter 8, Table 1, Apr. 9, 1980. A blank space signifies that no criteria have been established.

[c]Average values for the final round of restoration sampling. Sampling data is not known.

[d]Plus or minus standard deviation.

[e]Unfiltered sample.

Source: NRC, 1983a.

Table A.29. Water quality[a] in wells before, during, and after M-zone restoration at the Leuenberger mine site.

Parameter	Wyoming or EPA Drinking Water Stds.	Preleaching Baseline[b]	Post Leaching/ Prerestoration[c] 2/23/81	End of Phase 1[d] 4/21/81	End of Phase 2[d] 8/10/81	End of Phase 3[d] 9/01/81	End of Phase 4[d] 12/20/81	Post Restoration 1/08/82[e]	Post Restoration 6/82[e]	Post Restoration 12/82[f]
Chloride	250	5.7±2.0	78	68	19	25	11	5.8±3.4	7.5±2.2	7.5±2.1
Bicarbonate	500	223±28	1418	665	426	482	322	228±56	251±35	244±40
Sulfate	250	129±66	394	208	118	124	91	103±71	115±73	130±72
Sodium	--	27±2	440	212	102	133	79	53±19	55±18	60±18
Uranium (U_3O_8)	5.0	<0.10	21.8	8.3	5.7	4.5	1.0	0.02±1.03	0.37±0.36	0.58±0.60
TDS	500	381±108	1807	1208	721	868	553	349±118	388±97	384±97
Conductivity (μmhos/cm)	--	538±102	2535	1338	940	974	693	515±134	639±118	651±125

[a]Values are mg/L except as noted.

[b]Average values from wells 301, 305, 306, 307, 308, 575, 576, and MM-1, 2, 3, 4, 5, 6, 7, 9, 10.

[c]Average values from wells MR-1, MR-3, MR-5, 301, 306, and 308.

[d]Average values from wells MI-1, MI-3, MI-8, MI-12, MR-1, MR-3, MR-5, and 301.

[e]Average values from wells MI-1, MI-6, MI-10, MR-1, MR-3, MR-5, 301, 306, and 308.

[f]Fourteen wells within or near restored aquifer including those underlined in footnote b.

Source: TEDC, 1982a.

progressed. Pumping periods for Phases 2, 3, and 4 lasted 111 days, 22 days and 111 days, respectively. Approximately 7.5, 1.5, and 7.6 aquifer pore volumes of ground water were recirculated during Phases 2, 3, and 4, respectively. The overall M well field restoration program was terminated on December 20, 1981. Table A.30 presents water quality data for the M well field after the completion of restoration. The latter data show slight but statistically insignificant deterioration in water quality (with the exception of ^{226}Ra) during the post-restoration period. It is clear from the data presented that effective restoration of the M field has been achieved.

A total of 23.3 aquifer pore volumes of ground water were recovered and treated for uranium removal during the four-phase M well field restoration project. The completeness of restoration was demonstrated both by the wells generally achieving the appropriate water quality standards and by the recovery of nearly 90% of the chloride introduced during mining. Chloride was reduced from an average of 78 to 11 mg/L during the M well field restoration. The M well field also is considered to have been restored successfully to the highest potential premining use (NRC, 1983a). TEDC has been issued a permit to mine uranium commercially at the Leuenberger site.

Use of electrodialysis as a treatment process appeared to be effective at the M well field restoration project. Electrodialysis also reduced significantly the consumptive use of ground water during restoration. TEDC (1982) chose to use electrodialysis because of ease of operation, mechanical availability, comparatively low capital and operating costs, the unit's ability to provide 80% salt rejection, and the cooperation of the manufacturer. Compared to reverse osmosis treatment, electrodialysis uses less chemical pretreatment and filtration, can be operated in large pH and temperature ranges, and the membranes are more stable and accessible (TEDC, 1982). While both reverse osmosis treatment and electrodialysis theoretically are capable of restoring ground water to premining or better conditions, choice of one method over the other primarily is dependent upon economics and availability.

It is significant to note that TEDC recovered 23.3 pore volumes of contaminated ground water during restoration of the M well field. Of this water, 21.5 pore volumes of improved water were reinjected. Only 1.8 pore volumes amounting to 6.43×10^3 m^3 (1.7×10^6 gal) of water were disposed of as waste water. The restoration technique used for the M well field was better than 90% effective in conserving ground water (NRC, 1983a). TECD recovered 24.12 pore volumes of ground water during restoration of the N well field; this water was untreated and placed in solar evaporation ponds.

TEDC (1980) proposed to accomplish restoration at a commercial-scale project through a combination of ground-water sweeping and clean water recirculation. Aquifer restoration would be a continuous operation during and after mining and would include an electrodialysis unit to improve the water quality. The reader is referred to TEDC (1980) for further details regarding commercial-scale restoration plans.

Table A.30. Ground-water post-leaching water quality data[a] in the M ore zone of the Leuenberger mine site.

Parameter	Baseline Water Quality	Livestock Criteria[b]	Domestic Criteria[b]	Post Leaching/ Prerestoration February 1981	Post Restoration Water Quality		
					January 1982	June 1981	December 1982
Ammonia	0.24±0.35[c]				<0.5	-	≤0.104±.047
Total Nitrate/Nitrite	0.20±0.12	1/10	1/10		0.10±0.08	0.08±0.03	≤0.036±.0.030
Bicarbonate	233±27.9			1418	228±56.2	251.1±34.6	243±40.1
Carbonate	0±0				0	1.9±3.8	1.0±2.7
Calcium	70.5±13.4				54.6±31.4	61±29	60.9±22.4
Chloride	5.7±2.0	2000	250	78	5.8±3.4	7.5±2.2	7.5±2.1
Boron	<0.25	500	0.75		<0.01	0.15±0.07	0.22±0.08
Fluoride	0.49±0.05		1.4-2.4		0.37±0.14	0.34±0.17	0.25±0.09
Magnesium	18.6±7.3				10.4±6.1	13.8±6.1	15.3±5.9
Potassium	8.9±1.1				5.6±1.9	7.1±1.5	6.8±1.6
Sodium	27.0±1.9			440	52.7±18.8	54.5±18.2	60.0±17.8
Sulfate	128.9±65.5	3000	250	394	103±71.4	114.6±73.1	129.5±71.5
Aluminum	0.14±0.18	5.0			<0.05	0.05	≤0.051±0.003
Arsenic	0.009±0.010	0.2	0.05		<0.005	≤0.021±0.019	≤0.033±0.032
Barium	<0.50		1.0		<0.03	<0.10	<0.10
Cadmium	<0.05	0.05	0.01		<0.002	<0.01	<0.10
Chromium	<0.10	0.05	0.05		<0.01	<0.05	<0.05
Copper	<0.05	0.50	1.0		<0.01	<0.05	0.05
Iron	0.60±0.56		0.30		<0.01	<0.13±0.14	<0.24
Lead	0.04±0.05	0.10	0.05		<0.01	<0.05	<.05
Manganese	0.08±0.04		0.05		0.016±0.008	0.09±0.02	≤0.07
Mercury	0.001±0.0005	0.00005	0.002		<0.0005	<0.0002	≤0.0002
Molybdenum	<0.50				<0.5	<0.10	<0.10
Nickel	<0.05				<0.02	<0.05	0.05
Selenium	0.006±0.004	0.05	0.01		<0.005	<0.0012±0.0007	0.005
Uranium	<0.10	5.0	5.0	21.8	0.020±0.029	0.37±0.36	0.58±0.60
Vanadium	<1.0	0.10			<0.05	<2.0	<0.10
Zinc	0.092±0.110	25	5		<0.005	<0.01	<0.01
TDS	381±108			1807	348.9±117.5	388±97	383.9±97.4
Radium-226 (pCi/L)	235.96±335.4	5.0	5.0		352.9±394.3	240±250	280±273
pH (units)	7.6±0.32	6.5-8.5	6.5-8.5		7.36±0.32	7.71±0.31	7.83±0.24

[a]Values are mg/L except as noted.

[b]Criteria are based on water quality standards given in the Wyoming Water Quality Rules and Regulations, Chapter 8, Table 1, Apr. 9, 1980. A blank space signifies that no criteria have been established.

[c]Plus or minus standard deviation.

Source: NRC, 1983a.

A.6 ROCKY MOUNTAIN ENERGY COMPANY - RENO CREEK PROJECT

A.6.1 Description of the Site

Rocky Mountain Energy Company (RMEC) operates the Reno Creek research and development in situ uranium mine in Campbell County, approximately 110 km (70 mi) northeast of Casper, Wyoming (Fig. A.48). Topography of the mine site is dominated by grass covered plains and low-lying hills dissected by broad stream valleys. Elevation at the mine is about 1590 m (5200 ft) with about 12 m (40 ft) of relief. The mine site lies along a north-south trending drainage divide which separates the Cheyenne River and Belle Fourche River basins. The Belle Fourche River located about 14.5 km (9 mi) north of the mine is the nearest perennial stream. Runoff from the eastern portion of the site flows towards the Cheyenne River, and runoff from the western portion flows toward the Belle Fourche River. Runoff is minimal, however, because of low precipitation and high evapotranspiration rates. Annual precipitation ranges between 30 and 38 cm (12 and 15 in).

Research and development testing was conducted at the Reno Creek mine (Fig. A.49) in two well patterns. Leaching of the first well field (Pattern I) was conducted from February to November, 1979. Subsequently, restoration efforts were initiated and they continued until March, 1981. The second well field (Pattern II) was leached from October to December, 1980, and restoration activities were conducted from December, 1980, to April, 1981.

A.6.2 History of Mining Operations

RMEC initiated research and development activities at Reno Creek to test modified mining techniques in a geologic setting different from that of their previous in situ project at Nine Mile Lake. The relatively deeply buried, low grade ore body at Reno Creek presented an opportunity to investigate the use of in situ techniques involving different lixiviants to evaluate the viability of commercial-scale mining.

Research and development testing at Reno Creek began in February, 1979. Pattern I is a five-spot well pattern consisting of a central production well surrounded by four injection wells (Fig. A.50). Each injection well is located about 12 m (40 ft) from the production well. The wells are completed in the Wasatch Formation, with one or more perforated intervals corresponding to horizons of mineralization within the ore-bearing sandstone. Approximately 90,700 kg (200,000 lb) of lixiviant at about 5 g/L of H_2SO_4 (pH 1.8) were injected at about 2.5 L/s (40 gal/min) into the ore zone during the test. Hydrogen peroxide (H_2O_2) was utilized as the oxidant. Further details pertaining to the chemical characteristics of the sulfuric acid lixiviant are not available.

Uranium was recovered from the produced fluid by ion exchange. Uranyl tricarbonate ions were stripped from the ion exchange resin in the elution circuit. Precipitation of uranium was then induced by the addition of ammonium (NH_4^+), and the resultant yellowcake slurry was shipped to an

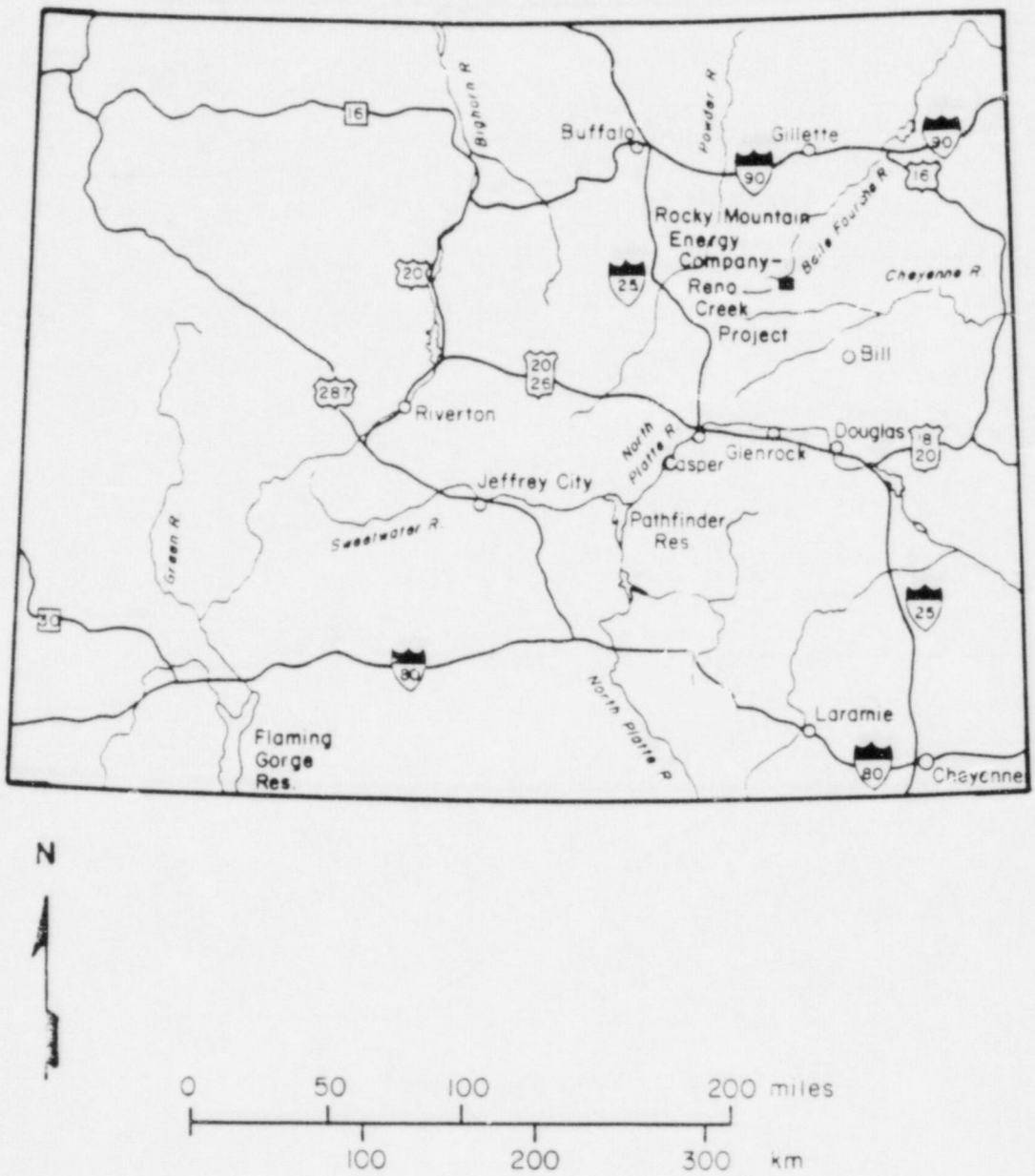


Fig. A.48. Location map of the Rocky Mountain Energy Company Reno Creek mine.

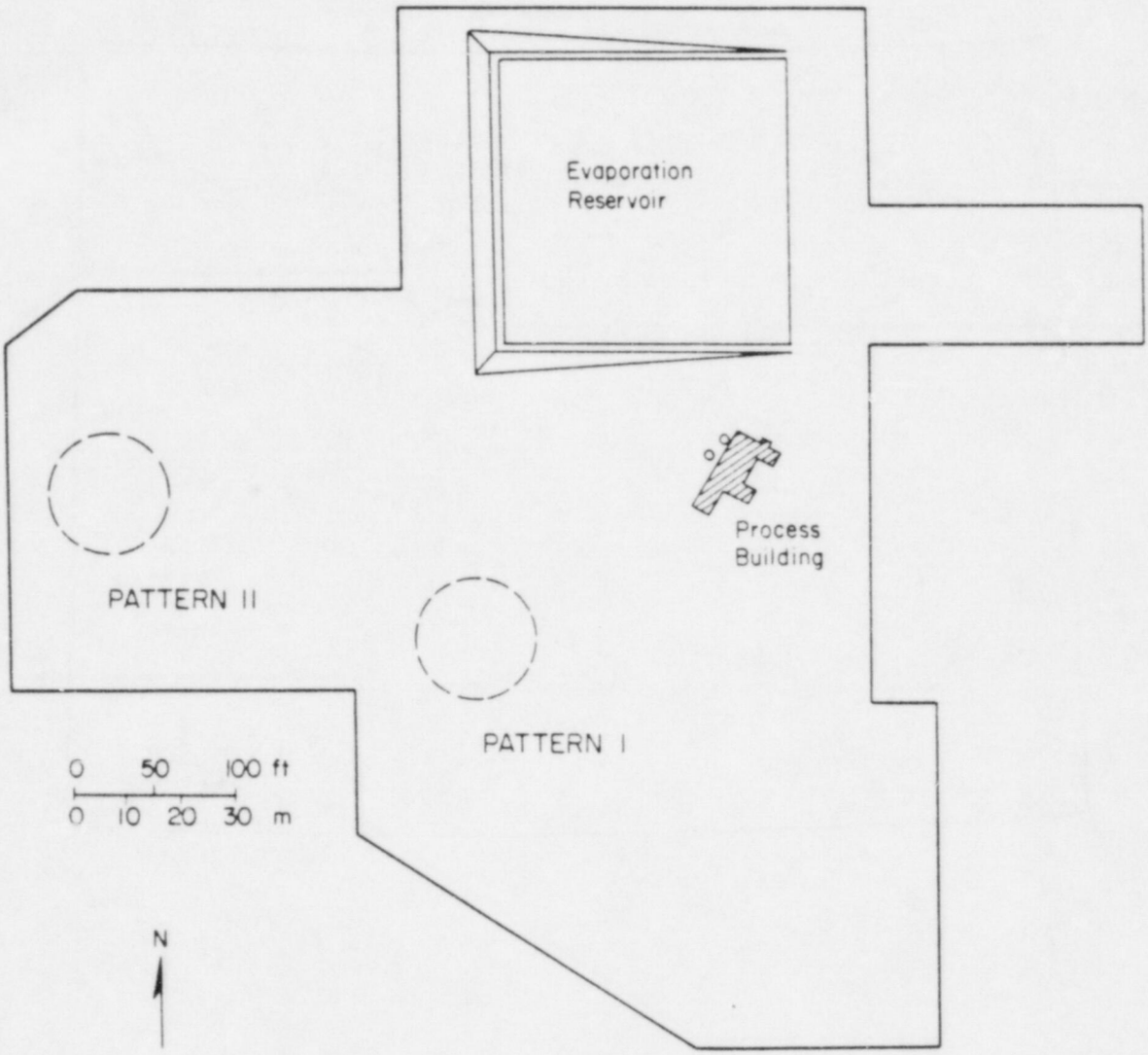


Fig. A.49. Reno Creek mine plan. *Source:* RMEC, 1978a.

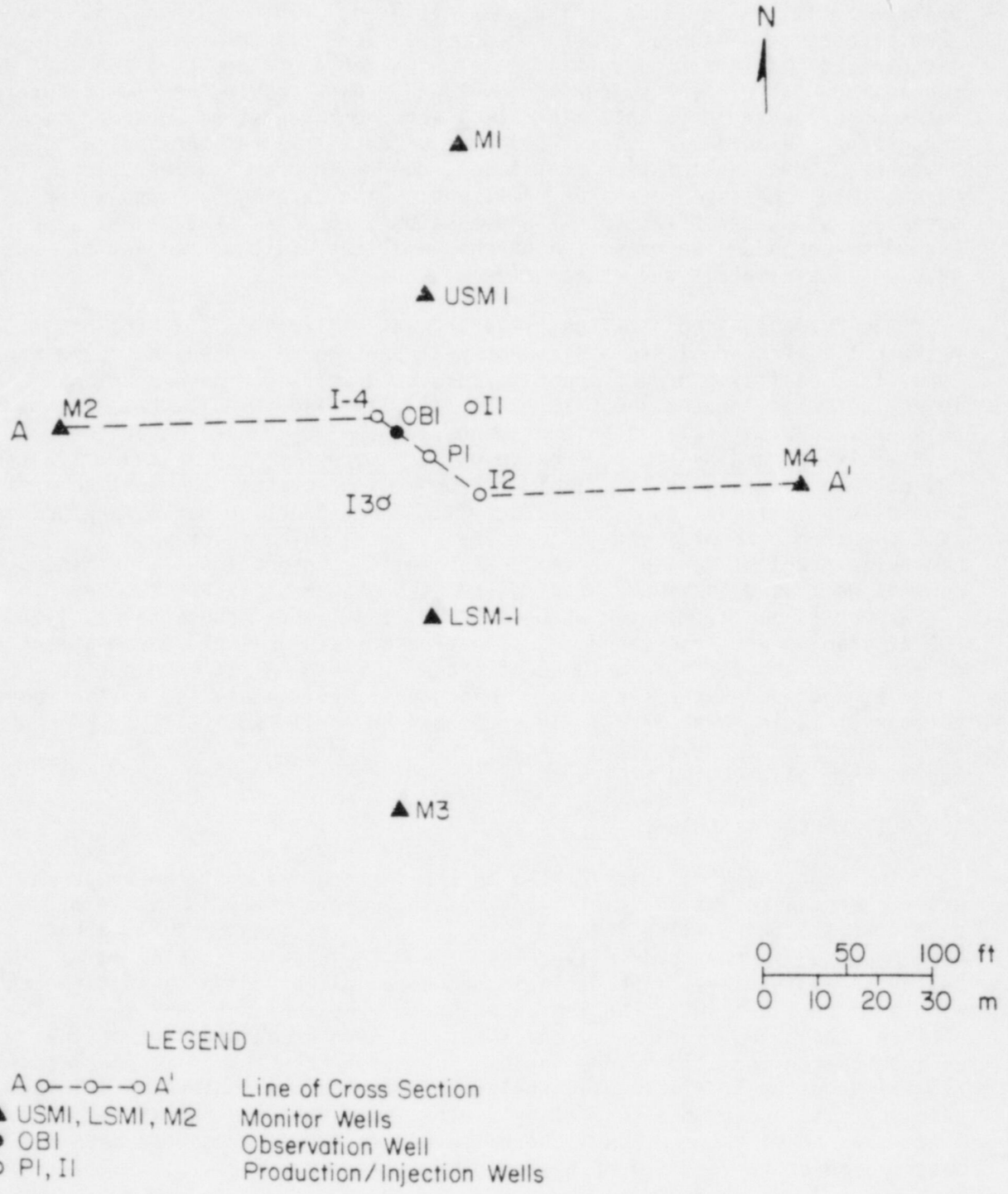


Fig. A.50. Location map of wells in Pattern I at the Reno Creek mine. *Source:* RMEC, 1978b.

existing mill. Soon after mining was initiated, problems with gypsum precipitation and "fungus growth" reduced the efficiency of well field circulation. Neither the details nor the evidence are provided for the conclusion that a "fungus growth" developed. As a result, uranium recovery rates were low and carbonate minerals in the host sandstone consumed high amounts of the acid lixiviant. Leaching of Pattern I was terminated in November, 1979. Restoration activities, conducted from November, 1979, to March, 1980, consisted of water treatment by ion exchange, ground-water sweeping, and water treatment with potassium carbonate. Potassium carbonate was added to raise the pH and facilitate further removal of calcium, heavy metals and radionuclides.

A carbonate-based alkaline lixiviant was utilized in leaching of Pattern II. Pattern II is a six-spot well pattern completed in the Wasatch Formation consisting of two central production wells surrounded by four injection wells located about 15 m (50 ft) away (Fig. A.51). Leaching of this pattern was initiated in September, 1980. Sodium carbonate/bicarbonate lixiviant and hydrogen peroxide oxidant were injected at about 1.3 L/s (20 gal/min) in Pattern II. Table A.31 shows the typical chemical composition of the lixiviant used in Pattern II. The production wells were pumped at a combined rate of about 1.6 L/s (25 gal/min) which maintained a hydraulic gradient toward the production wells. Adsorption ion exchange columns were used to remove uranium from the pregnant lixiviant. Leaching of Pattern II was terminated on December 2, 1980, after about 544 kg (1200 lb) of uranium were recovered. Restoration activities included treatment of about two aquifer pore volumes of produced fluid by ion exchange followed by a ground-water sweep which produced approximately 4.5 aquifer pore volumes of fluid. Pattern II was shut down on April 16, 1981.

A.6.3 Regional Setting

A.6.3.1 Regional Geology

The Reno Creek mine is located on the eastern flank of the Powder River Basin on relatively flat lying strata approximately 40 km (25 mi) east of the Irigaray site (Fig. A.5). No major faulting or folding has been reported in the Powder River Basin, and the area is considered tectonically inactive. The geologic sequence in the region surrounding the mine site includes units ranging in age from Precambrian to Holocene. The surface strata and uranium-bearing strata at Reno Creek are part of the Eocene Wasatch Formation. The Wasatch Formation consists of interbedded fluvial sandstones, siltstones, shales, claystones and coal seams. Thick sand deposits occur in a central belt with facies grading outward to finer-grained flood deposits. Sharp and Gibbon (1954) present a detailed description of the geology of the southern Powder River Basin.

A.6.3.2 Regional Ground-water Flow Systems

The Wasatch Formation is the major water-bearing unit in the area of the Reno Creek mine. Both confined and unconfined aquifers are present, although unconfined aquifers occur only at shallow depths (RMEC, 1978a).

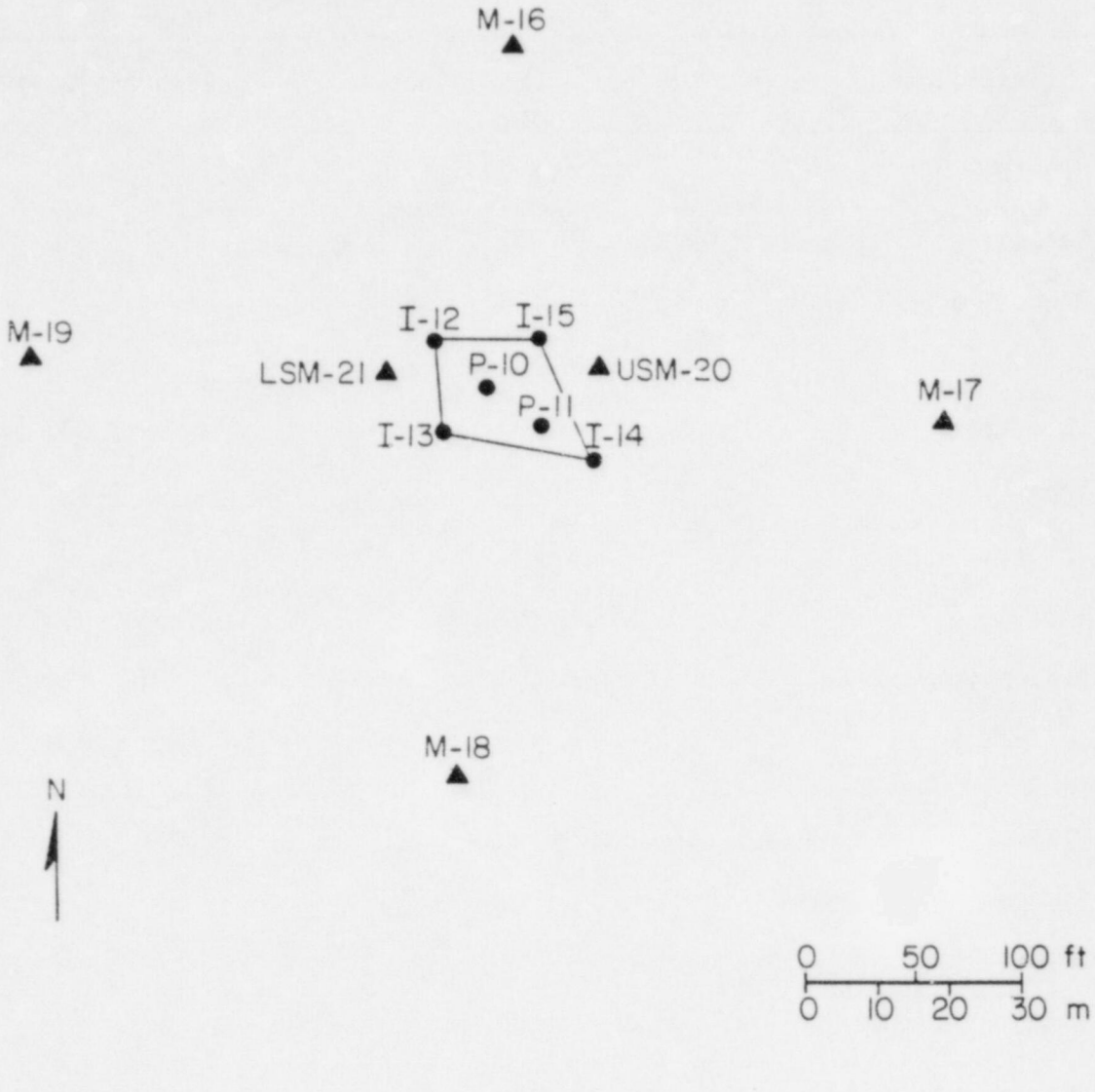


Fig. A.51. Location map of wells in Pattern II at the Reno Creek mine. *Source:* RMEC, 1980c.

Table A.31. Characteristics[a] of lixiviant used in leaching Pattern II at the Reno Creek mine.

Constituent	Injection Lixiviant	Pregnant Lixiviant
Uranium (U_3O_8)	0.1-2.0	50-200
Calcium	80-250	80-250
Magnesium	50-100	50-100
Sulfate	50-150	50-150
Chloride	50-200	50-200
Iron	1-20	1-2
Vanadium	1-20	1-30
Sodium	1500-2500	1000-2000
Bicarbonate	1000-4000	750-12500
pH (units)	6.0-8.5	6.0-8.0

[a]Values are mg/L except as noted.

Source: RMEC, 1980c.

Ground-water potential increases with depth in the vicinity of the mine site; some flowing wells are reported in low valley areas.

The Wasatch Formation typically yields water from the sandstone strata. However, thin, jointed coal seams and clinker beds may yield significant quantities of water also. Yields from wells in the local area of the Reno Creek mine range from 0.3 to 2.5 L/s (5 to 40 gal/min). RMEC (1978a) reported that properly constructed wells open to a sufficient thickness of an aquifer may yield up to 3.1 L/s (50 gal/min).

Recharge of the Wasatch aquifers occurs primarily from the infiltration of snowmelt and runoff from local precipitation events. Infiltration of ponded water and ephemeral streamflow occurs in areas underlain by coarser-grained sands and clinker deposits. It was reported by In-Situ Consulting, Inc. (1978b) that natural discharge occurs principally as small springs and seeps along the valleys of the Powder and Little Powder rivers. RMEC (1978a) reported that some water from the Wasatch Formation also moves downward and recharges underlying formations. Stratification of the Wasatch Formation and incised major stream drainages are factors controlling regional ground-water flow.

Regional ground-water flow generally is toward the north in the Powder River Basin, but ground water in shallow aquifers may discharge to smaller drainages. Aquifer heterogeneity and anisotropy appear to influence subsurface flow. The reader is referred to Hodson et al (1973) for a general description of the hydrostratigraphic units in the Powder River Basin.

A.6.4 Description and Analysis of Data Collected

A.6.4.1 Exploration

Exploration boreholes were drilled at the Reno Creek mine site by RMEC to define the ore body and evaluate the subsurface stratigraphy. Little information concerning the details of this work is available. Several mineral resource development projects are located nearby, such as the Wyoming Mineral Corporation Irigaray in situ uranium mine. Stripper oil wells have operated in the Manning Oil Field which surrounds the test site. Therefore, it is reasonable to expect that other companies have drilled exploration boreholes in the Reno Creek area. It is not known whether these boreholes were plugged properly. Unplugged boreholes may facilitate vertical migration of lixiviant from the ore zone into overlying and/or underlying aquifers causing contamination of these aquifers. This is an inherent risk with in situ solution mining in an area of known exploration.

RMEC (1978a) presents copies of geophysical and lithological logs recorded for 13 boreholes in the Reno Creek permit area. Natural gamma and resistance logs were recorded for each hole. The nature of the log response in certain lithologies enabled identification of separate stratigraphic units. Apparently no caliper logs were recorded for these

boreholes. These logs were terminated a short distance below the ore zone.

A.6.4.2 Aquifer Tests

Several aquifer tests were conducted at the Reno Creek mine site. One multiple-well pumping test, three injection tests, and one step drawdown test were conducted in May, 1978, to evaluate aquifer properties, well characteristics, and optimal operating flow rates for injection and recovery of lixiviant. Table A.32 summarizes the aquifer tests conducted at the Reno Creek mine. The results of these tests are presented by In-Situ Consulting, Inc. (1978b). Much of the following discussion is based on information contained in that report.

A multiple-well pumping test was conducted in May, 1978. Well OB-1, which is open throughout the entire thickness of the production sandstone, was pumped at a constant rate of 0.98 L/s (15.6 gal/min) for approximately three hours. An automatic flow regulator was used to maintain a constant pumping discharge which was measured with a turbine flow meter with reported accuracy of +5% (In-Situ Consulting, Inc. 1978b). Flow rates were checked periodically by measuring the time for the discharge stream to fill a 18.9 L (5 gal) bucket. Nine monitor wells completed in the production sandstone, one well completed in the overlying sandstone, and one well completed in the underlying sandstone were used as observation wells. Table A.33 presents completion details of the wells used for the pumping test. Water levels were measured with a steel tape and a water level recorder prior to initiating the pumping test. Pressure transducers and test. Pressure transducers and a steel tape were utilized to measure water level drawdowns in the observation wells. Drawdowns of up to 1.46 m (4.8 ft) were measured in the production sandstone; water levels in the overlying and underlying sandstone were essentially unchanged during the pumping test, indicating that there is probably no connection between aquifers. The pumping tests conducted at the Reno Creek mine were sufficient to allow estimates of the hydraulic properties of the production aquifer.

Table A.34 presents a summary of the aquifer characteristics estimated by In-Situ Consulting, Inc. (1978b) and the authors for test no. 1. The method of Boulton (1963) for unconfined aquifers with delayed yield was used by In-Situ Consulting, Inc. (1978b) because the aquifer behaved as an unconfined water table aquifer although it is overlain by lower hydraulic conductivity strata. Because water levels dropped below the top of the aquifer in most wells, this method of analysis is applicable. An injection test, which would retain the water level above the top of the aquifer, might enable detection of leakage through the upper confining layer. The average early value of specific yield was estimated to be 4.6×10^{-4} which is too small for an unconfined aquifer. Therefore, it is the apparent value before gravity drainage was completed. The duration of the test was not long enough to evaluate the effects of delayed gravity drainage. The actual value of specific yield probably ranges between 0.1 and 0.01. Mean transmissivity was estimated to be about 22 m^2/day (240 ft^2/day).

Table A.32. Summary of aquifer tests conducted at the Reno Creek mine.

Test No.	Pumping/ Injection Well	Production Unit/ Well Field	Aquifer	Total Depth of Tested Well m (ft)	Number of Observation Wells	Units Monitored	Type of Test[a]	Date of Test	Length of Test (min)	Discharge Rate L/s (gal/min)
1	OB-1	Pattern I	Production Sandstone	120 (395)	11	Production, Upper and Lower Sandstones	CD	5/22/78	180	0.99 (15.64)
2	P-1	Pattern I	Production Sandstone	126 (413)	3	Production Sandstone	SD	5/21/78	140	0.21-1.14 (3.4-18.1)
3	I-1	Pattern I	Production Sandstone	126 (414)	0	Production Sandstone	I	5/23/78	32	0.47 (7.5)
4	I-2	Pattern I	Production Sandstone	126 (414)	0	Production Sandstone	I	5/23/78	52	0.18 (2.86)
5	I-3	Pattern I	Production Sandstone	127 (418)	0	Production Sandstone	I	5/23/78	37	0.39 (6.12)

[a]CD = Constant Discharge SD = Step Drawdown I = Injection.

Table A.33. Test well information for the Reno Creek mine.

Well No.	Sand Depth m (ft)	Aquifer	Perforated or Screen Interval m (ft)	Casing Total Depth m (ft)	Piezometric Surface Elevation (6/3/78) m (ft)
P1	87.20-122.26 (286-401)	Production	97.2-103.3 (319-339) 108.2-111.6 (355-366)	125.91 (413)	1503.40 (4931.15)
I1	86.43-122.41 (283.5-401.5)	Production	107.7-112.6 (353.5-369.5) 98.6-101.3 (323.5-332.5)	126.22 (414)	1503.38 (4931.09)
I2	86.43-122.41 (283.5-401.5)	Production	97.7-103.2 (320.5-338.5) 110.2-113.2 (361.5-371.5)	125.91 (413)	1503.42 (4931.22)
I3	86.43-122-41 (283.5-401.5)	Production	97.7-100.4 (320.5-329.5) 101.7-104.1 (333.6-341.5) 108.7-111.1 (356.5-364.5)	127.44 (418)	1503.43 (4931.25)
I4	86.28-122.00 (283-400)	Production	96.6-98.5 (317-323) 100.3-102.7 (329-337) 106.4-108.2 (349-355)	126.22 (414)	1503.40 (4931.16)
OB1	87.04-122.10 (285.5-400.5)	Production	84.7-120.4 (278-395)	120.43 (395)	1503.40 (4931.15)
M1	81.71-118.30 (268-388)	Production	81.1-117.7 (266-386)	117.68 (386)	1503.29 (4930.80)
M2	78.20-118.75 (256.5-389.5)	Production	75.9-123.4 (249-405)	123.48 (405)	1503.44 (4931.28)
M3	84.45-121.34 (277-398)	Production	82.3-124.1 (270-407)	124.09 (407)	1503.57 (4931.71)
M4	83.23-119.82 (273-393)	Production	80.8-116.7 (265-383)	116.77 (383)	1503.60 (4931.82)
USM-1	55.64-58.08 (182.5-190.5)	Upper Sand	53.6-65.5 (176-215)	65.56 (215)	1546.38 (5027.11)
LSM-1	133.54-139.94 (438-459)	Upper Sand	132.6-138.7 (435-455)	140.55 (461)	1499.61 (4918.72)

Source: In-Sito Consulting, I nc.,1978b.

Table A.34. Summary of aquifer characteristics estimated by In-Situ Consulting and by the authors from pumping/injection test data for the Reno Creek mine.

Test No.	Pumping/ Injection Well	Aquifer	Analysis Conducted By[a]	Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Method of Analysis[b]	Type of Test[c]
1	OB-1	Production Sandstone	ISC	2.5×10^{-4} to 2.2×10^{-3} (0.7 to 6.2)	22 (236)	4.6×10^{-4}	B	CD
			Authors	2.5×10^{-4} to 5.7×10^{-4} (0.7 to 1.6)	13 (143)	8.7×10^{-4}	B	
2	P-1	Production Sandstone	ISC	(Data Not Available)				SD
3	I-1	Production Sandstone	ISC	(Data Not Available)				I
4	I-2	Production Sandstone	ISC	(Data Not Available)				I
5	I-3	Production Sandstone	ISC	(Data Not Available)				I

[a] ISC - In Situ Consulting

[b] B = Boulton (1963)

[c] CD = Constant Discharge; SD = Step Drawdown; I = Injection

Source: In-Situ Consulting, Inc., 1978b.

Analysis of the pumping test data by the authors using the Boulton (1963) method yielded similar ranges for aquifer properties. The average early value of specific yield was estimated to be 8.7×10^{-4}. Mean transmissivity was estimated to be 13 m^2/day (140 ft^2/day). However, this analysis does not include wells M1, M2, M3, and M4 because sufficient drawdown data are not available to allow reasonably certain matches to the Boulton type curves. The authors attempted to utilize the Neuman (1975) method of analysis for unconfined aquifers considering delayed gravity response to estimate aquifer hydraulic properties. However, the Neuman method could not be used because either the observation wells did not fully penetrate the aquifer although sufficient drawdown data may have existed, or sufficient drawdown data was not collected (as with wells M1, M2, M3, and M4).

Very little information is available for review concerning the injection tests and the step drawdown test. In-Situ Consulting, Inc. (1978b) indicates that the data from these tests were used to calculate well efficiencies. Well P1 was pumped for about 2.5 hours at controlled rates ranging from about 0.22 to 1.14 L/s (3.5 to 18.1 gal/min) in the step drawdown test. Water levels were measured in wells I-2, I-4 and OB-1 (Fig. A.50). The step drawdown test produced drawdowns up to 1.43 m (4.7 ft) in the observation wells and about 12.8 m (42 ft) in the pumped well. Wells I-1, I-2, and I-3 were tested by injecting water into the production sandstone at various rates. These tests lasted approximately 30 minutes to one hour with average injection rates ranging from 0.18 to 0.47 L/s (2.9 to 7.5 gal/min).

In-Situ Consulting, Inc. (1978b) concluded that the upper and lower mudstones were good confining layers because water level declines were not observed in the observation wells in the aquifers above and below the ore zone. However, the aquifer test data from these observation wells are insufficient to support this conclusion, because antecedent water level trends and barometric fluctuations were not recorded, and the pumping tests were too short.

A.6.5 Hydrogeology of the Mine Site

Uranium mineralization at the Reno Creek site occurs in fluvial sandstones of the Wasatch Formation. The Wasatch Formation consists of approximately 490 m (1600 ft) of interbedded sandstones, siltstones, claystones, carbonaceous shales and thin coal seams (RMEC, 1978b). The ore-bearing sandstone units range from about 150 m (500 ft) to several kilometers wide by 1.6 to 12 km (1 to 8 miles) long and 3 to 34 m (10 to 110 ft) thick. Thick, coarser-grained sands were deposited in a central belt probably corresponding to an ancient stream channel; finer-grained materials were deposited along the flanks of the belt as less permeable floodplain deposits. Uranium deposits typically are of the roll-front type, with the nose pointing down-dip; the tails occur along zones of finer-grained material in the sandstone. The uranium mineralization occurs along a red sandstone/gray sandstone contact.

Fig. A.52 is a geologic cross section developed from geophysical and driller's logs recorded for production, injection and monitor wells in Pattern I. This figure shows that the ore-bearing sandstone occurs at a depth of about 85 m to 120 m (280 to 400 ft). Approximately 76 m (250 ft) of mudstone overlies the host sand; however, coal seams about 3 m (10 ft) thick and sandstone stringers are interbedded within the mudstone. The cross section shows the presence of mudstone below the host sand; the thickness of the mudstone stratum is uncertain but appears to be at least 4.5 m (15 ft). It is not known whether the boreholes were only drilled slightly deeper than the host sand or if the boreholes were logged to that depth only. The lower mudstone stratum may not have been penetrated fully. Fig. A.53 is a generalized stratigraphic column of the mine site area.

Ground water occurs under confined and unconfined conditions at the Reno Creek mine site. Shallow sandstones and siltstones which overlie a mudstone stratum form an unconfined aquifer. The production sandstone aquifer which averages 36 m (118 ft) thick underlies this mudstone stratum. Ground water in the production sandstone is under slight confining pressures. An unknown thickness of mudstone separates the production sandstone aquifer from an underlying sandstone aquifer. Hydraulic properties of the water table aquifer and the mudstone units have not been evaluated by aquifer testing at the Reno Creek mine site to date.

Investigations conducted near the site by the University of Wyoming suggest ground water in the water table aquifer generally flows in a northerly direction (In-Situ Consulting, Inc., 1978b). Static water levels in the ore zone aquifer measured prior to aquifer testing indicated local ground-water flow in a direction N36°E. Piezometric surface data contained in Table A.33 indicate a downward potential gradient exists at the site.

A.6.6 Monitor Well System and Well Construction

Well field monitoring in Pattern I consists of six monitor wells. Wells M-1, M-2, M-3, and M-4 are completed in the ore sandstone; wells USM-1 and LSM-1 are completed in the overlying sandstone and underlying sandstone, respectively (Fig. A.50). Well field monitoring in Pattern II consists of six monitor wells also. Wells M-16, M-17, M-18, and M-19 are completed in the ore sandstone; wells USM-20 and LSM-21 are completed in the overlying and underlying sandstone, respectively (Fig. A.51).

Although monitor wells USM-1 and USM-20 are completed in an overlying sandstone to detect vertical excursions, RMEC (1980c) reported difficulties with obtaining representative water samples from the wells. Similar difficulties were experienced with the lower sandstone monitor wells LSM-1 and LSM-21. Well USM-20 was completed in a silty sandstone sequence which occupies the same stratigraphic horizon as the overlying sandstone monitored in Pattern I. A lateral facies change from a channel sandstone to a floodplain silty sandstone evidently occurs between Pattern I and Pattern II in that horizon. The low hydraulic conductivity of this unit in Pattern II limited production of water from well USM-20 to less than 0.06 L/s (1 gpm). Analyses of samples bailed from the well reportedly indicated contamination from drilling fluids. The well was never developed fully;

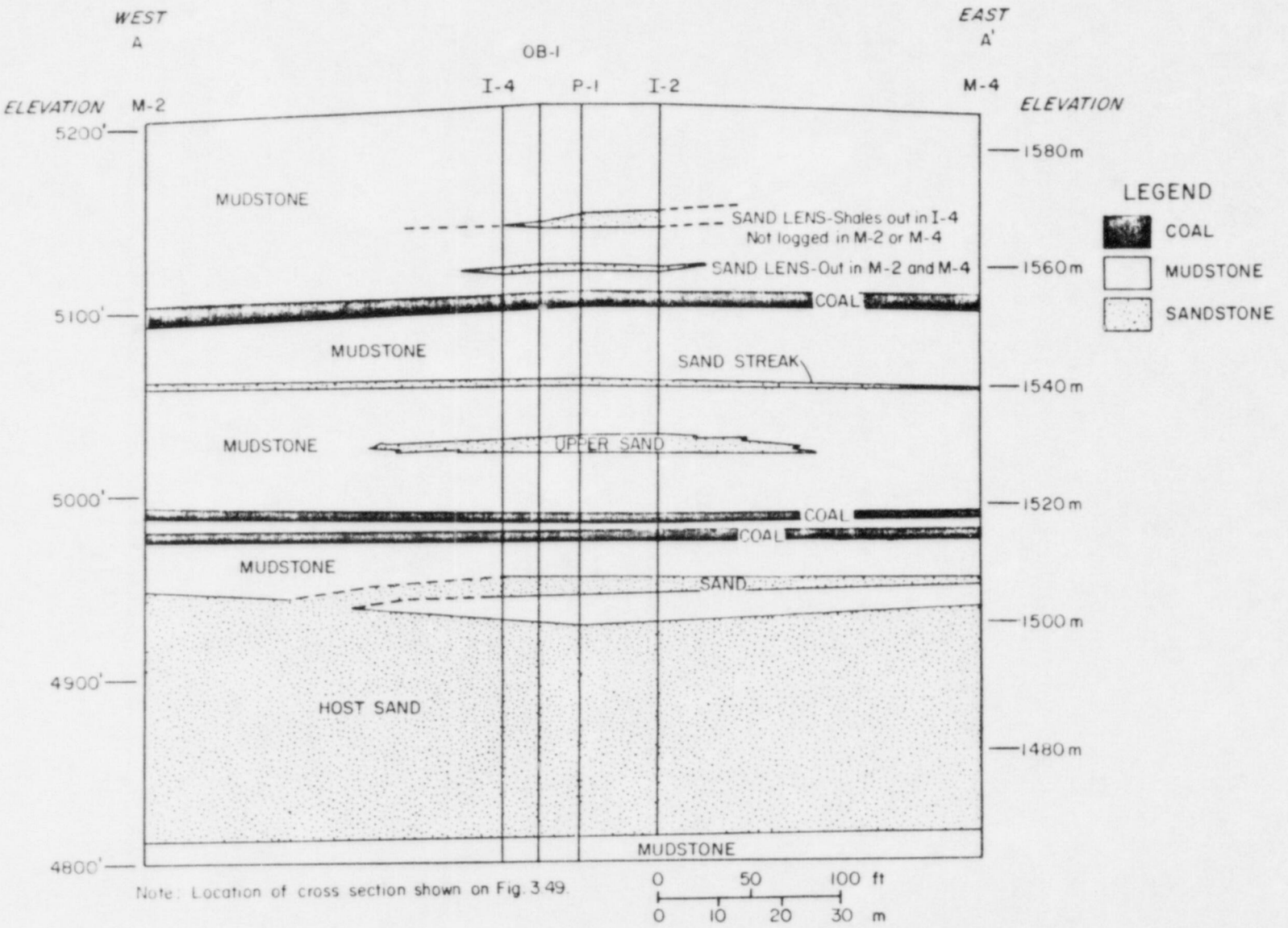


Fig. A.52. Geologic cross section through Pattern I at the Reno Creek mine. *Source:* RMEC, 1978b.

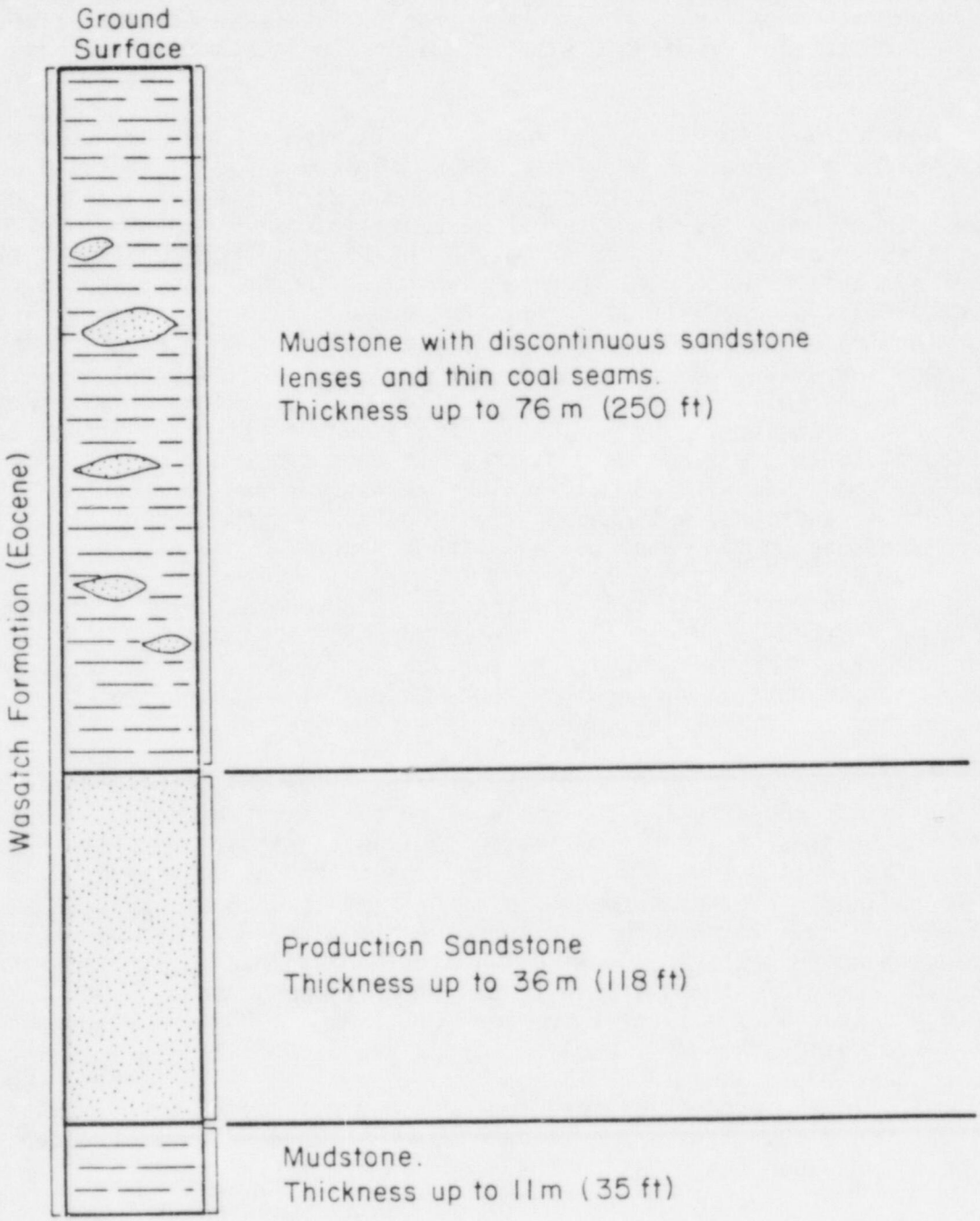


Fig. A.53. Generalized stratigraphic column for the Reno Creek project area.

therefore, routine sampling of the well is not possible. Comparison of water quality data for upper and lower monitor wells in Pattern II with those for Pattern I indicates similar evidence of contamination from drilling fluids.

Production, injection, and monitor wells were constructed at the Reno Creek mine in the following manner (RMEC, 1978a,b). Boreholes were drilled to a depth below the production sandstone and cased with fiberglass or PVC pipe. Injection wells in Pattern I were drilled as 16.5 cm (6.5 in) holes. The holes were cased with 7.8 cm (3 in) inside diameter PVC plastic pipe, and the annuli were grouted from the bottom of the holes up through the production sandstone with acid-resistant cement. Regular cement was used from the top of the production zone to the land surface. After the cement hardened the casing was perforated in the mineralized horizons. The same procedure was followed for the construction of production wells in Pattern I, except larger boreholes [17.8 cm (7 in)] were drilled and larger casing [11 cm (4.36 in)] was used. Monitor wells were constructed similarly but each was completed with 10 cm (4 in) PVC plastic screen over the entire production sandstone, with the annulus grouted from the top of the production sandstone up to ground surface with cement (Fig. A.54).

Wells in Pattern II were constructed in essentially the same manner as those in Pattern I. Monitor wells were grouted with cement which was mixed continuously as it was pumped into the annulus of each well. The production wells in Pattern II were grouted with cement which was batch mixed and then pumped into the annulus of each well from the bottom up.

Pattern II wells were completed originally by selectively perforating the casing or underreaming the wells after the cement hardened. However, baseline samples of water from the production sandstone indicated possible well contamination. RMEC modified the production and injection wells by underreaming to create an open-hole completion of 26.6 cm (10-1/2 in) diameter. These wells were then pumped for an extended period of time during which pH measurements were taken at hourly intervals. The average pH dropped from approximately 11.5 to 8.4 after pumping about 38,000 to 57,000 L (10,000 to 15,000 gal) of water per well (RMEC, 1980d). It was postulated eventually that dry, unmixed cement had dissolved slowly in the wells prior to recompletion and/or that water naturally high in sulfate (800 to 900 mg/L) in the production sandstone may have dissolved the cement. These factors would have caused the elevated pH levels and elevated concentrations of pH dependent constituents.

Production and injection wells in Pattern II were tested following their construction to assure integrity of the casings. Each well was pressure tested by installing an inflatable packer within the casing above the production zone. Water and/or air was injected into the well, through a control valve at the top of the well, following inflation and seating of the packer. A pressure of about 5.4×10^5 N/m^2 (75 psi) was created within the well with the control valve sealed, and pressure level gauges were observed for 10 minutes. If a significant drop in pressure occurred, the well was inspected further to ascertain the cause of the apparent leak.

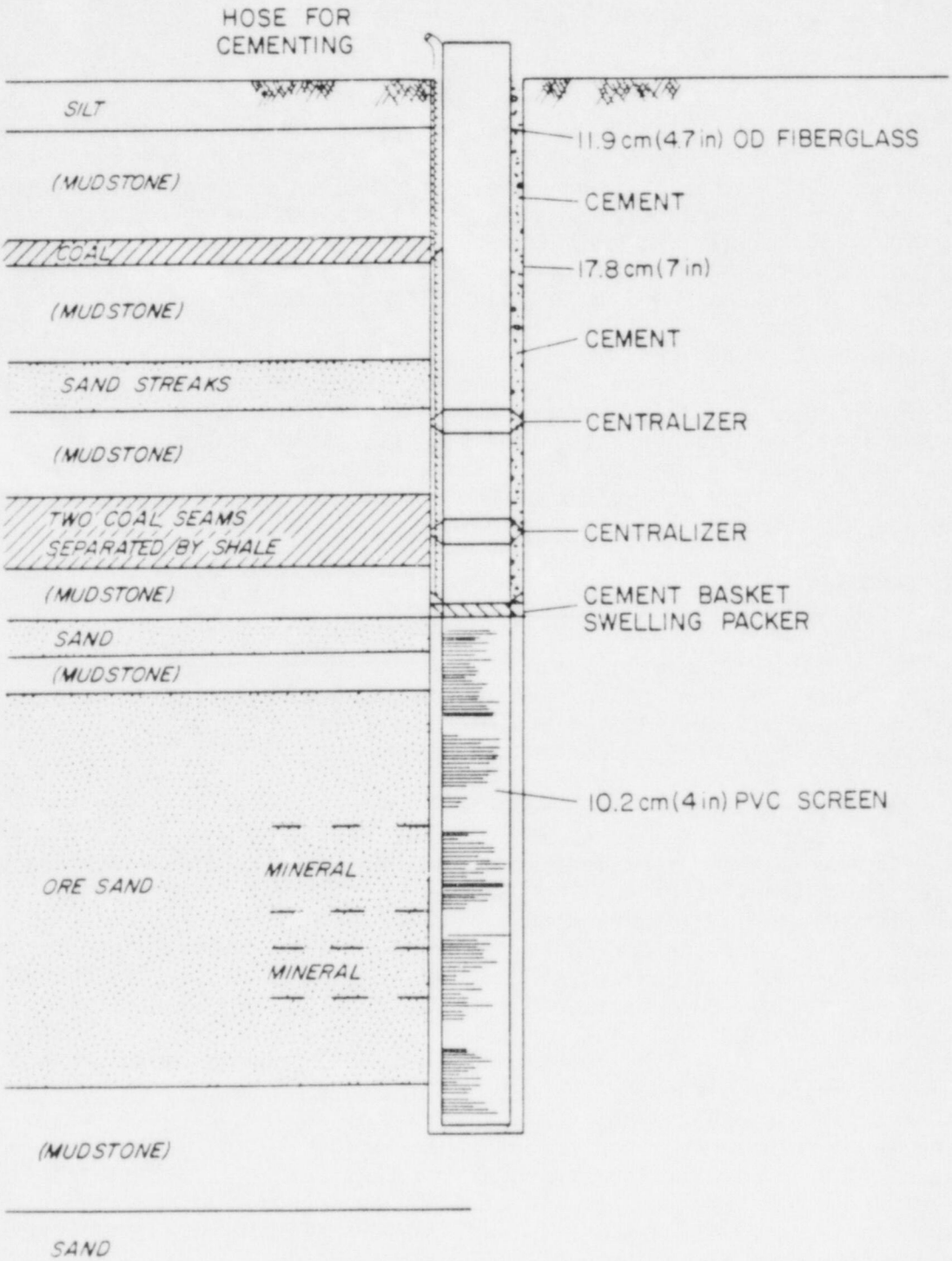


Fig. A.54. Typical construction of monitor wells at the Reno Creek mine. *Source:* RMEC, 1978b.

All production and injection wells were tested in this manner until each one passed without significant leakage.

A.6.7 Analysis of Excursions

Both in situ mining tests conducted at the Reno Creek mine site were completed with no observed excursions. Excursion indicators for the acid lixiviant test (Pattern I) were chosen and UCL's for these constituents (pH, sulfate and conductivity) were established following baseline water quality evaluation. Table A.35 presents the ranges of baseline sample concentrations for major and minor constituents in Pattern I. Baseline sampling of the upper and lower sand units detected contamination by drilling fluids. Evaluation of actual water quality in these units was hampered further by very low yields in monitor wells LSM-1 and USM-1. UCL's were set for sulfate and conductivity and lower control limits were set for pH (Table A.36); however, there is some uncertainty concerning the method used to calculate these limits (RMEC, 1981c). Comparison of the reported baseline ranges and UCL's indicates that the UCL's for sulfate were established by adding 10% of the highest measured baseline value to the baseline sulfate value for each well.

Monitor wells were sampled for excursion indicators plus uranium every two weeks in Pattern I. Samples also were taken on a quarterly basis from each monitor well and analyzed for an expanded list of constituents. If any two excursion indicator constituents had been detected in excess of the UCL's (below the lower control limit for pH) and verification sampling within 48 hours confirmed the excess, corrective actions would have been required to mitigate the apparent excursion.

No legal excursions were reported during Pattern I leaching activities. However, it is questionable whether the UCL's were sensitive enough to detect an excursion, given the uncertainty over baseline sampling procedures and methods of establishing UCL's. Furthermore, the problems experienced with ascertaining premining water quality in the overlying and underlying silty sandstone units, and difficulties in obtaining samples from wells LSM-1 and USM-1 during mining make it questionable whether vertical excursions would have been detected had they occurred. One instance was reported where UCL's for conductivity and sulfate were exceeded. A sample taken from well M-1 on November 12, 1980, contained an excessive level of sulfate; however, the pattern had been in restoration for approximately one year. Baseline values reported by RMEC (1981c) for well M-1 were lower than those reported for other monitor wells. RMEC concluded there had been no excursion based on the fact that concentrations of uranium and calcium were not elevated as had occurred at their Nine Mile Lake mine site (Section A.3.7). Very few details are available concerning this occurrence.

Baseline sampling of monitor and production wells in Pattern II was conducted during the first half of 1980 (Table A.37). RMEC interpreted the data to indicate that there was a problem with contamination of the wells by drilling fluids and/or well construction materials. Measured pH values ranged from 9.3 to 11.2, which were higher than expected. Concentrations

Table A.35. Baseline water quality[a] for Pattern I at the Reno Creek mine.

Parameter	Well							
	M-1	M-2	M-3	M-4	P-1	OB-1	USM-1	LSM-1
Alkalinity	50-225	30-162	15-75	ND-90	ND-90	29-121	---	---
Aluminum	.02-.15	ND-.49	ND-.66	ND-.91	ND--	ND-.21	ND-1.45	.07-1.03
Ammonia	ND-.27	ND-.19	ND-.74	ND-.64	ND-.5	ND-.19	ND-.27	ND-16.8
Arsenic	ND-.03	ND-.03	ND-.02	ND-.01	ND-.02	ND-.01	ND-.03	ND-.01
Bicarbonate	113-190	56-161	0-65	15-120	15-98	65-122	129-305	0-0
Boron	ND-.26	ND-1.49	ND-.36	ND-1.24	ND-2.6	ND-.78	1.14-1.25	.04-.22
Calcium	83-182	88-119	72-113	96-116	80-120	99-130	10-31	114-203
Carbonate (as CO_3)	0-48	0-32	0-25	0-65	0-24	0-16	12-194	97-281
Chloride	12-30	6-21	10-34	8-24	8-17	12-62	17-38	13-66
Fluoride	ND-.34	ND-.57	ND-.19	ND-.25	ND-.40	ND-.33	.42-1.16	.29-.65
Hardness (as $CaCO_3$)	281-728	320-348	260-340	338-400	279-380	356-380	---	---
Iron	ND-<1.0	ND-<1.0	ND-<1.0	ND-3	ND-3	ND-.22	ND-1.0	ND-1.0
Magnesium	13-51	11-28	10-29	19-29	9-40	15-44	2.7-19	1.0-12.0
Manganese	.02-.22	.02-.09	ND-.01	ND-.04	ND-.01	.04-.09	ND-.04	ND-.02
Nitrate	ND-.24	ND-.26	ND-7	ND-.8	ND-.5	ND-.22	ND-.5	ND-.35
Potassium	7-10	7-8	10-14	8-9	10-25	8-9	5-6.1	16-29
Selenium	ND-.01	ND-.02	ND-.02	ND-.02	ND-.05	ND-.02	ND-.01	ND-.04
Silica	ND-8.7	ND-8.6	ND-5.5	ND-5.3	ND-7.2	ND-7.9	ND-.9	ND-1.12
Sodium	145-280	215-290	220-272	200-322	210-323	215-310	87-110	203-245
Sulfate	486-750	640-860	625-907	670-953	700-900	700-1006	11-76	25-99
TDS	1006-1443	1103-1392	1063-1597	970-1566	1124-1492	1256-1496	283-404	964-1377
Uranium (U_3O_8)	.102-.55	.007-.9	.25-1.1	.27-2	.20-1.05	.032-.9	.001-.014	.001-.003
Vanadium	ND-<1.0	ND-<1.0	ND-<1.0	ND-8	.18-8	ND-<1.0	ND-.07	ND-.10
pH (units)	7.61-8.35	6.4-8.4	8.1-10.82	6.93-9.42	7.7-11.2	7.8-8.8	7.9-8.7	11.7-12.2
Conductivity (µmhos/cm)	1220-1800	1250-1775	1300-2000	1220-1900	1400-2000	1350-2000	300-500	1840-3050

[a]Values are mg/L except as noted; ND = Non Detectable; < indicates data below detectable limits.

Source: RMEC, 1978a and 1981a.

Table A.36. Excursion control limits for monitor wells at the Reno Creek mine.

Wells	Constituent					
	pH	Conductivity µmhos/cm	Sulfate mg/L	Chloride mg/L	Bicarbonate mg/L	Uranium mg/L
Pattern I						
M-1	6.1	1788	873	NS	NS	NS
M-2	6	1959	971	NS	NS	NS
M-3	7.5	2200	988	NS	NS	NS
M-4	6.5	2104	1064	NS	NS	NS
USM-1	6.3	645	86	NS	NS	NS
LSM-1	9.9	3496	124	NS	NS	NS
Pattern II						
M-16	9.7	2479	NS	14.3	176	0.384
M-17	9.9	2331	NS	14.7	157	0.209
M-18	9.9	2219	NS	16.3	139	0.108
M-19	9.7	2443	NS	15.2	130	0.141
USM-20	13.3	2862	NS	349	ND	0.073
LSM-21	13.5	4515	NS	74	1	0.09

NS = Upper control limits were not set for these parameters for these wells.

ND = Bicarbonate not detected in baseline samples; no control limits established.

Source: RMEC, 1981a and 1981b.

Table A.37. Pattern II baseline water quality[a] for monitor wells[b] at the Reno Creek mine.

Parameter	P-11		M-16		M-17		M-18		M-19		USM-20	LSM-21
	1	2	1	2	1	2	1	2	1	2		
Bicarbonate	96	108	47	133	61	123	11	109	55	101	0	0
Carbonate	2	4.7	2	8.2	12	10.8	32	9.7	12	5.4	328-350	273-318
Alkalinity ($CaCO_3$)	76	89	45	109	76	101	89	89	45	83	547-583	455-530
Calcium	104	136	106	135	104	129	102	120	115	134	78-115	16-177
Chloride	12	16.7	8.4	9.8	8.4	9	8.7	12	8.5	10	108-217	32-287
Magnesium	13	24	14	26	23	25	12	23	20	26	1.0-2.2	1.2-7.6
Potassium	10.3	7.4	7.7	8.1	17.1	8.2	44	8.1	16	8.1	31.7-54.1	16.8-34.0
Sodium	283	303	288	309	273	305	287	307	297	312	281-378	226-326
Sulfate	899	917	933	921	923	890	905	869	947	975	329-450	53-578
TDS	1397	1440	1413	1524	1398	1460	1384	1392	1461	1506	960-1490	360-1720
Uranium (U_3O_8)	0.086	0.025	0.007	0.105	0.033	0.080	0.022	0.048	0.026	0.056	0.021-0.078	0.001-0.075
Vanadium	0.06	0.20	0.06	0.04	0.06	0.06	0.06	0.04	0.07	0.04	0.06-0.21	0.05-0.13
pH (units)	9.27	8.55	9.54	8.43	9.46	8.57	11.19	8.75	9.5	8.44	11.55-11.85	11.45-12.0
Conductivity (µmhos/cm)	1907	1981	1957	2114	1928	1991	2060	1949	1968	2063	1438-4286	1437-3250

[a]Values are mg/L except as noted.

[b]Sample 1 prior to well recompletion; sample 2 following well recompletion

Source: RMEC, 1980e and 1981a.

of pH dependent constituents also were found to be higher than expected. Subsequently, the wells were underreamed and recompleted. Additional baseline sampling was conducted. Table A.37 presents a comparison of water quality data collected before and after the wells were recompleted; water quality generally was poorer after recompletion.

Excursion indicators for the carbonate lixiviant test in Pattern II were pH, chloride, bicarbonate, uranium and conductivity (Table A.36). UCL's for excursion indicators were set for each well according to the formula:

$$UCL = 1.1 (x + 2s)$$

where x = average baseline concentration for each well and s = standard deviation of the baseline samples for each well. Lower control limits (LCL) for pH were set according to a similar formula:

$$LCL = .9 (x + 2s)$$

Difficulties with obtaining representative baseline water quality for the overlying and underlying aquifers were encountered in Pattern II. Both the upper and lower monitor wells (USM-20 and LSM-21) were completed in sandstones which yielded limited amounts of water. Attempts to remove residual drilling fluids from well completion materials by sustained pumping of the wells was not successful (RMEC, 1980d). Samples were obtained from the upper sandstone monitor well by bailing; the lower sandstone monitor well produced only about 0.3 to 0.6 L/s (5 to 10 gal/min) for about three to ten minutes before the wellbore was dewatered. Values of pH for wells USM-20 and LSM-21 were unusually high (11.5 and 11.8, respectively).

Monitor wells in Pattern II were sampled for excursion indicators every two weeks. In addition, once each month samples were analyzed for ^{226}Ra, ^{230}Th, arsenic and selenium. Samples were analyzed quarterly for an expanded list of constituents. No excursions (any two excursion indicator constituents exceeding UCL's) were detected during Pattern II leaching. However, the conditions mentioned previously regarding the upper and lower sandstone monitor wells may have prevented detection of vertical excursions.

A.6.8 Restoration

Injection of acid lixiviant in Pattern I was discontinued on November 12, 1979. The Pattern I leaching test failed due to the high buffering capacity and acid consuming properties of the ore host rock. Problems encountered during leaching, such as reduced well field circulation due to scaling in wells and precipitation of gypsum ($CaSO_4$) within the aquifer, also complicated restoration efforts. Flow to the ion exchange columns was halted when levels of uranium and calcium in the produced fluid stabilized

at about 4 mg/L and 300 to 400 mg/L, respectively. Initial restoration involved injection of barren solution from the ion exchange columns into the ore sandstone. Approximately 0.7 pore volumes [3,664,000 L (968,000 gal)] of fluid were produced, and about 1,472,000 L (389,000 gal) of treated fluid were injected during this first phase of restoration. A ground-water sweep was conducted beginning in December, 1979. Well P-1 was pumped initially at about 1.14 L/s (18 gal/min), Within one month, an additional well, OB-1, was pumped at about 0.63 L/s (10 gal/min). The flow rate of P-1 was reduced to about 0.95 L/s (15 gal/min) for a combined production rate (wells P-1 and OB-1) of about 1.57 L/s (25 gal/min). The ground-water sweep was discontinued in March, 1980, after approximately 2.1 pore volumes [10,976,000 L (2,900,000 gal)] of fluid were produced. Lack of evaporation pond storage capacity was cited as the reason for discontinuing the sweep; however, no details were reported regarding the storage capacity.

Ground-water sweeping efforts failed to reduce significantly the concentration of hydrogen ions (initially present in the sulfuric acid lixiviant) which had replaced calcium ions in clay lattice structures of the ore sandstone (RMEC, 1981a). About 90,900 Kg (200,000 lb) of acid were used in the leaching test, and approximately 10,640 Kg (23,400 lb) of calcium were removed from the production sandstone by the leaching process (RMEC, 1981a). Ground-water pH rose to about 5.0 as a result of the restoration efforts described above. Free acid levels dropped to 331 mg/L, and concentrations of pH dependent constituents, such as sulfate, remained at levels in excess of premining concentrations.

An in situ water treatment circuit was installed in March, 1980, to reduce the hydrogen ion concentration further. Potassium salts were injected into the ore sandstone to increase ground-water pH. In this phase of restoration, approximately 1.8 pore volumes [9,084,000 L (2,400,000 gal)] of fluid were produced; of this, approximately 6,434,500 L (1,700,000 gal) of treated fluid were reinjected into the ore sandstone. The treatment circuit was modified later by adding $Ca(OH)_2$ to the injection-stream fluid to raise the pH. This enhanced restoration effort continued until March, 1981. Continual well plugging by gypsum scaling hampered the effectiveness of this technique; in addition, ground-water pH was not increased significantly above about 5.0.

Table A.38 presents water quality data collected during leaching and initial restoration activities. Low pH water in the ore zone aquifer contained levels of sulfate in excess of baseline concentrations; ^{226}Ra also was found at levels exceeding baseline (NRC, 1982d). Additional restoration alternatives to increase pH and reduce sulfate and ^{226}Ra concentrations were investigated. RMEC suggested that injection of a high TDS solution would result in the displacement of hydrogen ions from clay lattice structures in the production sandstone; they suggested also that hydrogen ions could be removed in the restoration process, thereby increasing ground-water pH which would lead to reductions in sulfate and ^{226}Ra concentrations. However, regulatory agencies did not allow this procedure because they were unsure of the technique and suspected it might have adverse effects on the aquifer. The Wyoming Department of Environmental

Table A.38. Pattern I ground-water quality[a] during leaching and restoration at the Reno Creek mine.

Activity/Date	Gallons Removed	Calcium	Sulfate	Uranium	Vanadium	Iron	pH
Leaching							
February 79	607,060	156.2	981	<0.1	<0.1	<0.1	7.6
March 79	1,152,396	434.2	1,772	<0.1	<0.1	1.0	7.1
April 79	1,163,936	665.6	2,887	<0.1	<0.1	30.1	6.3
May 79	1,487,341	570.4	3,226	3.2	2.5	98.2	5.1
June 79	588,769	500.8	2,972	10.5	3.3	138.0	4.2
July 79	61,744	739.4	3,928	22.0	6.5	225.0	3.8
August 79	1,516,491	482.3	2,926	18.4	4.9	181.9	3.5
September 79	1,269,266	422.7	2,951	13.8	7.9	191.2	3.0
October 79	1,583,040	395.0	3,021	12.1	9.0	242.4	2.7
November 79	932,609	380.7	3,198	9.2	6.1	210.6	2.9
Restoration							
December 79	986,003	377.1	2,456	7.4	1.8	130.8	3.0
January 80	1,070,763	329.1	2,281	7.6	1.5	130.4	3.3
February 80	1,025,432	259.2	1,998	5.5	0.7	96.5	3.1
March 80	1,161,795	213.0	1,667	4.0	2.0	98.0	3.9
April 80	889,533	204.0	1,562	3.0	1.0	81.0	4.1
May 80	730,750	229.4	1,675	3.1	0.3	65.1	4.3
June 80	455,742	229.4	1,675	3.2	0.1	52.2	4.3
July 80	451,650	242.3	1,758	3.6	0.5	78.5	4.4
August 80	509,220	235.4	1,712	2.7	1.0	73.0	4.4
September 80	462,440	244.7	1,653	2.4	0.8	59.6	4.3
October 80	479,510	219.8	1,501	2.8	0.8	53.9	4.3
November 80	281,458	225.9	1,447	1.2	0.7	50.0	4.4
December 80	657,160	193.1	1,394	1.1	0.3	44.5	4.7
January 81	660,915	224.5	1,378	1.2	0.3	49.7	4.9
February 81	203,385	217.0	1,482	1.4	0.4	38.5	4.9
June 81	0	230.0	1,385	1.1	0.2	29.1	4.8

[a]Values are monthly averages for wells P-1 and P-2 in mg/L except pH.

Source: RMEC, 1981b.

Quality in June, 1981, authorized a period of stabilization during which the production aquifer was monitored to observe the effects of natural geochemical processes on water quality. Pattern I was in a post-restoration stabilization mode as of June, 1984.

Very little post-restoration water quality improvement has occurred at the Reno Creek mine site. Ground-water quality data for production wells for the first quarter of 1983 indicate that since the end of active restoration efforts (March, 1981): 1) pH levels in the aquifers have not changed significantly, 2) calcium and sulfate concentrations have not changed significantly (270 and 1500 mg/L, respectively), and 3) uranium concentrations have decreased slightly to less than 1.0 mg/L (RMEC, 1983). Table A.39 presents the latest reported water quality data for Pattern I wells. TDS levels which were about 1400 mg/L before mining was conducted have increased to about 2700 mg/L.

Restoration of Pattern II began on December 22, 1980, when chemical refortification of the produced fluids was discontinued. In the initial phase of restoration, production fluid was routed to the processing plant for uranium removal; the water was reinjected into the ore zone aquifer. Two months later, an ion exchange (IX) process, whereby divalent cations were stripped from produced fluids using a weak acid resin, was utilized for restoration. The ion exchange process continued from February 23 to March 13, 1981; approximately two pore volumes [1,961,000 L (518,000 gal)] of fluid were circulated through the aquifer during this phase of restoration. The ion exchange restoration process reduced concentrations of all constituents except uranium and vanadium to within baseline ranges. Concentrations of uranium and vanadium were reduced to about 2 mg/L and about 1 mg/L, respectively (RMEC, 1981b).

A ground-water sweep was conducted from March 13 to April 16, 1981, in an attempt to reduce ^{226}Ra, uranium, and vanadium concentrations further. The well field was pumped at about 1.6 L/s (25 gal/min), and approximately 4.5 pore volumes [4,411,000 L (1,166,000 gal)] of ground water were removed in this final phase of restoration. Little improvement in water quality resulted from the ground-water sweep which probably was the principal reason for cessation. Table A.40 presents water quality data for the various restoration phases.

RMEC (1981d) reported that ground-water quality data suggested that "fresh" ground water drawn into the aquifer by the ground-water sweep began to create an oxidizing environment and might have caused remobilization of some metals (i.e., uranium and vanadium). A stabilization monitoring program to observe water quality changes which might result from natural geochemical processes was initiated after the ground-water sweep. During the stabilization period, samples were taken monthly and analyzed for key constituents. Table A.41 which presents uranium concentrations in production and injection wells after 22 months of post-restoration stabilization shows that concentrations of uranium increased at five of the six wells. The average concentration in Pattern II was 1.05 mg/L at the beginning of the stabilization period; 22 months later (February, 1983) the production pattern average was 3.69 mg/L.

Table A.39. Water quality data[a] for Pattern I for first quarter, 1983, at Reno Creek mine.

	Sample Date	pH	TDS	Calcium	Sulfate	Uranium
Monitor Wells						
M-1	CL[b]	6.5	2858	286	1606	1.102
	1/12/83	8.0	1265	86	829	0.024
	2/17/83	7.7[c]	1258	89	811	0.006
	3/01/83	8.0[c]	1245	90	777	0.016
M-2	CL	6.5	2744	278	1644	1.027
	1/17/83	8.3	1109	71	613	0.049
	2/17/83	7.9[c]	1120	76	715	0.019
	3/02/83	8.0[c]	1272	83	779	0.018
M-3	CL	6.5	3104	286	1657	1.750
	1/17/83	9.6[c]	1215	91	693	0.027
	2/17/83	10.7[c]	1190	76	786	0.024
	3/02/83	9.4[c]	1310	99	836	0.171
M-4	CL	6.5	3096	304	1783	1.550
	1/13/83	11.7[c]	1262	132	608	0.001
	2/17/83	11.8[c]	1272	136	748	0.049
	3/01/83	11.9[c]	1242	137	665	0.029
LSM-1	CL	6.5	994	207	137	1.022
	1/12/83	11.3[c]	350	42	50	0.024
	2/16/83	11.6[c]	375	45	30	0.124
	3/01/83	11.8[c]	380	68	157	0.157
USM-1	CL	6.5	987	32	137	1.029
	1/14/83	7.6[c]	365	26	2.0	0.001
	2/17/83	7.6[c]	415	28	10.0	0.003
	3/01/83	7.7[c]	375	28	6.0	0.010
Production Wells:						
P-1	CL	3.8	2771	267	1673	5.000
	1/13/83	5.5	2240	240	1346	0.921
	2/16/83	5.6	2322	221	1505	0.259
	3/03/83	5.5	2452	247	1513	0.738
P-2	CL	3.8	2771	267	1673	5.000
	1/13/83	5.1	2500	264	1504	0.822
	2/16/83	5.3	2750	275	1776	0.701
	3/03/83	5.1	2649	278	1588	0.542

[a]Values are mg/L except for pH.
[b]CL--Stabilization Control Limits
[c]Bailed Sample
Source: RMEC, 1983.

Table A.40. Water quality data[a] for the Pattern II production zone at the Reno Creek mine.

Parameter	Baseline[b] Range	Phase I (Leaching)	Phase II (Post Leach)	Phase III (Post IX)	Phase IV (Post Sweep)
Bicarbonate	89-178	1800	1670	160	125
Calcium	108-153	330	207	69	87
Chloride	7.0-18.8	240	113	19	15
Sodium	287-360	900	770	305	322
Iron	0.03-0.61	8.0	0.6	0.16	0.39
Uranium (U_3O_8)	0.012-0.287	65	16	1.64	1.37
Vanadium	0.05-0.34	6	3	1.05	0.45
Radium-226 (pCi/L)	106-768	-	311	238	222
pH (units)	8.2-8.9	7.2	7.4	7.7	7.7
Conductivity (μmhos/cm)	1890-2234	3500	3400	2000	1995

[a]Values are mg/L except as noted.

[b]Baseline range is for all Pattern II wells following removal of outlying data points.

Source: RMEC, 1981b.

Table A.41. Comparison of uranium concentrations (mg/L) in Pattern II wells at the Reno Creek mine.

Well	Uranium 4/16/81	Uranium 2/8/83
P-10	1.6	4.50
P-11	1.7	1.89
I-12	4.5	3.42
I-13	1.2	4.87
I-14	1.9	5.0
I-15	0.82	2.48
Average	1.05	3.69

Source: RMEC, 1983.

The Wyoming Department of Environmental Quality and the NRC have decided that adequate restoration of Pattern II has been achieved. Although uranium concentrations exceed baseline averages, the pattern average after 22 months of post-restoration stabilization was below the 5 mg/L Wyoming drinking water standard. Therefore, no further restoration effort is required for Pattern II. Water quality in Pattern I is still being investigated and no final decision has been made regarding the adequacy of restoration there.

The phenomenon of increasing uranium concentrations following restoration has been observed at other facilities (NRC, 1983a). It is questionable whether this is a result of oxygenated water being drawn into the mined zone remobilizing the uranium, the slow leakage of leaching solution from storage in the confining strata, or simply the result of the geochemical system proceeding toward a new equilibrium. More accurate characterization of the hydrostratigraphy might enable one to ascertain the cause of this phenomenon. This illustrates the need for thorough aquifer testing, and sophisticated data analysis techniques to provide information pertinent to aquifer restoration as well as operational excursion control.

A.7 CLEVELAND CLIFFS IRON COMPANY - COLLINS DRAW PROJECT

A.7.1 Description of the Site

The Cleveland Cliffs Iron Company (CCIC) has operated the Collins Draw mine in Campbell County, Wyoming. The mine site is located about 96 km (60 mi) northeast of Casper, Wyoming in the Powder River Basin (Fig. A.55). Topography of the mine site is characterized by a rolling plain dissected by stream drainages which originate in the group of small mesas known as the Pumpkin Buttes. Elevations range from about 1460 to 1650 m (4800 to 5400 ft) at the site. Intermittent streams in the vicinity drain to the Powder and Belle Fourche Rivers. The Powder River is located about 19 km (12 mi) west of the mine site. Annual precipitation usually is between 30.5 and 35.5 cm (12 and 14 in).

Research and development testing began in the A-1 well field (Fig. A.56) in March, 1980, and was expanded to the B well field in November, 1980. Ammonium carbonate lixiviant was used in both well fields.

A.7.2 History of Mining Operations

The A-1 well field covers approximately 1/4 acre and consists of 12 wells spaced about 6.1 m (20 ft) apart arranged in a staggered pattern (Fig. A.56). Rows of injection wells alternate with rows of production wells. Approximately 189,000 L (50,000 gal) of water purified by reverse osmosis were injected into the production zone before leaching began. This was done to displace formation water with high calcium concentrations, which would react with ammonium carbonate lixiviant causing precipitation of calcium carbonate. Injection of the ammonium carbonate lixiviant and an oxidant of hydrogen peroxide and/or oxygen commenced on April 2, 1980. The lixiviant, which contained from 1 to 5 g/L of carbonate and from 1 to 4 g/L

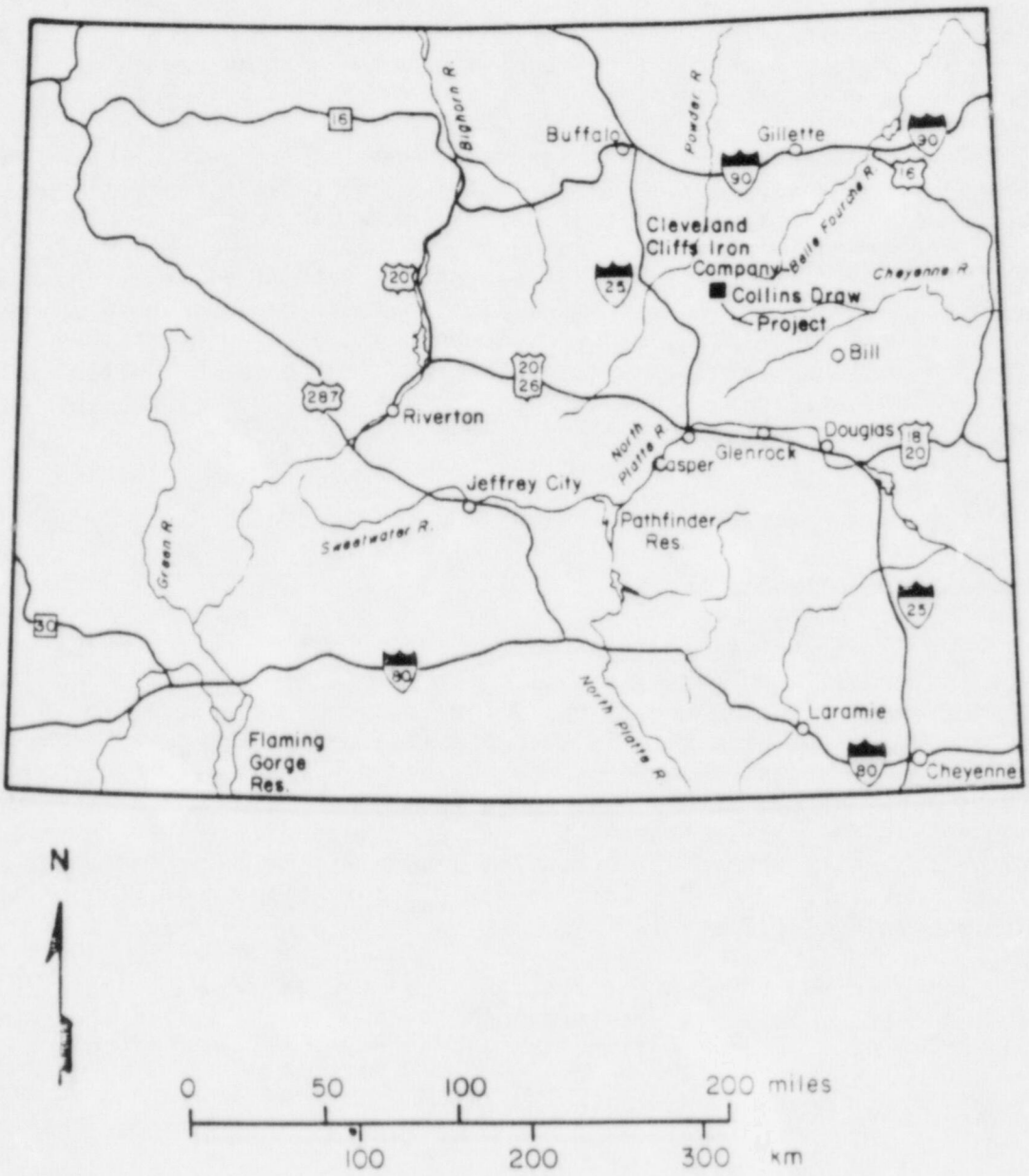


Fig. A.55. Location map of the Cleveland Cliffs Iron Company Collins Draw mine.

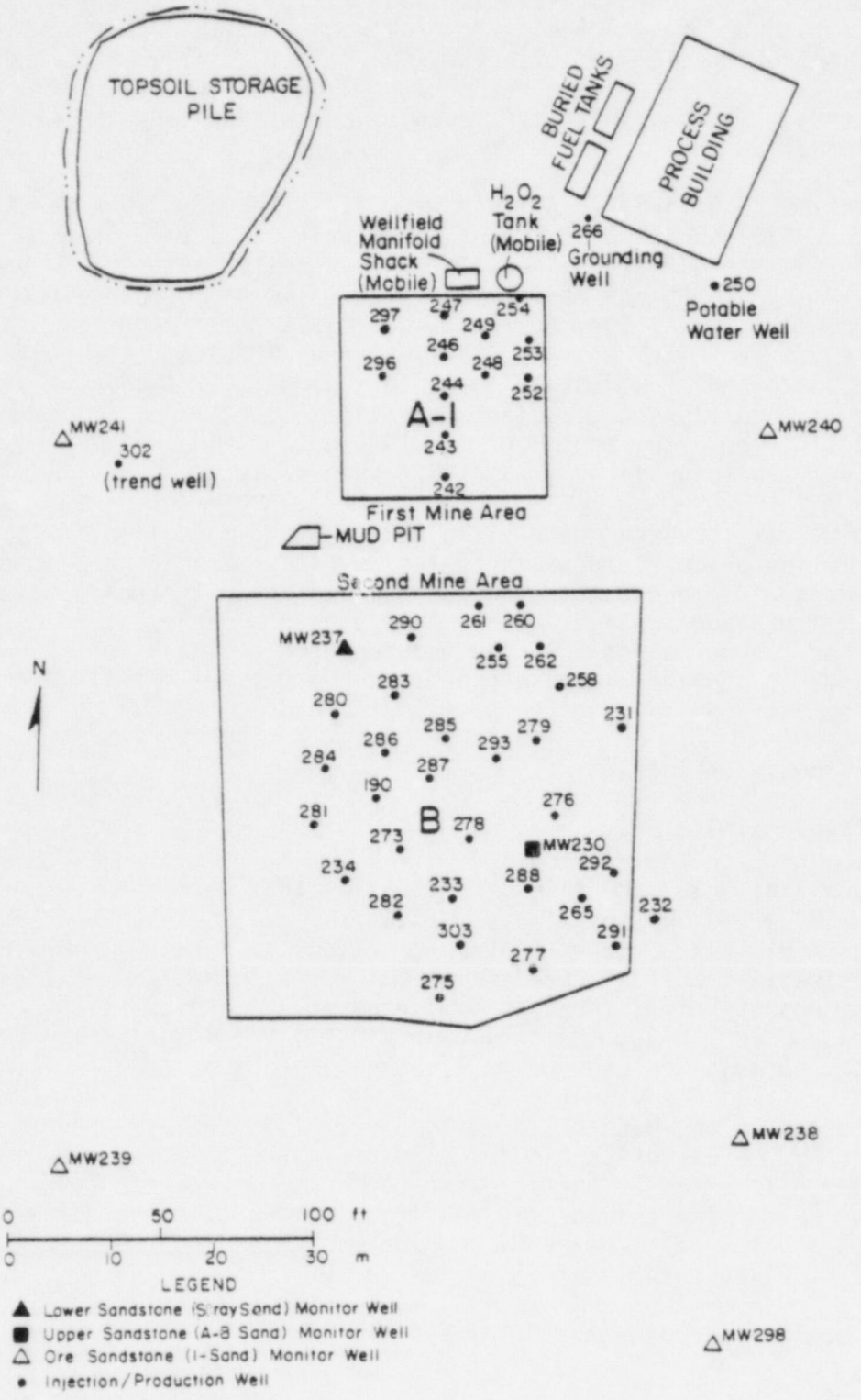


Fig. A.56. Location map of the facilities at the Collins Draw mine. *Source:* CCIC, 1981.

of ammonium, was injected at rates of 0.6 to 1.9 L/sec (10 to 30 gal/min) under well head pressures ranging from 1.1×10^5 to 2.1×10^5 N/m^2 (15 to 30 psi). Approximately 4.9×10^6 L (1.3×10^6 gal) of lixiviant were used in the A-1 well field. Mining was conducted until November, 1980, with balanced production and injection rates (i.e., no excess water was produced during mining).

Injection of lixiviant at the B well field began on November 4, 1980, with the transfer of solution from the A-1 well field production zone into the B well field production zone. The B well field consists of 33 wells; however, mining operations utilized only 20 wells which formed four contiquous production cells (Fig. A.57). These cells are irregularly shaped seven-spot patterns with well spacings of about 6 to 12 m (20 to 40 ft). The operational mode (production or injection) of any given well varied depending on conditions. Lixiviant injection and solution recovery were conducted similar to the mining of the A-1 well field. Mining continued in the B well field until July 23, 1981, when restoration was initiated.

Uranium was recovered by passing the produced solution through ion exchange resins which captured the ammonium uranyl tricarbonate complex. A strong ammonium carbonate solution was used to strip the uranium complex from the ion exchange resins. Table A.42 presents representative chemical analyses for the solutions injected and produced during B well field operations. Solutions injected and produced at the A-1 well field were of similar composition to those of the B well field.

A.7.3 Regional Setting

A.7.3.1 Regional Geology

The Collins Draw mine site is located in the Pumpkin Buttes area of the Powder River Basin approximately 16 km (10 mi) south of the Irigaray site (Fig. A.5). The Eocene Wasatch Formation crops out in the area; the ore zone strata at Collins Draw are part of this formation. The Wasatch Formation consists of alternating sequences of sandstones, siltstones, claystones and coal beds. Tectonic activity in the Powder River Basin has been minor; no major folding or faulting has been reported.

Sediments of the Wasatch Formation were deposited by meandering and braided streams. Extensive floodplain deposits of silt and clay resulted from periodic flooding of the streams; coarse sands were deposited in the stream channels. The thicknesses and areal extent of these deposits are variable; lithology also may vary considerably over short distances. Sharp et al (1964) discuss the geology of the Pumpkin Buttes area in detail.

A.7.3.2 Regional Ground-water Flow Systems

Hodson et al (1973) present a description of the hydrostratigraphic units of the Powder River Basin. In-Situ Consulting, Inc. (1978c) investigated the regional and local hydrogeology in the area of the mine site also. The following discussion is derived from these references.

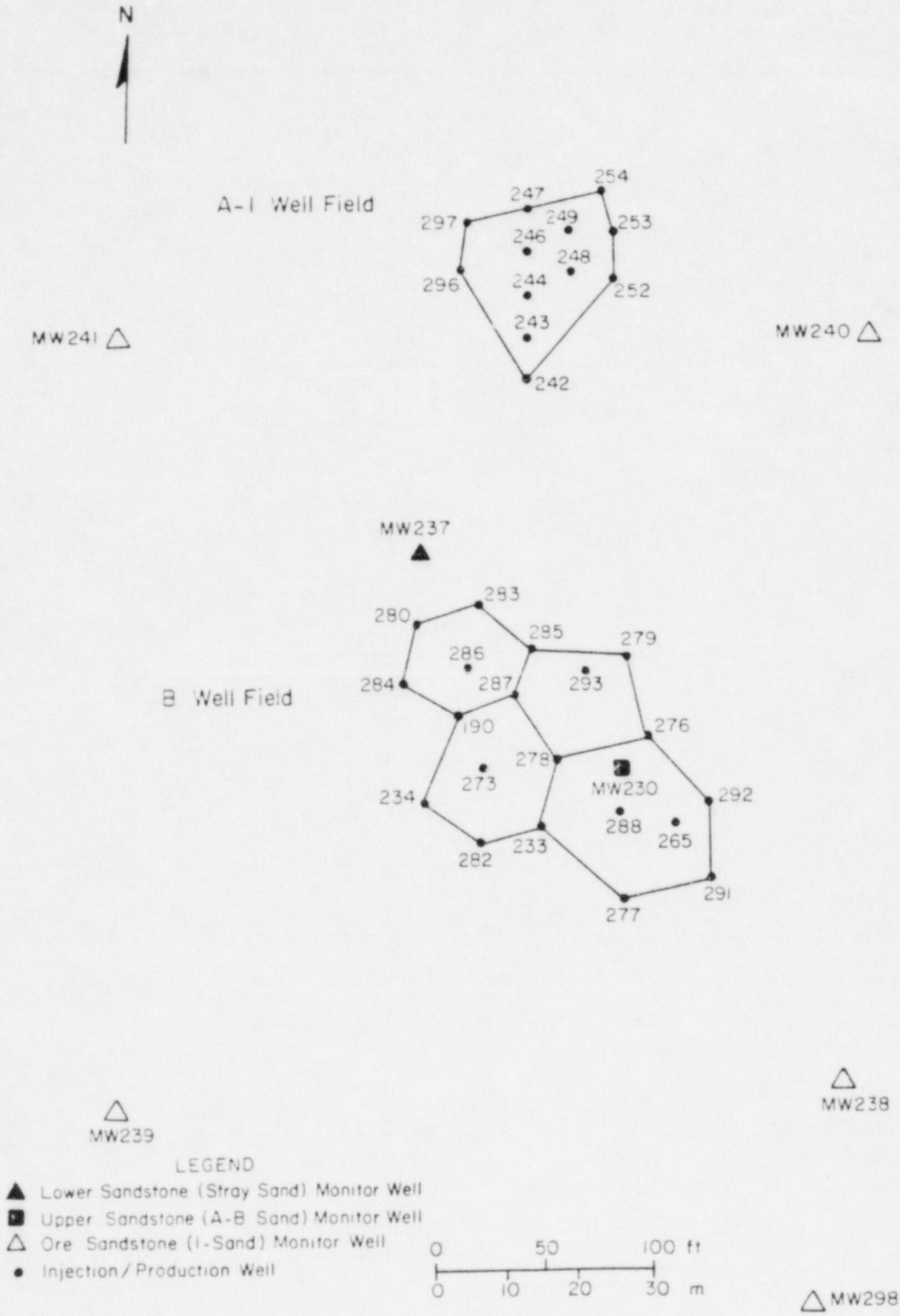


Fig. A.57. Location map of monitor wells and production/injection wells at the Collins Draw mine. *Source:* CCIC, 1981.

Table A.42. Chemical analyses[a] of lixiviant and produced solution in the B well field at the Collins Draw mine.

Parameter	Injected Lixiviant	Production Solution
TDS	1112	1112
Sodium	138	138
Calcium	40	40
Magnesium	2	2
Sulfate	420	420
Chloride	475	475
Carbonate	1000-5000	1404
Bicarbonate	1000-5000	--
Hydroxide	82	82
Ammonia (NH4 as N)	1000-4000	420
Nitrate	10.0	10.0
Nitrite	1.25	1.25
Fluoride	0.57	0.57
Total Alkalinity	2582	2582
Total Hardness	108	108
Boron	< 0.01	< 0.01
Aluminum	< 0.05	< 0.05
Arsenic	0.056	0.056
Barium	< 0.03	< 0.03
Cadmium	< 0.002	< 0.002
Chromium	< 0.01	< 0.01
Copper	< 0.01	< 0.01
Iron	< 0.01	< 0.01
Lead	< 0.01	< 0.01
Manganese	< 0.01	< 0.01
Mercury	< 0.0005	< 0.0005
Nickel	< 0.02	< 0.02
Selenium	2.2	2.2
Zinc	< 0.005	< 0.005
Molybdenum	0.06	0.06
Uranium	0-2	30-150
Vanadium	4.2	4.2
Radium-226 (pCi/L)	237.6	237.6
pH (units)	9.1-9.7	9.1
Conductivity (μmhos/cm)	4147	4147

[a]Values are mg/L except as noted.

Source: CCIC, 1981.

Generally, the hydrostratigraphy of the Wasatch Formation consists of sandstone aquifers confined by siltstone and claystone aquitards. Shallow water table (unconfined) aquifers which may also be present are of limited areal extent occurring mainly along the more prominent streams. Wells in the confined sandstone aquifers may yield several liters per second. Jointed coal beds also produce water in some areas, their yield is much less than the sandstone aquifers.

Recharge to the confined sandstone aquifers of the Wasatch Formation occurs along the front of the Bighorn Mountains and in the Black Hills; precipitation over the basin also recharges the Wasatch aquifers (In-Situ Consulting, Inc. 1978c). Natural discharge in the basin occurs primarily along the Powder and Little Powder rivers and their tributary valleys. Ground-water levels exhibit minor seasonal fluctuations.

Ground-water movement in the Wasatch Formation generally is to the north toward the Powder River and its tributaries. In shallow aquifers, ground-water movement may be controlled locally by other drainages such as the Belle Fourche River. The regional ground-water gradient in the area of the mine site, estimated from water level data for 91 wells, was reported as 0.006 (In-Situ Consulting, Inc. 1978c).

A.7.4 Description and Analysis of Data Collected

A.7.4.1. Exploration

Exploration boreholes were drilled throughout the Collins Draw mine area to evaluate the ore deposit; however information concerning the number and locations of these boreholes is not available. No details regarding methods used to plug these boreholes are available and consequently no conclusions may be drawn as to the existence of preferential pathways for vertical migration of lixiviant resulting from abandoned boreholes.

No information is available pertaining to geophysical logging at the Collins Draw mine. CCIC (1981) reported that sandstone beds of the Wasatch Formation were identified and traced laterally utilizing information derived from drill holes. It is not known whether this information included geophysical logs.

A.7.4.2 Aquifer Tests

CCIC conducted seven constant discharge pumping tests, one injection test and one slug test at the Collins Draw mine site. During five of the seven pumping tests, water levels were measured in three or more observation wells. Details of the injection test and several pumping tests have not been reported. The slug test was conducted in monitor well MW237 (Fig. A.57) completed in a silty sandstone that underlies the production sandstone to determine if the unit was suitable for monitoring vertical excursions. Table A.43 presents a summary of the aquifer tests conducted at the Collins Draw site.

Table A.43. Summary of aquifer tests conducted at the Collins Draw mine.

Test No.	Pumping Well	Location	Aquifer[a]	Total Depth Tested Well m (ft)	No. of Observation Wells	Units Monitored[a]	Type of Test[b]	Date of Test	Length of Test (min)	Discharge Rate L/s (gal/min)
1	139	B-Field	1-Sand	148 (485)	1		CD	7/15/76		2.2 (35.0)
2	139	B-Field	1-Sand	148 (485)	3		CD	7/30/76		2.5 (40.0)
3	190	B-Field	1-Sand	147 (481)	3		CD	7/30/76		2.2 (35.0)
4	139	B-Field	1-Sand	148 (485)	3		CD	8/05/76		2.1 (34.0)
5	146	B-Field	1-Sand	148 (485)	3	1-Sand, AB-Sand	CD	8/23/76	>4,000	1.6 (25.6)
6	232	B-Field	1-Sand	139 (457)	2	1-Sand, AB-Sand		11/05/76	100	
7	139	B-Field	1-Sand	148 (485)	6	1-Sand, AB-Sand	CD	11/17/76	>1,000	1.3 (20.0)
8							I			
9	237	B-Field	Stray Sand		0	Stray Sand	S	5/15/78	4.5	2.9 (45.9)

[a]1-Sand, AB-Sand, Stray Sand = Cleveland Cliffs Iron Company designations.

[b]CD = Constant Discharge; I = Injection; S = Slug.

Source: CCIC, 1981.

A summary of aquifer and aquitard characteristics estimated by CCIC from the aquifer test data is presented in Table A.44. Drawdown data are not available for aquifer tests 1 through 8. Graphical representations of test data were reported for aquifer tests 5, 6, 7, and 9. Three methods used by In-Situ Consulting, Inc. (1978c) to analyze the pumping test data are as follows: Hantush and Jacob (1955), Cooper and Jacob (1946), and Chow (1952). Data for the slug test conducted in the silty sandstone underlying the production sandstone were analyzed by the method of Cooper et al (1967). Aquifer properties estimated by this method were checked against those estimated by analysis of recovery data by the method of Ferris et al (1962). Values of horizontal hydraulic conductivity, transmissivity and storativity estimated by this method were checked against those estimated by analysis of recovery data by the method of Ferris et al (1962). Values of horizontal hydraulic conductivity, transmissivity and storativity estimated by In-Situ Consulting, Inc. (1978c) for the mined aquifer are presented in Table A.44. Values of vertical hydraulic conductivity for the aquitards overlying and underlying the tested aquifer cannot be evaluated accurately based on available test data.

In-Situ Consulting, Inc. (1978c) indicated that drawdown data for aquifer test 7 did not match the leaky type curves of Hantush and Jacob (1955), but the data were matched to the non-leaky Theis (1935) type curve. This suggests that water was not being supplied to the production aquifer by leakage from other aquifers during the pump test. However, drawdown data were not recorded for the first 10 to 15 minutes of pumping in most cases, and without the early drawdown data it is not possible to obtain a unique match to the leaky type curves. The water level in the overlying aquifer (measured in monitor well MW230) remained constant during the aquifer tests, indicating that pumping the production aquifer for the duration of the pumping test did not stress the overlying aquifer. In-Situ Consulting, Inc. (1978c) also estimated values of transmissivity and storativity for the aquifer by the Jacob straight line method (Cooper and Jacob, 1946), even though the data record was not long enough to satisfy requirements for applying this method.

Although aquifer tests 6 and 7 were not conducted for sufficient periods of time to stress large portions of the aquifer and estimates of the aquifer properties may not be representative of the entire aquifer, In-Situ Consulting, Inc. (1978c) concluded that the production aquifer was heterogeneous and anisotropic. The ratio of horizontal to vertical hydraulic conductivity was estimated to be between one and three. This ratio probably reflects stratification of the aquifer, although the presence of features such as braided stream channels and the possibility that some wells were completed in localized zones of higher or lower hydraulic conductivity also may explain the observed variability. Major transmissivity was estimated to be about 3.8 m^2/day (40 ft^2/day). In addition, a flow retarding boundary was tentatively identified about 73 m (240 ft) from the mine site trending in a north-south direction. This boundary could reflect the pinching out of the production aquifer or a facies change to less permeable strata.

Table A.44. Summary of aquifer characteristics estimated by In-Situ Consulting and by the authors from pumping/injection data for the Collins Draw mine.

Test No.	Pumping/ Injection Well	Aquifer[a]	Analysis Conducted By[b]	Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Method of Analysis[c]	Type of Test[d]
1	139	1-Sand	ISC	--	2.6 to 2.8 (28 to 30)	8.2×10^{-5}	CJ,C	CD
			Authors	1.9×10^{-4} to 2.0×10^{-4} (0.54 to 0.58)	--	--		
2	139	1-Sand	ISC	--	1.8 to 3.3 (19 to 35)	1.2×10^{-4} to 3.2×10^{-4}	CJ,C,H	CD
			Authors	1.3×10^{-4} to 2.4×10^{-4} (0.37 to 0.67)	--	--		
3	190	1-Sand	ISC	--	1.7 to 3.1 (18 to 33)	3.1×10^{-4} to 1.5×10^{-3}	CJ,C,H	CD
			Authors	1.2×10^{-4} to 2.2×10^{-4} (0.35 to 0.67)	--	--		
4	139	1-Sand	ISC	--	1.7 to 3.1 (18 to 33)	3.1×10^{-4} to 5.5×10^{-3}	CJ,C	CD
			Authors	1.3×10^{-4} to 2.2×10^{-4} (0.38 to 0.63)	--	--		
5	146	1-Sand	ISC	--	1.8 to 2.4 (19 to 26)	1.1×10^{-4} to 3.5×10^{-3}	CJ,C,H	CD
			Authors	1.3×10^{-4} to 1.8×10^{-4} (0.38 to 0.69)	--	--		
6	323	1-Sand	ISC	--	1.9 to 3.3 (20 to 36)	4.4×10^{-5} to 6.0×10^{-5}	CJ, H	CD
			Authors	1.3×10^{-4} to 2.4×10^{-4} (0.38 to 0.69)	--	--		
7	139	1-Sand	ISC	--	1.3 to 3.3 (14 to 36)	5.2×10^{-5} to 5.6×10^{-4}	CJ,C, H	CD
			Authors	1.0×10^{-4} to 2.4×10^{-4} (0.27 to 0.69)	--	--		
8	--	--	--	--	--	--	--	--
9	237	Stray Sand	ISC	8.8×10^{-6} (0.025)	0.03 (0.3)	7.9×10^{-6}	F	S
			Authors	8.8×10^{-6} (0.025)	--	--		

[a] 1-Sand = Cleveland Cliffs Iron Company designation.

[b] ISC - In-Situ Consulting.

[c] CJ = Cooper and Jacob (1946); C = Chow (1952); H = Hantush (1960); F = Ferris, et al (1962).

[d] CD - Constant Discharge; S = Slug.

Source: CCIC, 1981 and In-Situ Consulting, Inc., 1978c.

In-Situ Consulting, Inc. (1978c) concluded from aquifer test 9 that the Stray Sand unit underlying the mineralized sandstone has very low hydraulic conductivity and storativity values. Therefore, the unit was considered not to exhibit leakage and was not monitored for vertical excursion detection. These conclusions are suspect, however, due to lack of information concerning the completion of monitor well MW237 (tested well). Ferris et al (1962) point out that serious errors in estimates of aquifer parameters can result if the well does not fully penetrate the aquifer. As with any single well testing method, slug tests provide estimates of aquifer parameters at a point within the aquifer, but may not be representative of the entire aquifer.

A.7.5 Hydrogeology of the Mine Site

Uranium mineralization at the Collins Draw mine occurs as roll-front deposits in a sandstone bed of the Wasatch Formation. This sandstone is designated the 1-Sand by CCIC. The 1-Sand is present at a depth of about 130 m (425 ft); it has an average thickness of about 15 to 17 m (50 to 55 ft) (CCIC, 1981). All the sandstone units at Collins Draw are similar in composition. They are gray, feldspathic to arkosic fluvial deposited units, which may be altered locally to yellow-brown or red. The quartz grains range from fine to very coarse and may be rounded to angular. Interstitial clays may comprise as much as 25% to 30% of the total sandstone volume (CCIC, 1981). The sandstone units are separated by claystone units which contain considerable silt. The units identified by CCIC are depicted in cross section on Fig A.58. Detailed lithologic information on these units is not available. Figure A.59 presents a generalized stratigraphic section for the mine site.

The 1-Sand was divided into three different layers in the hydrologic evaluation of the site conducted for CCIC (In-Situ Consulting, Inc. 1978c). The upper 4.6 m (15 ft) and lower 4.6 m (15 ft) of the sand unit were observed to be cleaner (i.e., less clayey) than the 6 to 7.6 m (20 to 25 ft) interior portion. The interior portion has lower porosity and hydraulic conductivity than the upper and lower portions of the unit.

There are three major water-bearing sandstone units (the 1, AB, and C sands) at the Collins Draw mine site; however, available information pertains only to local ground-water conditions in the 1-Sand. The 1-Sand is a confined aquifer, with ground-water flow in a direction 19° west of north and a local gradient of about 0.008 (In-Situ Consulting, Inc., 1978c). Water level fluctuations observed over a one and one-half year period were on the order of a few centimeters.

A.7.6 Monitor Well System and Well Construction

Monitoring for lateral excursions at the Collins Draw mine was accomplished with five monitor wells which surround the production well fields (Fig. A.57). These wells are located approximately 61 m (200 ft) from the perimeter of the A-1 and B well fields and are completed in the ore-bearing 1-Sand aquifer. Two additional monitor wells, MW230 and MW237, were constructed to detect vertical excursions from the production sandstone.

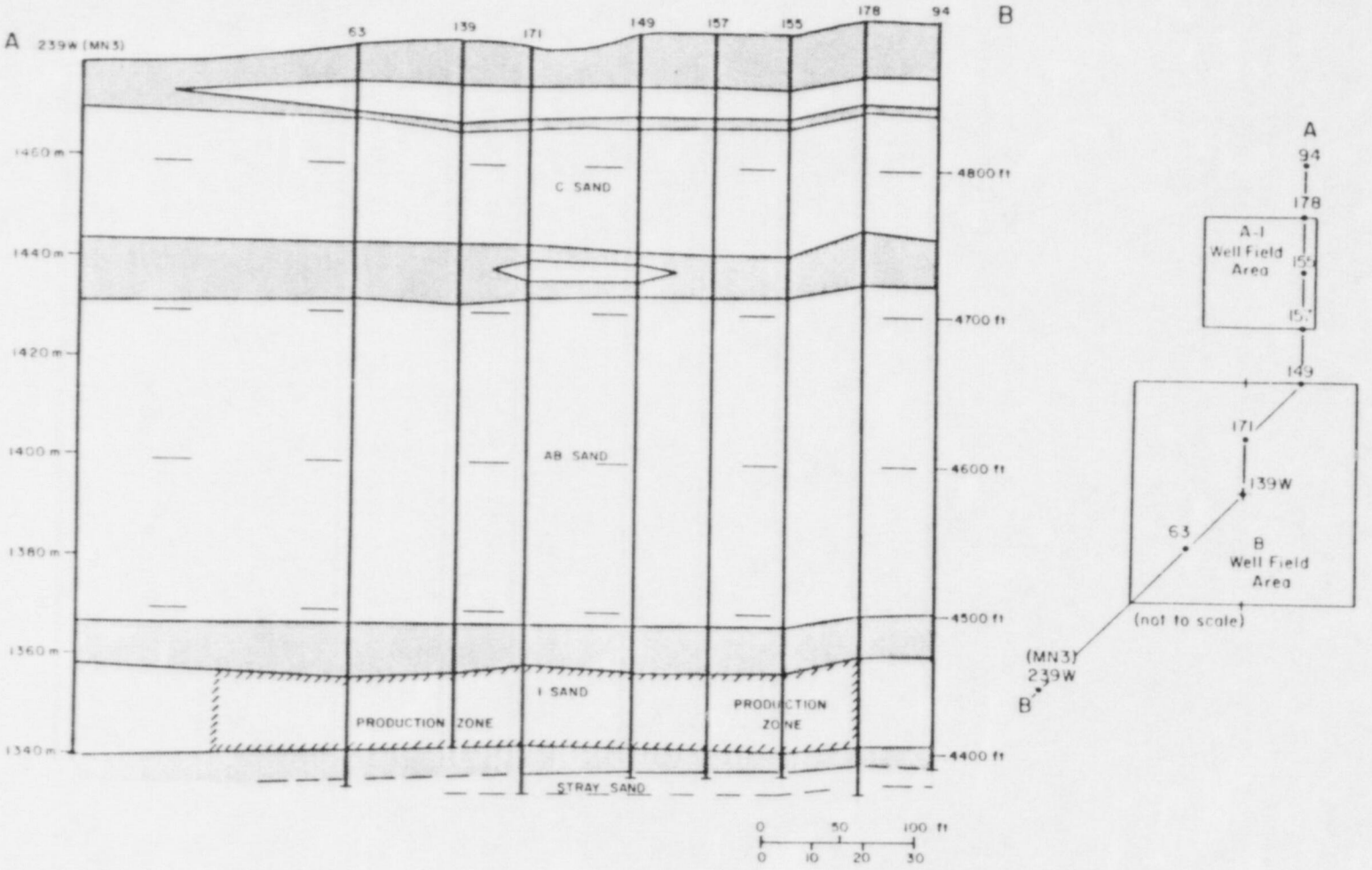


Fig. A.58. Cross section through the Collins Draw mine area. *Source:* CCIC, 1981.

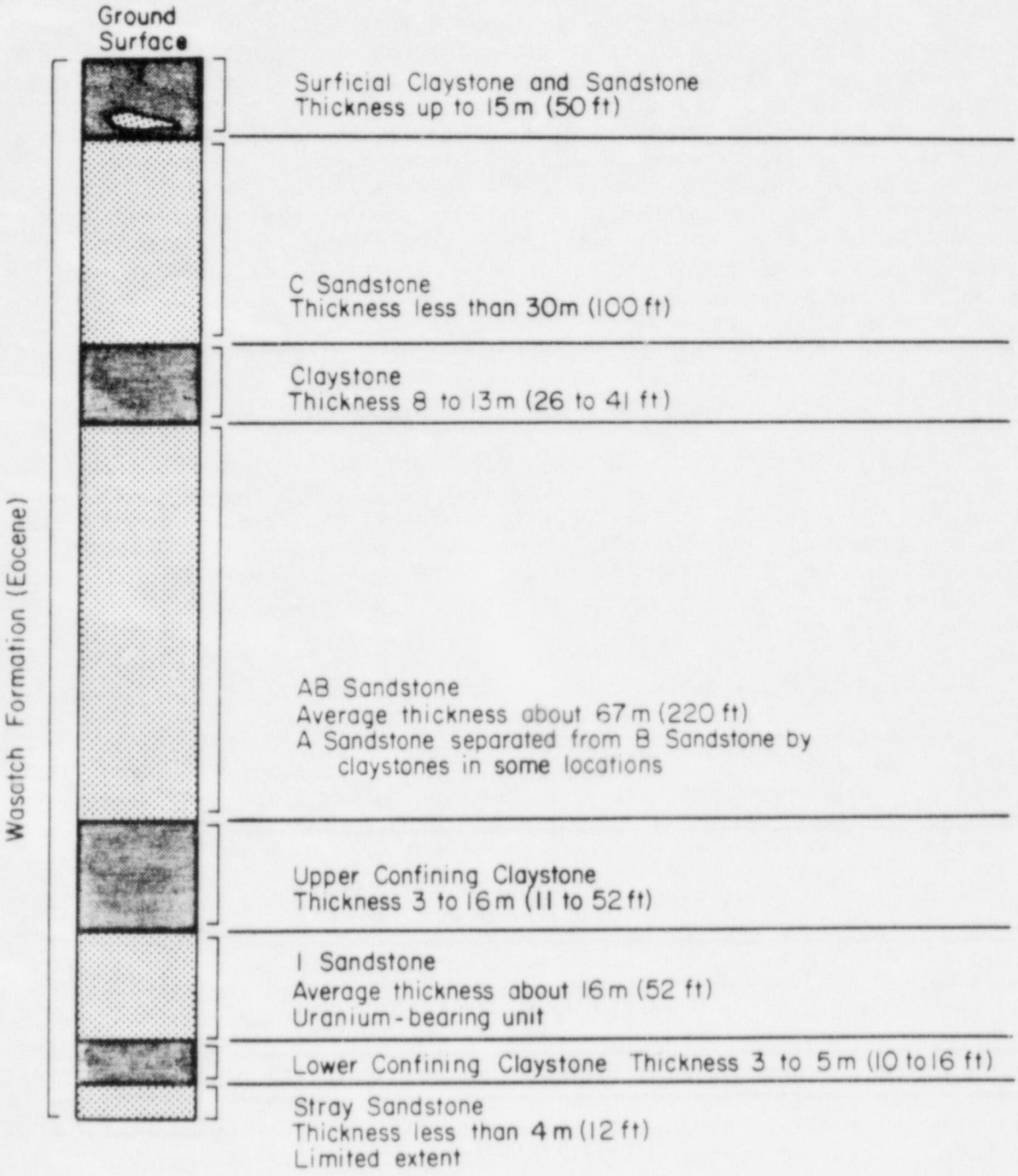


Fig. A.59. Generalized stratigraphic section for the Collins Draw project area. *Source:* CCIC, 1981.

Monitor well MW230 was completed in the AB-Sand which overlies the 1-Sand; monitor well MW237 was completed in the Stray Sand which underlies the 1-Sand. Both wells are located within the B well field.

Monitor and production/injection wells were sampled to evaluate baseline water quality and facilitate establishment of excursion UCL's. Table A.45 presents the average baseline concentrations of selected parameters for each aquifer monitored. Excursion UCL's were set for the excursion indicators pH, TDS, bicarbonate plus carbonate, ammonia, sulfate and uranium. These limits were established by adding two units to average baseline pH, adding 1 mg/L to average baseline uranium concentration, and adding 15% to the average baseline concentration of the remaining parameters (Table A.46). Monitor wells were sampled every two weeks for these parameters. A monitor well was placed on excursion status when the concentration of one parameter exceeded the UCL as verified by a confirmation sample taken within 48 hours of the original sample. Monitor well MW237, which was constructed to detect vertical excursions into the strata underlying the production zone, was apparently not sampled regularly due to low yields of the well, and data are not available concerning excursions.

Little information is available concerning well construction techniques utilized at the Collins Draw mine. CCIC (1981) reported that boreholes were drilled to the top of the mineralized sand, and casing was set and cemented. The boreholes then were deepened by drilling through the mineralized sandstone. The mineralized sandstone interval was screened, left as open hole, or cased with perforated pipe. Fig. A.60 shows a typical well construction.

A.7.7 Analysis of Excursions

The Collins Draw mine has experienced excursions in five monitor wells. One of these excursions has been determined to be a false vertical excursion detected in the overlying AB-Sand aquifer. The remaining four occurred in the 1-Sand. A summary of these excursions is presented in Table A.47.

The concentration of sulfate in monitor well MW230 in the overlying AB-Sand aquifer, exceeded the UCL on the first day of operation. Further analysis of this event revealed that 1.9×10^6 L (5×10^5 gal) of water were withdrawn from the AB-Sand aquifer to make up the lixiviant used in mining the A-1 well field. Such a withdrawal of water probably caused increased ground-water flow velocities in the AB-Sand and may have caused a rise in the concentration of sulfate due to dissolution of sulfide minerals (NRC, 1981b). Based on the timing of this event and the circumstances involved, the NRC resolved that there had been no migration of mining solutions from the production zone upward into the AB-Sand aquifer. Samples collected from the well during April and May, 1980, also contained sulfate in excess of the UCL. By May 28, 1980, the sulfate concentration dropped below the UCL. No other excursion indicators were found to be above baseline concentrations.

Table A.45. Average baseline concentrations[a] for the AB-Sand, 1-Sand, and Stray Sand aquifers at the Collins Draw mine.

Parameter	AB-Sand	1-Sand	Stray Sand
TDS (calculated)	543	394	322
TDS (105°C)	544	407	323
Sodium (calculated)	143	106	103
Sodium (observed)	147	105	90
Potassium	7	7	9
Calcium	37	28	12
Magnesium	5	2	2
Sulfate	156	150	68
Chloride	93	19	44
Carbonate	0	9	28
Bicarbonate	207	150	114
Ammonia as N	0.33	0.11	0.2
Nitrate as N	0.02	0.14	0.4
Nitrite as N	0.09	0.04	0.2
Aluminum [.05][b]	ND[c]	ND	ND
Arsenic [.01]	0.024	ND	ND
Barium [.05]	ND	ND	ND
Boron [1.0]	ND	ND	ND
Cadmium [.002]	ND	ND	ND
Chromium [.01]	ND	ND	ND
Copper [.01]	0.01	ND	ND
Fluoride	0.13	0.17	0.4
Iron [.01]	2.85	0.24	0.1
Lead	ND	ND	ND
Manganese [.01]	0.05	0.02	ND
Mercury [.001]	ND	ND	ND
Selenium [.01]	0.07	ND	ND
Nickel [.04]	ND	ND	ND
Zinc [.01]	0.01	ND	ND
Molybdenum [.05]	ND	ND	ND
Vanadium [.05]	ND	ND	ND
Uranium [.001]	0.03	0.09	0.02
Radium-226 (pCi/L)	4.4	16.2	6.2
Temperature °C (Field)	14.2	14.1	15
Conductivity 77°F (Lab) (µmhos/cm)	743	621	540
Conductivity 25°C (Field) (µmhos/cm)	1607	1245	967
pH (Lab) (units)	8.0	8.2	8.8
pH (Field) (units)	7.5	7.6	9.1

[a]Values are mg/L except as noted.

[b][ ] Detection limits.

[c]ND--Not detected.

Source: Modified from NRC, 1983b.

Table A.46. Excursion indicators and upper control limits for the Collins Draw mine.[a]

Parameter	Well					
	MW230	MW238	MW239	MW240	MW241	MW298
TDS	720	453	568	508	488	474
Bicarbonate + Carbonate	365	190	263	182	197	201
Ammonia	0.53	0.41	0.16	0.14	0.14	0.31
Sulfate	297	170	178	167	161	207
Uranium	1.06	1.47	1.18	1.03	1.07	1.05
pH (units)	10.1	10.6	10.1	10.2	10.5	10.1

[a]Values are mg/L except as noted.

Source: NRC, 1981b and CCIC, 1981.

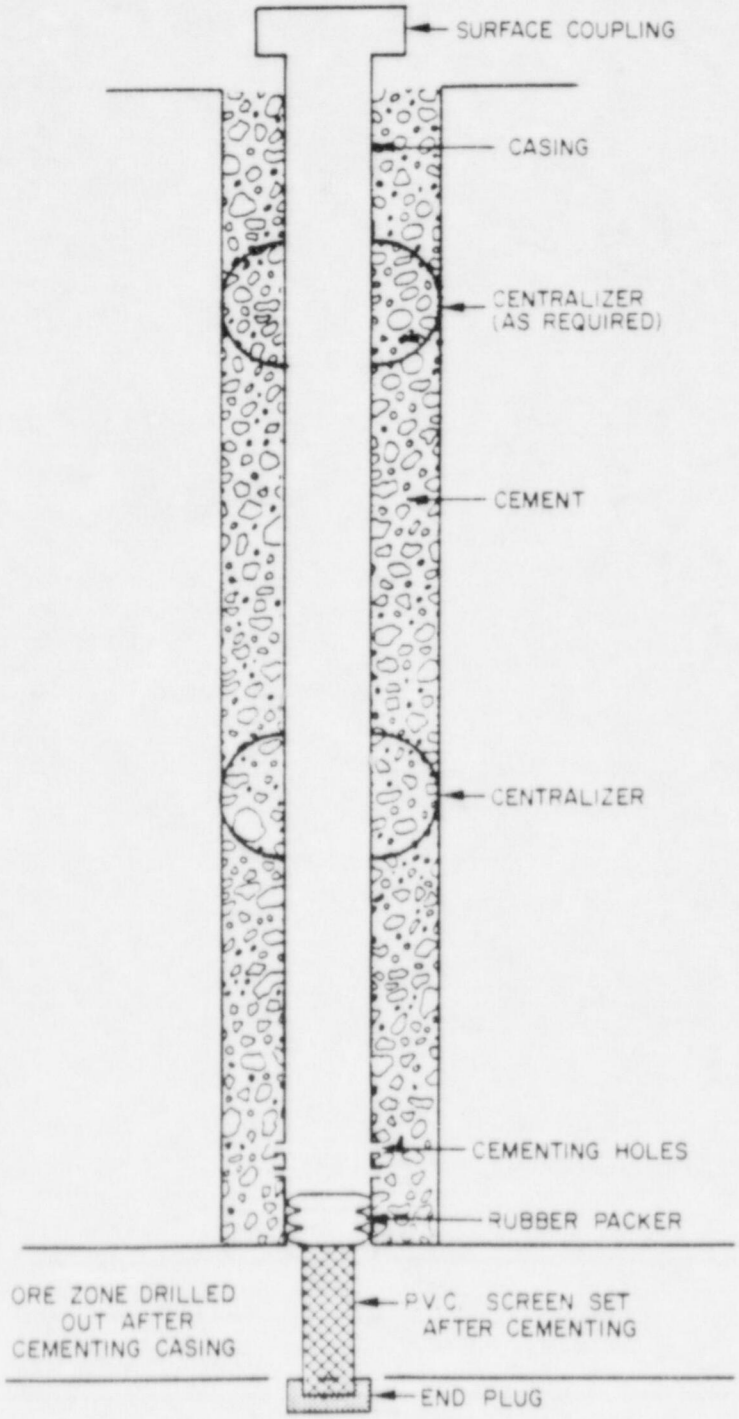


Fig. A.60. Typical well completion at the Collins Draw mine. *Source:* CCIC, 1981.

Table A.47. Summary of excursions at the Collins Draw mine.

Well	Location	Aquifer[a]	Date of Sample[b]	Excursion Parameters TDS mg/L	Excursion Parameters SO_4 mg/L	Excursion History On	Excursion History Off	Excursion History Duration
MW230	B Field	AB-Sand	UCL	720	297	4/2/80	5/28/80	2 months
			4/2/80	---	---			
MW238	SE of B Field	1-Sand	UCL	453	170	1/21/81	c	c
			1/25/81	---	225			
MW241	NW of B Field	1-Sand	UCL	488	161	4/14/81	7/81	3 months
			4/14/81	694	255			
MW240	NE of B Field	1-Sand	UCL	508	167	4/81	--[d]	--
MW298	SE of B Field	1-Sand	UCL	474	207	8/81	--[d]	--

[a]AB-Sand, 1-Sand = Cleveland Cliffs Iron Company designations.

[b]UCL = Upper Control Limits

[c]Monitor well MW238 did not go off excursion status; its status was changed to that of a trend well (7/81) and a new monitor well (MW298) took its place.

[d]Remained on excursion status during restoration operations, details unavailable.

Source: NRC, 1981c and 1983b.

On January 21, 1981, ore zone monitor well MW238, which is located 33.5 m (110 ft) from the B well field, was reported on excursion status due to TDS concentrations in excess of the UCL. On February 21, 1981, sulfate concentrations in addition to TDS were found to be in excess of the UCL's. By July, 1981, uranium also exceeded the UCL with concentrations greater than 3.0 mg/L. Significant increases in the concentrations of most major and minor constituents including potassium, chloride, nitrate, calcium, selenium, and ^{226}Ra were detected during the period April 9 to May 6, 1981. CCIC altered production operations in the B well field in an attempt to control the excursion at monitor well MW238. Monitor well 291 (the closest well to MW238) was pumped in April, 1981, to draw mining solutions back toward the well field. This corrective action failed to bring excursion indicator concentrations below UCL's. Due to the short distance of well MW238 to the production well field and the inability by CCIC to effectively mitigate the excursion, the NRC allowed the replacement of well MW238 by monitor well MW298 at a distance about 61 m (200 ft) from the production well field. Subsequently, well MW238 was not considered a monitor well for excursion detection purposes but was utilized as a trend well.

In April, 1981, ore zone monitor well MW241 was placed on excursion status due to concentrations of TDS and sulfate in excess of UCL's. Soon thereafter, ore zone monitor well MW240 also was placed on excursion status due to elevated TDS and sulfate concentrations. Pumping of wells 296 and 297 in the A-1 well field was initiated to control these excursions, but these wells remained on excursion status until restoration of the B well field began.

In August, 1981, the new ore zone monitor well MW298 was placed on excursion status due to TDS and sulfate concentrations in excess of UCL's. By November, 1981, uranium concentrations in monitor well MW298 exceeded 41 mg/L. Wells MW238 and MW298 were pumped in an effort to control the excursion in this area.

The mode of operation at the Collins Draw mine (balanced injection and production rates with no bleed stream) almost certainly contributed to the occurrence of the excursions described above. CCIC was unable to maintain a hydraulic gradient toward the production wells during mining. Additionally, because this operational mode did not require construction of waste water evaporation ponds, CCIC did not have the option of pumping large quantities of water from the well fields to control excursions. This fact probably contributed to the problems experienced in mitigating the excursions.

A.7.8 Restoration

Several techniques were used to restore water quality in the ore zone aquifer at the Collins Draw mine. Restoration of the A-1 well field was initiated on November 4, 1980, with the transfer of lixiviant from the A-1 well field to the B well field. A total of about 3.48×10^6 L (9.19×10^5 gal) of solution, or about 5.4 pore volumes, were pumped from the A-1 well field ore zone. This resulted in a partial ground-water sweep of the mine area and a general improvement in ground-water quality (i.e., ammonia

concentrations were reduced from 720 mg/L to 420 mg/L). A more complete ground-water sweep was considered impractical by CCIC because no evaporation ponds had been constructed to contain the quantity of water which would be produced by an extended ground-water sweep.

Clean water recirculation with treatment by ion exchange was employed subsequent to the lixiviant transfer to reduce further the concentrations of ammonium (NH_4^+) and uranium in the ground water. However, the lower pH water injected into the ore zone dissolved calcium from the host sandstone, and the increased calcium concentrations resulted in the displacement of ammonium and hydrogen ions by calcium in the cation exchange resin and in the production zone clays. This resulted in an increase in ammonia concentration from 420 to 630 mg/L (NRC, 1983b). About 2.84×10^6 L (7.5×10^5 gal) of water were treated by ion exchange; the process was terminated due to poor efficiency and reduced well circulation.

Reverse osmosis technology was utilized next for restoration of the A-1 well field. This method was utilized for only 33 days because CCIC believed it was ineffective in reducing ammonia concentrations. About 6.5 pore volumes of water were treated by reverse osmosis. The waste stream was discharged to a process discharge drain field.

Removal of ammonia by air stripping was the final method of restoration used at the A-1 well field. Sodium and/or potassium was added to ground water produced from the well field to promote the volatilization of ammonia. The water then was circulated through an air stripping column to aid dispersion of ammonia to the atmosphere. The relatively small quantity of ammonia released to the atmosphere was found to have no adverse impact on air quality (NRC, 1981c). The treated water was reinjected into the ore zone following this procedure. Air stripping was discontinued on July 25, 1981, when ammonia levels apparently had stabilized at 25 mg/L. Approximately 26 pore volumes of water had been circulated through the air stripping column and subsequently reinjected.

Restoration of the B well field began on July 25, 1981. Air stripping for ammonia removal was utilized until January, 1982. Over 12 pore volumes of ground water were treated, and ammonia levels were reduced from about 3400 to 185 mg/L. A ground-water sweep of the B well field was initiated subsequent to the acquisition of an NPDES surface discharge permit. From February 2 to March 23, 1982, 1.6 pore volumes of ground water were removed from the ore zone, circulated through ion exchange columns to reduce contaminants to levels prescribed by the permit, and discharged to a dry stream bed at the site. Overall ground-water quality was improved and ammonia levels dropped to 120 mg/L following this restoration work. Air stripping was employed again from April to August, 1982, to reduce ammonia further. An additional 5.6 pore volumes of ground water were treated and reinjected during this time. Ground-water sweeping was conducted from August to November, 1982, to reduce further the concentrations of ammonia and other contaminants. Approximately 7.5×10^5 L (1.98×10^5 gal) of fresh water from the aquifer overlying the ore zone were injected into the ore zone near the end of this final phase of restoration of the B well field in an attempt to reduce high ammonia concentrations in localized

areas. Ammonia levels averaged 2.54 mg/L at the end of B well field restoration activities.

Following a review of water quality data for a six month post-restoration stability period in the A-1 well field, the NRC determined that TDS, ammonia, arsenic, selenium, vanadium, ^{226}Ra and pH were parameters whose average concentrations exceeded baseline or Wyoming Department of Environmental Quality Class I (Domestic) standards (Table A.48). Ten months of post-restoration stability monitoring in the B well field indicated that ammonia, TDS, selenium, uranium, sulfate, pH and ^{226}Ra continued to exceed baseline and/or domestic quality standards (Table A.49). Concentrations of ammonia, selenium and uranium in the B well field were found to be increasing over the stabilization period. For these reasons, the NRC feels that restoration of the Collins Draw mine is not adequate and further documentation of present ore zone water quality should be provided.

The absense of an evaporation pond contributed to the failure of the in situ uranium mining test in several ways. First, the ore zone aquifer was not restored adequately. Second, a short-term discharge of treated ground water into a dry gulch was necessary. Third, there was potential contamination of the sediments downstream of the point of discharge. Fourth, there was short-term deterioration of air quality through atmospheric venting of ammonia. Finally, inadequate handling capacity of waste streams contributed to the occurrence of excursions and the difficulty experienced in ameliorating the excursions.

A.8 WYOMING MINERAL CORPORATION - BRUNI PROJECT

A.8.1 Description of the Site

Westinghouse Electric Corporation, the parent company of Wyoming Mineral Corporation (WMC), operates the Bruni mine in Webb County, Texas. The mine is located on the Coastal Plain in southwestern Texas, about 11 km (7 mi) north of the town of Bruni (Fig. A.61). Topography of the mine site consists of a gently eastward sloping plain. The elevation is about 275 m (900 ft) at the site. All streams in the area are ephemeral and flow toward the Gulf of Mexico. Annual precipitation is about 50 cm (20 in).

The uranium recovery operations at Bruni are similar to those utilized at the WMC Irigaray mine in Wyoming; however, five-spot patterns are used at Bruni compared to the seven-spot patterns at Irigaray. WMC's total permit area at Bruni exceeds 599 ha (1480 acres), although only about 20 ha (50 acres) are designated as production areas. Ammonium carbonate/bicarbonate lixiviant is used to mine the Bruni uranium deposits.

A.8.2 History of Mining Operations

Research and development testing of in situ uranium recovery from the Bruni ore deposits began in 1972, but details of these activities are not available. The original permit issued by the Texas Department of Water

Table A.48. Comparison of water quality[a] before and after restoration of the A-1 well field at the Collins Draw mine.

Parameter	Baseline Average[b]	Class I Standard	Average after 6 Month Stability Period[c]
TDS	414	500	582
Ammonia	0.18	0.5	35
Arsenic	<0.02	0.05	0.335
Selenium	<0.01	0.01	0.79
Vanadium	<0.05	--	0.33
Radium-226 (pCi/L)	21.6	5	>100
pH (units)	7.5 to 8.7	6.5 to 8.5	9.1

[a]Values are mg/L except a noted.

[b]Average of all wells sampled in A-1 and B well fields.

[c]Average of A-1 wells.

Source: NRC, 1983b and CCIC, 1982.

Table A.49. Comparison of water quality[a] before and after restoration of the B well field at the Collins Draw mine.

Parameter	Baseline Average[b]	Class I Standard	Average After 10 Month Stability Period[c]
TDS	414	500	582
Ammonia	159	250	252
Ammonia	0.18	0.5	34.3
Selenium	<0.01	0.01	1.02
Uranium	0.05	5.0	5.28
Radium-226 (pCi/L)	21.6	5.0	54.3[d]
pH (units)	6.0 to 8.7	6.5 to 8.5	8.7

[a]Values are mg/L except as noted.

[b]Average of all wells sampled in A-1 and B well fields.

[c]Average of B wells.

[d]Value reported is for six months after restoration.

Source: NRC, 1983b and CCIC, 1982.

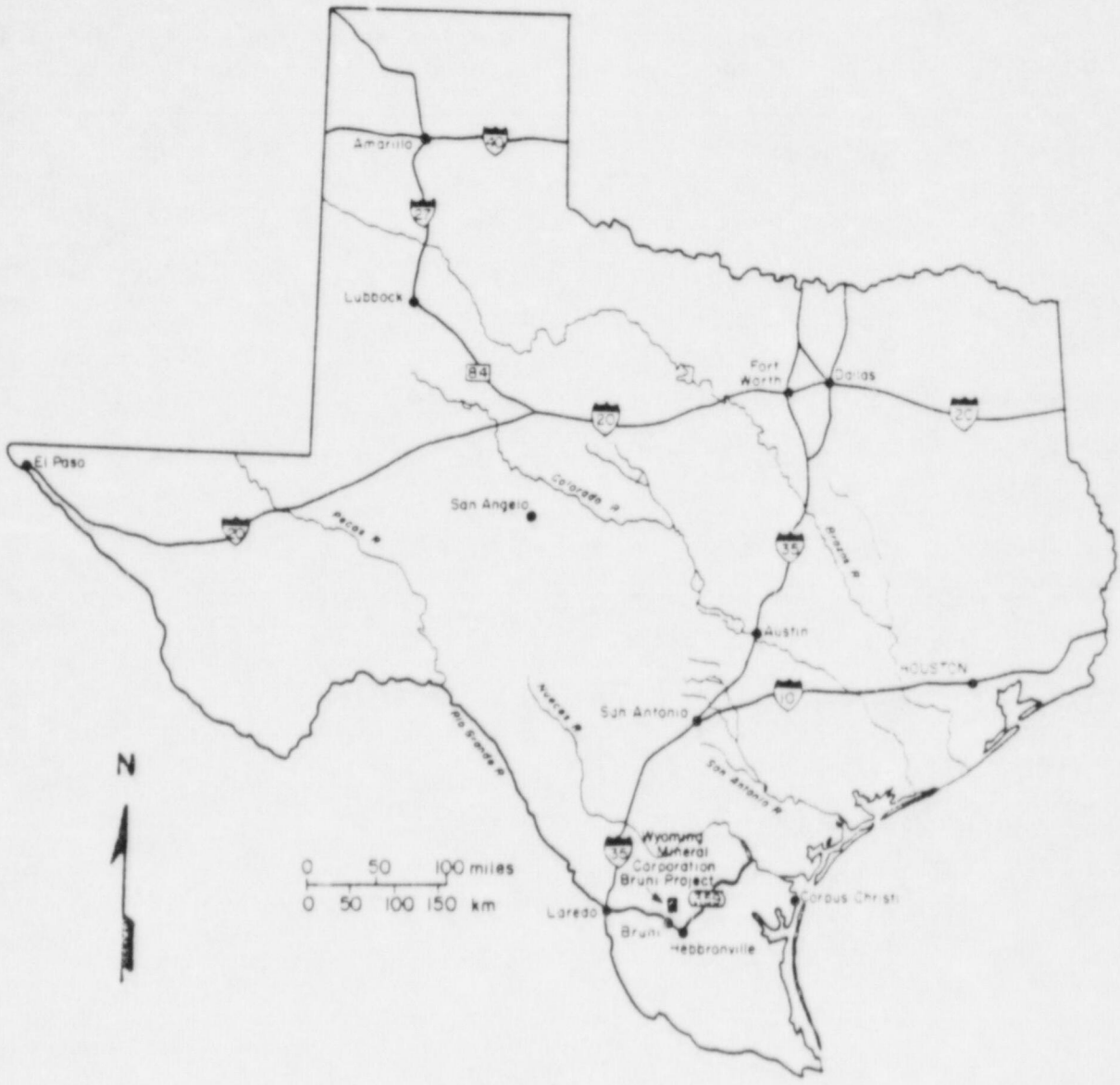


Fig. A.61. Location map of the Wyoming Mineral Corporation Bruni mine.

Resources in June, 1975, authorized commercial production of uranium from Production Areas 1 and 2 (Fig. A.62) which encompass approximately 4.8 ha (12 acres). Mining expanded into Production Area 3 in 1977 and into Production Areas 4, 5, and 6 between 1978 and 1981. Over 263,000 kg (595,000 lb) of yellowcake have been produced. WMC requested authorization from the Texas Department of Water Resources in November, 1981, to place the Bruni mine in a standby mode for a period of five years due to economic considerations. Mining solutions are contained within the production areas by pumping 24 production wells at a combined rate of 1.6 L/s (25 gal/min).

Five-spot well patterns with variable spacing are utilized at the Bruni mine throughout the production areas. Over 1720 injection/production wells exist at the site (WMC, 1982a). Due to the great number of wells and the lack of reliable information concerning their locations, we have not attempted to represent the locations of all wells within the production areas.

Ammonium carbonate/bicarbonate lixiviant with hydrogen peroxide as an oxidizing agent was injected into the production zone at 0.32 to 0.38 L/s (5 to 6 gal/min). Production wells were pumped at 0.32 to 0.5 L/s (5 to 8 gal/min). Table A.50 presents the chemical characteristics of the injected lixiviant. Uranium was recovered by passing produced solutions through anionic exchange resins which captured the complexed uranium ions. Ammonium chloride solution was used to strip the uranium from the ion exchange resins. Although no attempts to restore production zone water quality over a production area has been made, a small-scale restoration effort was conducted in Well Field AB from October, 1980, to March, 1981. Restoration is discussed in Section A.8.8.

Several problems have been encountered during the history of the Bruni mine. Soon after mining commenced in Production Areas 1 and 2, lixiviant was observed discharging at the land surface through unplugged exploration boreholes and around improperly constructed injection wells (WMC, 1977b). This contaminated a shallow gravel unit (the base of the Goliad Formation at the Bruni site). Additionally, leakage of process waste water from an evaporation pond due to failure of a bentonite liner contaminated the shallow gravel unit in portions of Production Areas 3, 4, and 5. As a result of the shallow gravel contamination and other violations of their permit conditions, a civil penalty was levied on WMC by the state of Texas in January, 1978. In addition, WMC was ordered to clean up the shallow gravel unit and monitor thoroughly the extent of contamination within it. Twenty-six wells have been utilized to inject clean water into the shallow gravel along the perimeter of the contaminated area; other wells completed in the shallow gravel withdraw ground water from the interior portions of the contaminated area. This procedure apparently has contained the contamination, although the shallow gravel remains contaminated over a large portion of the production area.

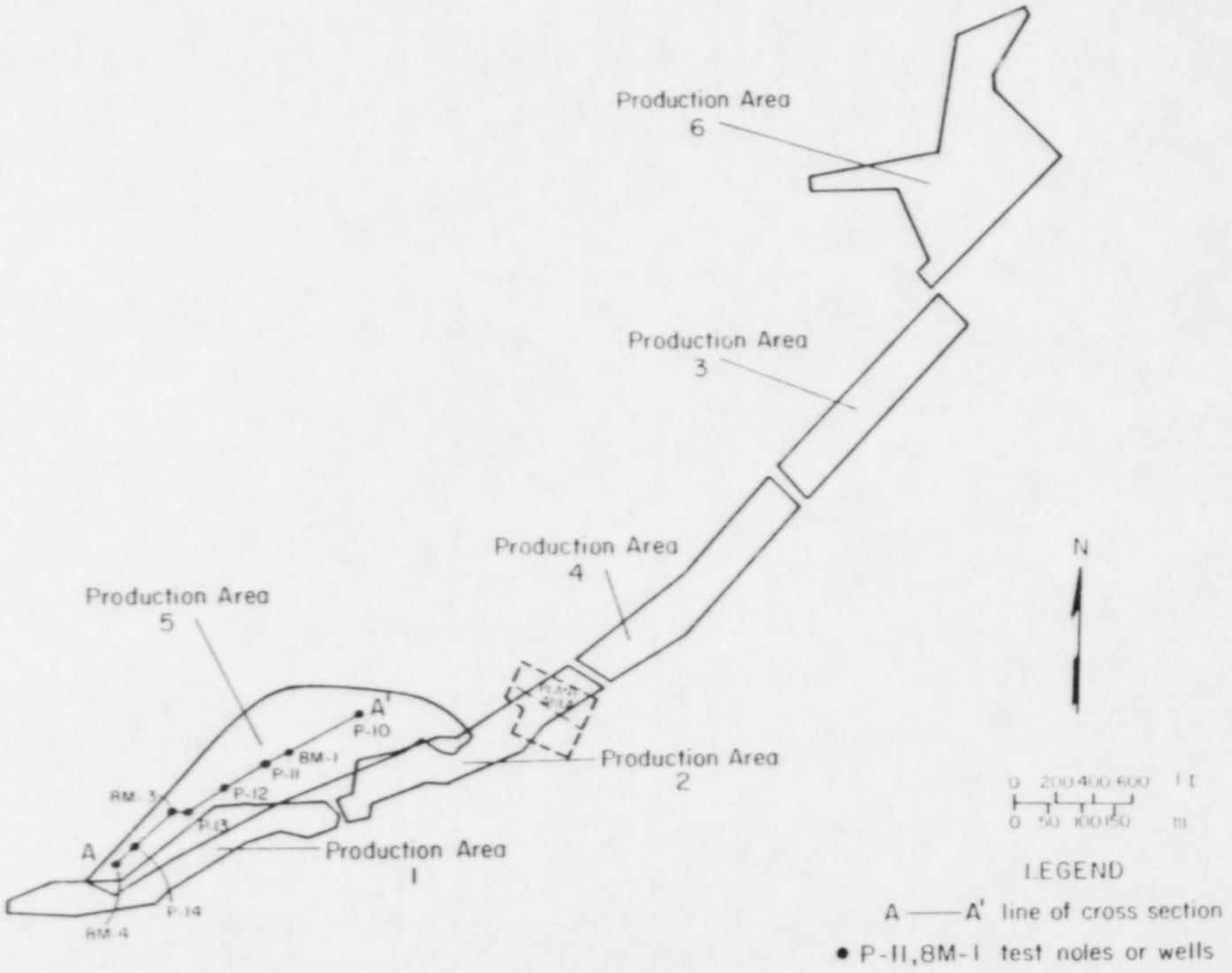


Fig. A.62. Location map of the Bruni mine facilities.

Table A.50. Typical composition of injected lixiviant at the Bruni mine.

Parameter	Concentration[a]
Ammonia (NH_4 as N)	2290 to 2360
Bicarbonate	1000 to 3000
Hydrogen Peroxide	0 to 1500
Sodium	400 to 800
Chloride	2000 to 3500
Sulfate	500 to 1500
Calcium	10 to 50
Molybdenum	0 to 3
Magnesium	150 to 350
Uranium (U_3O_8)	1.5 to 2.0
pH (units)	8.0 to 9.0

[a]Values are mg/L except as noted.

Source: WMC, 1977b and Texas Department of Water Resources, 1977.

A.8.3 Regional Setting

A.8.3.1 Regional Geology

The Bruni mine is located on the Hebronville Plain, a subdivision of the Texas Coastal Plain which slopes eastward toward the Gulf of Mexico. The Pliocene Goliad Formation which crops out in the area is mainly a sand and gravel unit with gravel usually occurring at or near the base of the formation. The Goliad Formation overlies unconformably the Miocene Catahoula Formation in the vicinity of the Bruni mine. The Bordas Escarpment, which lies approximately 8 km (5 mi) northwest of the mine site and rises to an elevation of about 305 m (1000 ft), marks the contact between the Goliad and Catahoula formations.

The ore zone strata at the Bruni mine belong to the Miocene Catahoula Formation. The Catahoula Formation was deposited on the subsiding margin of the Gulf Coast sedimentary basin (Galloway et al, 1977). The formation consists of an extensive deposit of sands and clays which dips about 1° to the south. The Catahoula Formation is approximately 244 m (800 ft) thick and is composed mainly of clay with relatively thin discontinuous clayey sand lenses in the area of the mine (WMC, 1977b). The formation contains fluvial channel-fill, crevasse splay, floodplain and lacustrine facies. Detailed discussion of the depositional systems, composition and structural development of the Catahoula Formation is presented by Galloway et al (1977).

The Oligocene Frio Clay underlies the Catahoula Formation in the vicinity of the mine. This unit is approximately 305 m (1000 ft) thick and is composed of relatively thick clay strata alternating with thinner localized sand or clayey-sand layers (WMC, 1977b).

A.8.3.2 Regional Ground-water Flow Systems

Generally, the hydrostratigraphy of the Catahoula Formation consists of sand and sandy-clay aquifers confined by clay and silty-clay aquitards. Water in the confined aquifers usually exists under artesian conditions. Lonsdale and Day (1937) reported that flowing wells existed in the Bruni area during the 1930's. Well yields and water quality from gravels and sands of the Goliad Formation are variable. In many places, shallow gravels contain saline water under unconfined conditions. WMC (1977b) reported that a number of wells in the Goliad Formation in Duval County, 16 to 24 km (10 to 15 miles) southeast of the Bruni mine, produced water at 63 L/sec (1000 gal/min) or more.

Ground-water movement in the Catahoula Formation generally is to the east and/or southeast toward the Gulf of Mexico. The principal controls on ground-water movement are the distribution of permeable sand strata and the proximity of the Gulf. Recharge occurs from precipitation in inland areas. Ground water moves downgradient toward the coast and discharges to the shallow unconfined aquifers in marsh and bay environments. The regional ground-water gradient in the vicinity of the mine, estimated from water level data for four wells, was reported to be 0.003 (WMC, 1977b). Detailed

discussions of the regional ground-water flow regimes of the Texas Coastal Plain are presented by Jones (1968) and Payne (1968).

A.8.4 Description and Analysis of Data Collected

A.8.4.1 Exploration

Exploration boreholes were drilled throughout the Bruni mine permit area to evaluate uranium deposits and the hydrostratigraphy of the area. Details of two such boreholes are reported by WMC (1977b). Test hole R1-4 was drilled in 1976 to a total depth of 151 m (497 ft). Circulated drill cuttings were collected and subsequently evaluated for lithologic properties. Geophysical logs recorded for the borehole included resistivity, spontaneous potential, natural gamma and caliper. The lithologic sample and geophysical logs for this well were compared to those recorded for a 171 m (501 ft) deep water supply well at the mine. Correlation of these logs indicated that some sandy horizons were present at each location, although their lateral continuity and clay content varied. Dense clay layers comprised approximately 46 m (150 ft) of the 58 m (190 ft) section between the production zone and an underlying aquifer.

WMC (1977b) reported that an effort was made to plug all open holes and repair leaking injection wells; however, numerous man-made connections may still exist between the production zone and the shallow strata.

A.8.4.2 Aquifer Tests

WMC conducted several aquifer tests in various production areas at the Bruni mine. Ten constant discharge pumping tests were conducted using two or more observation wells. In addition, one pumping test involved only one observation well. Table A.51 presents a summary of the aquifer tests conducted at the Bruni site.

A summary of aquifer and aquitard characteristics estimated by WMC and by the authors from the aquifer test data are presented in Table A.52. Drawdown data or graphical representations of test data are not available or are insufficient to allow analysis by the authors for tests other than numbers 6, 9, 10, and 11. Drawdown data for these four tests were analyzed by the authors using the Hantush and Jacob (1955) leaky artesian method and the Hantush (1960) modified method. Drawdown data curves were matched with reasonable certainty to the Hantush and Jacob (1955) solution for aquifer test 6. Drawdown data curves were matched with reasonable certainty to the type curves of the Hantush (1960) modified method for tests 9, 10, and 11. It should be noted, however, that potential partial penetration effects were not evaluated due to uncertainty with respect to the screened intervals of the tested wells. Values of horizontal hydraulic conductivity, transmissivity and storativity estimated by the authors for the tested aquifer are presented in Table A.52. Values of vertical hydraulic conductivity for the aquitard overlying the tested aquifer are presented also.

The Cooper and Jacob (1946) method was used by WMC to analyze all drawdown data. The method of Papadopulos (1965) was used to determine

Table A.51. Summary of aquifer tests conducted at the Bruni mine.

Test No.	Pumping/ Injection Well	Production Area	Aquifer	Total Depth of Tested Well m (ft)	Number of Observations Wells	Units Monitored	Type of Test[a]	Date of Test	Length of test (hrs)	Discharge Rate L/s (gal/min)
1		1	Production Sandstone		4	Production Sandstone	CD	Spring-Summer 1975	25	0.4 (6.4)
2		1	Production Sandstone		4	Production Sandstone	CD	Spring-Summer 1975	18	0.32 (5)
3	VI-10	2	Production Sandstone	55 (180)	4	Production Sandstone	CD	Spring-Summer 1975	18	0.4 (6.3)
4	VII-7	3	Production Sandstone		15	Production Sandstone, Shallow Gravel	CD	July 30, 1977	20	0.32 (5)
5	VII-7	3	Production Sandstone		1	Production Sandstone, Overlying Aquifer	CD	August 10, 1977	24	0.32 (5)
6	PW	4	Production Sandstone		18	Production Sandstone, Shallow Gravel, Underlying Aquifer	CD	January 14-15, 1978	24	0.52 (8.25)
7	22 Wells	5	Production Sandstone		13	Production Sandstone, Shallow Gravel	CD	October 31, 1978	12	7.0 (110)
8	RI-4	5	Production Sandstone		2	Production Sandstone, Overlying Aquifer	CD	November 30, 1978	3	0.32 (5)
9	41-H	6	Production Sandstone		10	Production Sandstone, Shallow Gravel, Underlying Aquifer	CD	August 20,1979	24	0.56 (8.9)
10	33-H	6	Production Sandstone	62.5 (205)	10	Production Sandstone, Shallow Gravel, Underlying Aquifer	CD	August 29, 1979	24	0.57 (9.1)
11	35-H	6	Production Sandstone	61.3 (201)	10	Production Sandstone, Shallow Gravel, Underlying Aquifer	CD	September 10, 1979	23.5	0.56 (8.9)

[a]CD = Constant Discharge.

Table A.52. Summary of aquifer and aquitard characteristics estimated by Wyoming Mineral Corporation and by the authors from aquifer test data for the Bruni mine.

Test No.	Pumping/ Injection Well	Aquifer	Analysis Conducted By[a]	Tested Aquifer[b]			Aquitard Overlying Tested Aquifer[b]	Method of Analysis[c]	Type of Test[d]
				Range of Horizontal Hydraulic Conductivity cm/s (ft/d)	Transmissivity m^2/d (ft^2/d)	Storativity	Range of Vertical Hydraulic Conductivity cm/s (ft/d)		
1		Production Sandstone	WMC	--	3.7 to 7.3 (39.8 to 78.5)	$2.2x10^{-4}$	--	P,CJ	CD
2		Production Sandstone	WMC	--	0.87 to 10.3 (9.4 to 110.8)	$4.7x10^{-5}$	--	P,CJ	CD
3	VI-10	Production Sandstone	WMC	--	0.87 to 8.8 (9.4 to 94.7)	$8.4x10^{-4}$	--	P,CJ	CD
4	VII-7	Production Sandstone	WMC	--	--	--	--	H,P,CJ	CD
5	VII-7	Production Sandstone	WMC	--	--	--	--	P,CJ	CD
6	PW	Production Sandstone	WMC	$3.3x10^{-4}$ to $4.2x10^{-4}$ (0.34 to 1.2)	3.5 to 4.5 (37.7 to 48.4)	$1.5x10^{-5}$	--	P,CJ,HJ	CD
			Authors	$8.7x10^{-5}$ to $1.0x10^{-4}$ (0.25 to 0.29)	2.3 to 2.4 (25 to 29)	$1.4x10^{-4}$	$8.7x10^{-6}$ to $1.4x10^{-5}$ (0.025 to 0.041)		
7	22 Wells	Production Sandstone	WMC	--	--	--	--	P,CJ	CD
8	R1-4	Production Sandstone	WMC	--	--	--	--	P,CJ	CD
9	41-H	Production Sandstone	WMC	--	9.8 to 12.9 (105.4 to 138.8)	$1.1x10^{-5}$ to $9.7x10^{-5}$	--	P,CJ,H	CD
			Authors	$3.1x10^{-4}$ to $6.0x10^{-4}$ (0.9 to 1.7)	6.7 to 12.6 (72 to 136)	$1.1x10^{-4}$ to $1.4x10^{-4}$	--		
10	33-H	Production Sandstone	WMC	--	11.7 to 23.5 (125.9 to 252.9)	$1.2x10^{-5}$ to $1.9x10^{-4}$	--	P,CJ,H	CD
			Authors	--	6.8 (73)	$1.1x10^{-4}$ to $1.3x10^{-4}$	--		
11	35-H	Production Sandstone	WMC	--	7.6 to 10.1 (81.8 to 108.7)	$6.2x10^{-5}$ to $1.2x10^{-4}$	--	P,CJ,H	CD
			Authors	$2.1x10^{-4}$ to $2.5x10^{-4}$ (0.6 to 0.7)	4.4 to 5.3 (47 to 57)	$6.1x10^{-5}$ to $8.6x10^{-5}$	--		

[a] WMC = Wyoming Mineral Corporation

[b] Values are based, in part, on the following assumptions: 1) the aquitard underlying the tested aquifer is impermeable, and 2) the specific storage of the overlying aquitard is two orders of magnitude larger than that calculated for the tested aquifer.

[c] P = Papadopulos (1965); CJ = Cooper and Jacob (1946); H = Hantush (1960); HJ = Hantush and Jacob (1955).

[d] CD = Constant Discharge

Source: WMC, 1977b, 1978b, 1979a, and 1979b.

directional transmissivities of the production aquifer. Methods of leaky aquifer analysis were not utilized by WMC to determine vertical hydraulic conductivity of confining units at the Bruni mine. Water levels in wells completed in overlying and underlying aquifers were observed during several of the aquifer tests with no significant fluctuations reported. WMC based their assertion that the confining units are essentially impermeable on these observations.

The horizontal hydraulic conductivity of the production sandstone is estimated to range from about 3.0×10^{-4} to 6×10^{-4} cm/s (0.85 to 1.7 ft/d), and storativity averages about 1.4×10^{-5}. The direction of hydraulic conductivity estimated by WMC from their aquifer tests is approximately N45°W, which is consistent with the major trend of the ore body. The production sandstone appears to be heterogeneous and anisotropic.

Results from the aquifer tests indicate that water may leak through the overlying and/or underlying aquitards under pumping conditions. Use of the Hantush and Jacob (1955) leaky artesian method by the authors allowed an estimate of vertical hydraulic conductivity of the overlying aquitard. The values of vertical hydraulic conductivity range from about 8.7×10^{-6} to 1.4×10^{-5} cm/s (0.2 to 0.4 ft/d). Specific storage of the aquitards cannot be estimated using the Hantush and Jacob (1955) method of analysis.

The pathways for leakage cannot be evaluated from results of the aquifer tests conducted at the Bruni mine, although reasonably certain matches of drawdown data from aquifer test 6 to the Hantush and Jacob (1955) type curves suggest that water may be leaking through aquitards overlying and/or underlying the production aquifer in Production Area 4. However, curve matches to the Hantush and Jacob (1955) type curves probably reflect leakage through abandoned boreholes.

A.8.5 Hydrogeology of the Mine Site

Uranium mineralization at the Bruni mine occurs as roll-front deposits in a 15 to 18 m (50 to 60 ft) thick clayey sand of the Catahoula Formation. The uranium minerals uraninite and coffinite are concentrated along a continuous northeast-southwest linear boundary between oxidized, altered sediment and sulfide-rich reduced materials. Molybdenum and selenium also are concentrated along these oxidation-reduction fronts (Galloway et al, 1977). The host strata are volcanic, plagioclase rich, medium- to fine-grained sandstones containing numerous mudstone lenses.

Geologic cross sections were constructed by WMC (1977b) based on electric and lithological logs. A typical cross section is shown in Fig. A.63. A generalized stratigraphic section is shown in Fig. A.64. Fig. A.65 is a surface geologic map of the Bruni area.

A.8.6 Monitor Well System and Well Construction

Monitoring for lateral excursions at the Bruni mine is accomplished utilizing numerous (over 120) production zone monitor wells located approximately 61 m (200 ft) from the production area perimeter and spaced

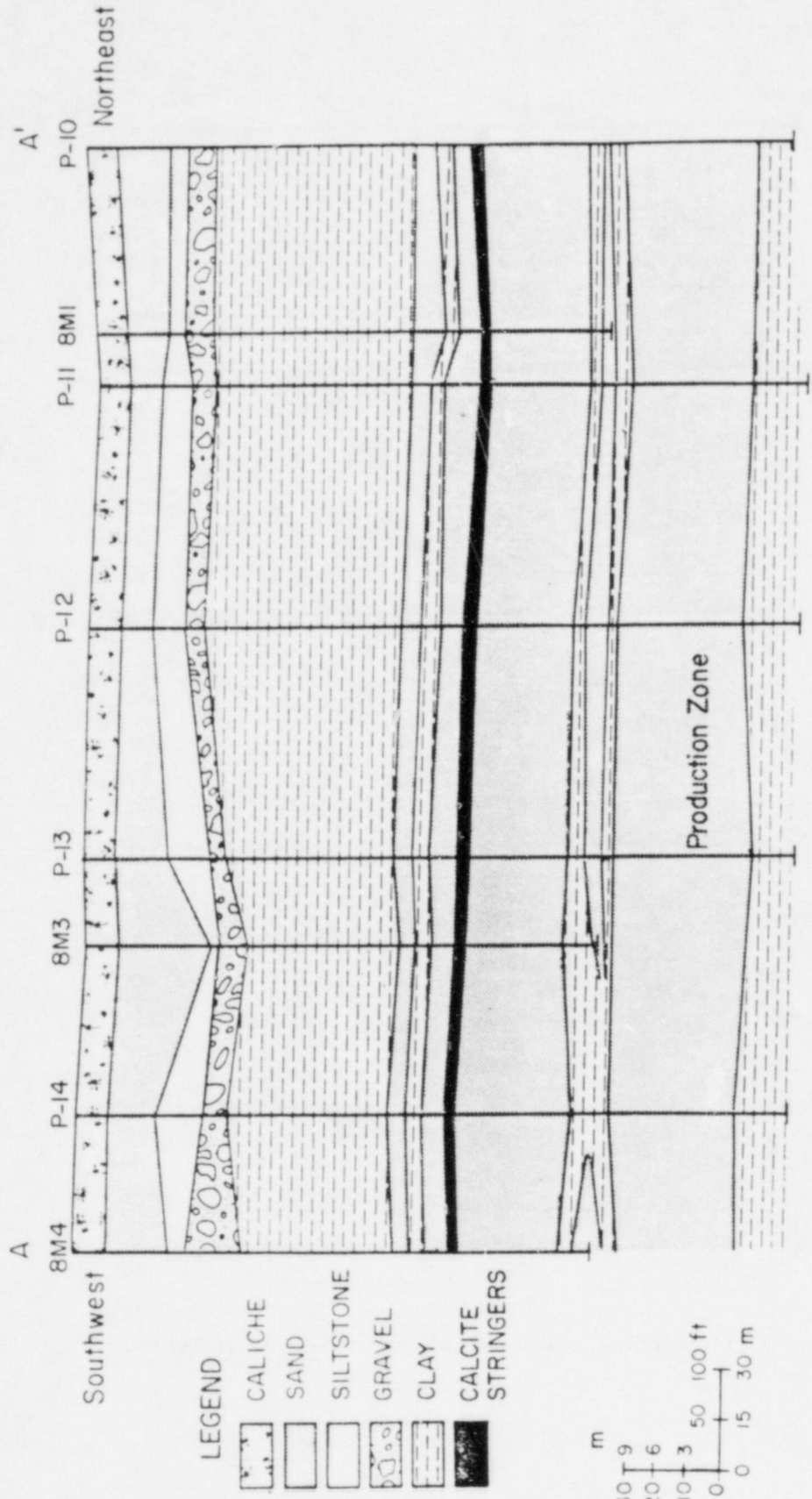


Fig. A.63. Simplified geologic cross section through the Bruni mine site. *Source:* WMC, 1979a.

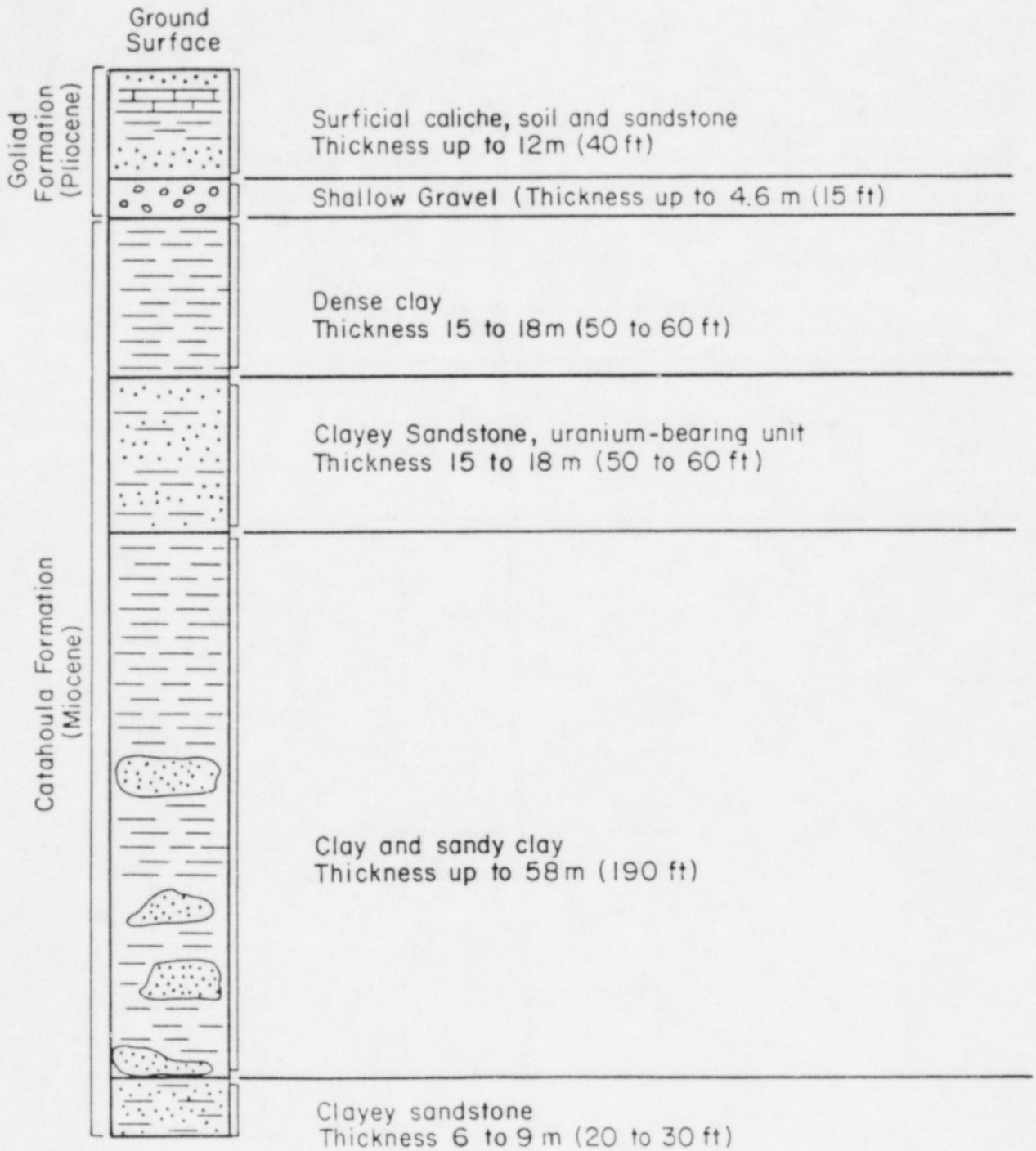


Fig. A.64. Generalized stratigraphic section for the Bruni project area.

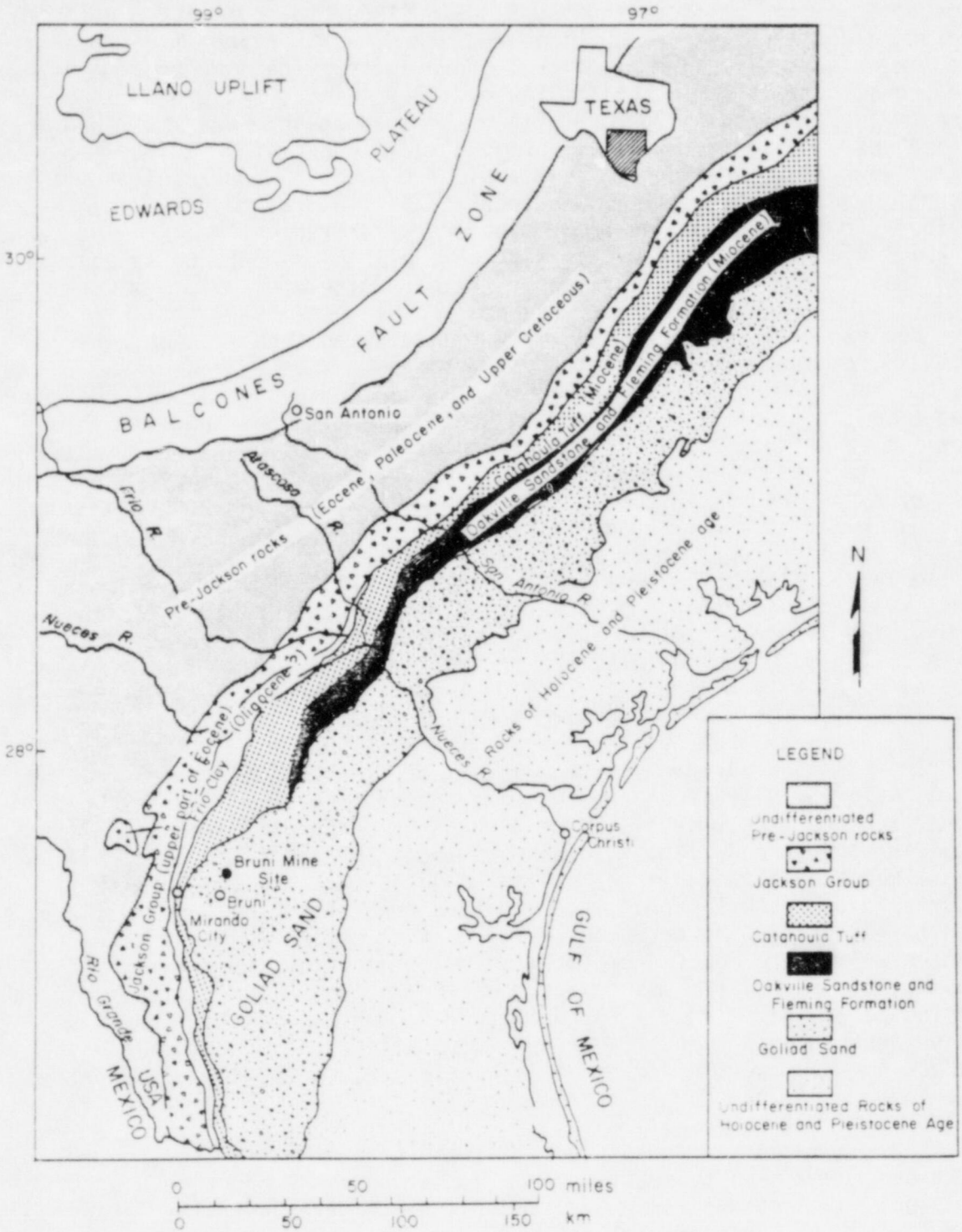


Fig. A.65. Surficial geologic map of the Bruni project area. *Source:* WMC, 1981f.

approximately 107 m (350 ft) apart. These wells are screened or perforated over the entire production zone thickness. Additionally, lines of production zone monitor wells extend about 305 m (1000 ft) southeastward (downgradient) from the production areas to detect horizontal excursions in the predominant direction of natural ground-water flow. Monitor wells have been constructed also in the shallow gravel unit to detect vertical excursions. These wells are located within the production areas and along their perimeters. No monitor wells are completed in underlying aquifers to detect downward migration of lixiviant. A production zone monitor well and a shallow gravel monitor well are located about 0.8 km (0.5 mile) in each of four directions from the production areas for regional monitoring (WMC, 1977b). Over 240 monitor wells exist at the Bruni mine (WMC, 1982b). Fig. A.66 shows the monitor wells used at the mine site.

Monitor wells were sampled every two weeks and the samples were analyzed for the excursion indicators ammonia, sulfate, uranium and conductivity. UCL's were set for these excursion indicators for each production area (i.e., a single UCL for uranium applied to all monitor wells in a specific production area). These limits were set following baseline water quality evaluations. Originally, baseline averages and ranges were established by one round of sampling from one well within the production area and four wells located around its perimeter (Table A.53). Baseline was established for Production Areas 5 and 6 by sampling each well in the production area. The UCL's were set as follows: 1) for uranium and ammonia, 5 mg/L was added to the highest baseline concentration; 2) for conductivity, the limit was set at 125% of the maximum baseline value; and 3) for sulfate, 150 mg/L was added to the maximum baseline concentration (Texas Department of Water Resources, 1979). Table A.54 presents the excursion control limits for each production area.

Injection, recovery, and production zone monitor wells were drilled as 17 cm (6.75 in) diameter boreholes to approximately 52 m (170 ft) below ground surface (the base of the production sandstone). Boreholes of the same diameter were drilled to depths of about 15.2 m (50 ft) for shallow gravel monitor wells. All wells were at least partially cased with 11.4 cm (4.5 in) diameter PVC pipe. Injection and recovery wells were completed as open holes, except where caving of the host sandstone required screens. Monitor wells were constructed as open holes or were completed with screens. Cement slurry was pumped down the inside of the casing and up the annulus to ground surface. Approximately 4.6 m (15 ft) of slurry was left inside the bottom of the casing; this was drilled out after hardening. Production wells were underreamed to create a 28 cm (11 in) diameter open hole in the ore zone.

After the shallow gravel contamination problem was discovered, various techniques were used to seal improperly constructed production wells. In addition, numerous shallow gravel monitor wells were installed. Fig. A.67 depicts several types of well construction used at the Bruni mine.

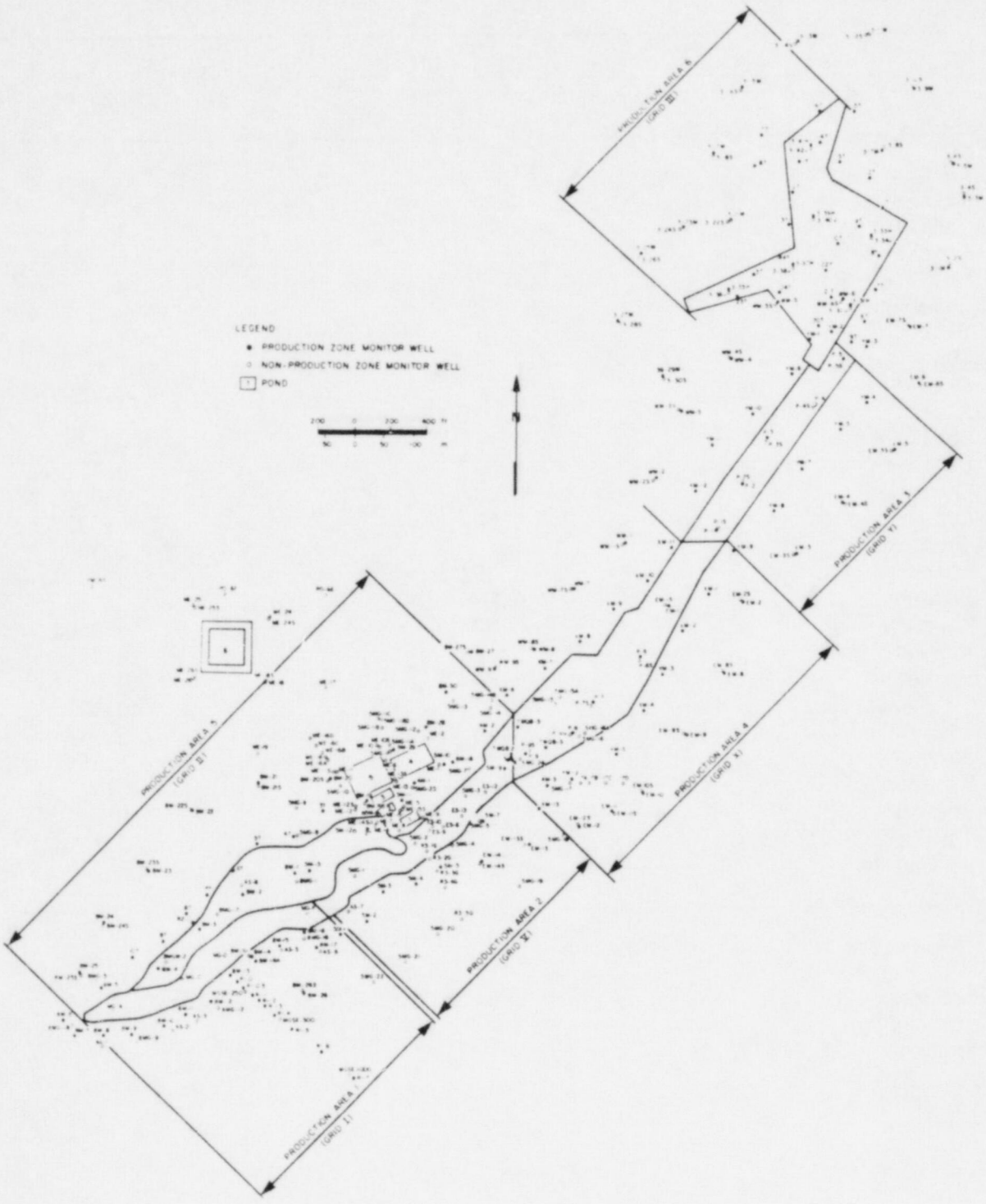


Fig. A.66. Location map of the monitor wells at the Bruni mine site. *Source:* WMC.

Table A.53. Average baseline water quality[a] for the Bruni mine.

Parameter	Production Zone Average Baseline[b]	Shallow Gravel Average Baseline[b]
Calcium	267	799
Magnesium	68	297
Sodium	413	879
Bicarbonate	175	164
Sulfate	142	588
Chloride	1090	2696
Nitrate	2.2	10.5
TDS	2312	5682
Ammonia	1	<0.64
Arsenic	0.02	<0.006
Barium	0.15	0.134
Cadmium	0.007	<0.007
Chromium	0.002	0.009
Copper	0.007	<0.021
Iron	1.2	1.95
Lead	0.052	<0.023
Manganese	0.086	0.14
Mercury	0.001	<0.0004
Molybdenum	0.42	<0.05
Nickel	0.34	<0.01
Selenium	0.05	<0.021
Zinc	0.42	0.13
Boron	1.2	4.35
Vanadium	0.05	<0.07
Uranium	0.33	<0.05
Radium-226 (pCi/L)	129	3.3
pH (units)	7.5	7.31

[a]Units are mg/L except as noted.

[b]Average baseline values obtained from Production Areas 1 and 2.

Source: WMC, 1979a.

Table A.54. Excursion upper control limits[a] for the Bruni mine.

Parameter	Production Area									
	1 and 2		3		4		5		6	
	Ore Zone	Shallow Zone	Ore Zone	Shallow Zone	Ore Zone	Shallow Zone	Ore Zone	Shallow Zone	Ore Zone	Shallow Zone
Ammonia	6	7	5	5	5	6	8	5.2	6	6
Uranium	6	6	6	5	11	5	8.66	5.69	5.5	5.5
Sulfate	300	900	125	265	403	725	890	1480	N/A	N/A
Conductivity (µmhos)	5000	12,000	3750	7400	4700	8600	8831	12,125	5238	6309

[a]Units are mg/L except as noted.

Source: WMC, 1982a and Texas Department of Water Resources 1977, 1979, and 1980.

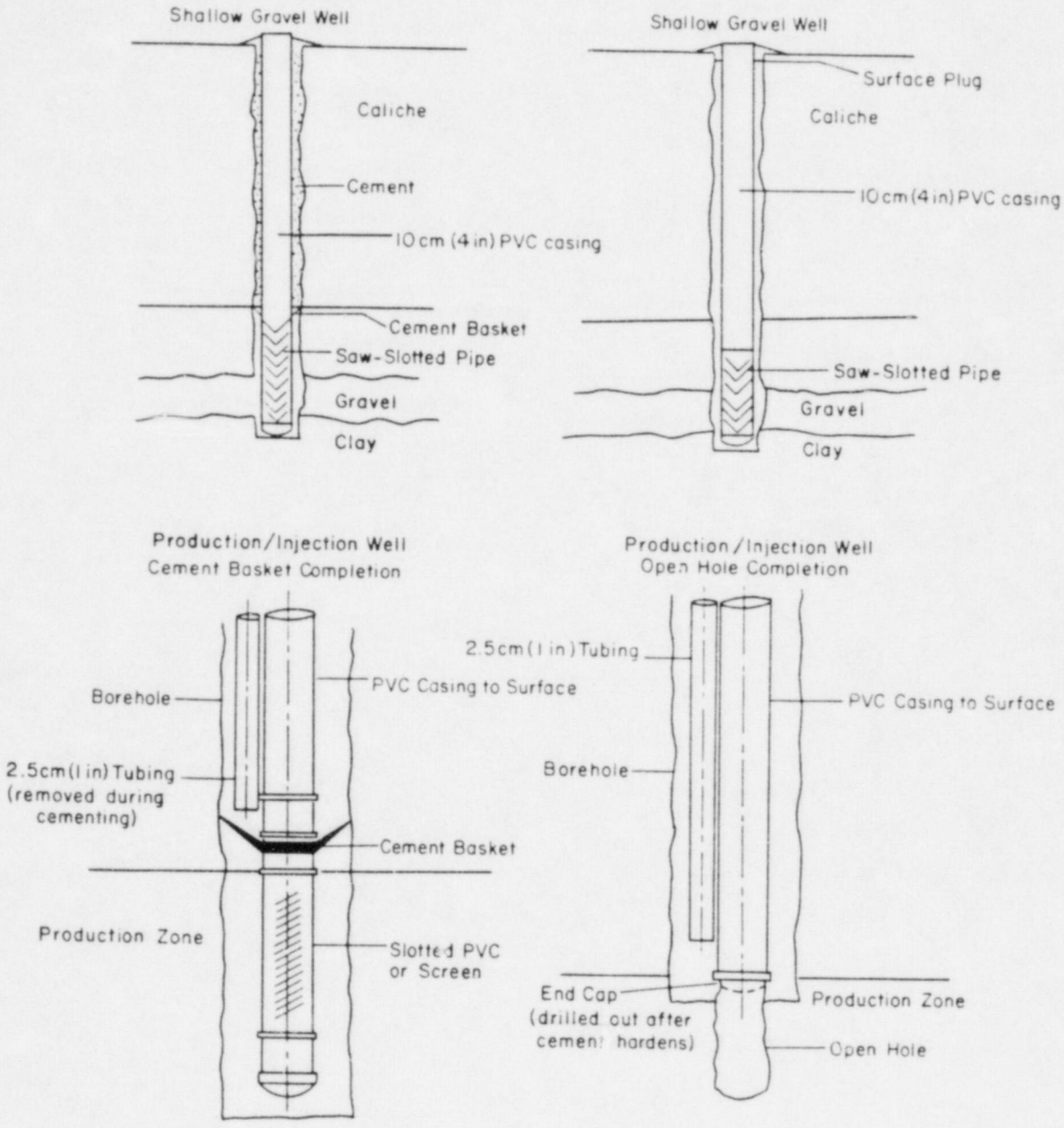


Fig. A.67. Typical well completion methods for the Bruni mine. *Source:* WMC, 1979a.

A.8.7 Analysis of Excursions

WMC's Bruni mine has a history of numerous excursions. Most excursions have been detected in the shallow gravel unit. Lixiviant migration into the shallow gravel was noticed shortly after mining commenced in 1976. On April 24, 1978, the state of Texas fined WMC for excessive permit violations. Mining was allowed to continue, although changes were made in the mining permit pertaining to reporting requirements. Additionally, WMC was ordered to restore the shallow gravel aquifer.

Due to the large number of excursions (an undetermined number prior to 1978 and sixty company-reported excursions that occurred between April, 1978, and October, 1982) and the lack of adequate documentation of these events, the authors are precluded from conducting a detailed analysis of excursions at the Bruni mine. Furthermore, it is not possible to determine which shallow gravel monitor wells were contaminated by leakage from an evaporation pond and which were contaminated by excursions. Table A.55 presents a summary of the excursion information contained in Texas Department of Water Resources files. Fig. A.68 shows the areas where the shallow gravel aquifer has been affected by either migration of lixiviant from the production zone or leakage from the evaporation pond.

There have been numerous horizontal excursions within the production zone at the Bruni mine. Very few details pertaining to these excursions are available. References in Texas Department of Water Resources files to WMC's ability (or inability) to control these production zone excursions by altering production/injection rates could not be substantiated.

WMC's corrective actions for the shallow zone contamination problem included attempts to contain the contaminated area by injecting fresh water into wells that encircled the affected areas while simultaneously withdrawing contaminated water from the center of the affected areas. This plan had limited success, and numerous shallow zone wells remained on excursion status when the facility was placed on standby in 1982.

It appears that unplugged exploration boreholes and improperly constructed injection/production wells created preferential pathways for vertical lixiviant migration from the production zone. This finding reinforces the importance of locating and plugging exploration boreholes and thoroughly testing well integrity to prevent or minimize excursions.

A.8.8 Restoration

WMC conducted field restoration tests on a small-scale in 1977 at the Bruni Mine. Ground-water sweeping, clean water recycling and chemical restoration techniques were tested (Resource Engineering and Development, Inc., 1980). A two-well test pattern was leached with ammonium carbonate/bicarbonate lixiviant for 29 days. The wells were pumped subsequently at about 0.1 L/s (1.5 gal/min) to effect a ground-water sweep of the affected aquifer. Approximately 302,800 L (80,000 gal) of water were withdrawn from the production aquifer within the test pattern. Following the ground-water

Table A.55. Summary of excursions at the Bruni mine during mining operations.

Well	Location[a]	Aquifer	Date of Sample[b]	Excursion Parameters[c]			Excursion History[d]	
				Conductivity (μmhos/cm)	Ammonium (mg/L)	Uranium (mg/L)	On	Off
MGD	PA1	Shallow Gravel	2/27/76	--	X	X	2/27/76	
MG8-1	PA1	Shallow Gravel	3/26/76	--	--	--	3/26/76	
M8-3	PA1	Shallow Gravel	4/16/76	X	--	--	4/16/76	
M8-14	PA1	Shallow Gravel	4/16/76	X	--	--	4/16/76	
8M-1	PA1	Production Sandstone	9/26/76	--	X	--	9/26/76	
MS8-1	PA1	Shallow Gravel	11/05/76	X	--	--	11/05/76	
8M-8	PA5	Production Sandstone	12/30/76	X	--	--	12/30/76	
8M-14	PA1	Production Sandstone	1/21/77	--	--	--	1/21/77	
MG8-17	PA1	Shallow Gravel	3/10/77	X	--	--	3/10/77	
MG8-1	PA1	Shallow Gravel	3/12/77	--	--	--	3/12/77	
MG8-16	PA1	Shallow Gravel	3/12/77	--	--	--	3/12/77	
5MG-2	PA2	Shallow Gravel	7/06/77	X	--	--	7/06/77	
5M-8	PA2	Production Sandstone	7/08/77	--	--	--	7/08/77	
8MG-12	PA2	Shallow Gravel	7/14/77	X	--	--	7/14/77	
8M-5	PA1	Production Sandstone	8/11/77	--	--	--	8/11/77	8/18/77
8M-14	PA1	Production Sandstone	8/11/77	--	--	--	8/11/77	
5MG-2	PA2	Shallow Gravel	8/11/77	--	--	--	8/11/77	9/06/77
5MG-3	PA2	Shallow Gravel	8/18/77	--	--	--	8/18/77	9/06/77
MGSE-250	PA1	Shallow Gravel	8/19/77	--	--	--	8/19/77	
8M-4	PA1	Production Sandstone	9/07/77	X	--	--	9/07/77	
5M-3	PA5	Production Sandstone	9/12/77	--	--	--	9/12/77	10/12/77
5MG-11	PA5	Shallow Gravel	10/11/77	X	--	--	10/11/77	11/07/77
5MG-14	PA2	Shallow Gravel	10/11/77	X	--	--	10/11/77	11/07/77
5MG-15	PA2	Shallow Gravel	10/11/77	X	--	--	10/11/77	11/07/77
5MG-16	PA2	Shallow Gravel	10/11/77	X	--	--	10/11/77	11/07/77
5MG-17	PA2	Shallow Gravel	UCL	12,000	--	--		
			10/11/77	19,500	--	--	10/11/77	8/30/78
5MG-14B	PA2	Shallow Gravel	10/26/77	--	--	--	10/26/77	
5M-3	PA5	Production Sandstone	10/28/77	--	--	--	10/28/77	
5MG-17A	PA2	Shallow Gravel	10/26/77	--	--	--	10/26/77	
5M-13	PA5	Production Sandstone	10/28/77	--	--	--	10/28/77	
5MG-11	PA5	Shallow Gravel	UCL	12,125	--	--		
			11/17/77	13,500	--	--	11/18/77	6/05/78
8M-3	PA1	Production Sandstone	UCL	6,380	--	--		
			11/28/77	6,840	--	--	11/29/77	
8M-4A	PA1	Production Sandstone	12/02/77	X	--	--	12/02/77	
8M-3	PA1	Production Sandstone	12/02/77	X	--	--	12/02/77	
5MG-17A	PA2	Production Sandstone	12/02/77	X	--	--	12/02/77	
5MG-11A	PA5	Shallow Gravel	12/05/77	--	--	--	12/05/77	
EM-6	PA3	Production Sandstone	1/03/78	--	--	--	1/03/78	
P-5S	PA3	Shallow Gravel	3/23/78	--	X	--	3/23/78	6/15/78
P-7S	PA4	Shallow Gravel	5/25/78	--	X	--	5/25/78	
P-1S	PA3	Shallow Gravel	UCL	7,400	--	--		
			6/13/78	10,990	--	--	6/13/78	

Table A.55. (Continued)

Well	Location[a]	Aquifer	Date of Sample[b]	Excursion Parameters[c]			Excursion History[d]	
				Conductivity (µmhos/cm)	Ammonium (mg/L)	Uranium (mg/L)	On	Off
P-7S	PA4	Shallow Gravel	6/05/79	--	X	--	6/05/79	
P-7S	PA4	Shallow Gravel	6/26/79	--	X	--	6/26/79	
P-1S	PA3	Shallow Gravel	6/26/79	--	--	--	6/26/79	
P-5S	PA3	Shallow Gravel	6/26/79	--	--	--	6/26/79	
P-7S	PA4	Shallow Gravel	11/27/79	X	--	--	11/27/79	
P-8S	PA4	Shallow Gravel	UCL	8,600	--	--		
			11/07/79	11,500	--	--	11/07/79	
ME-16			5/12/80	X	--	--	5/12/80	
EM-1S	PA4	Shallow Gravel	7/22/80	--	X	--	7/24/80	
5MG-11	PA5	Shallow Gravel	UCL	12,125	--	--		
			8/14/80	14,100	--	--	8/14/80	10/09/80
EM-12S	PA5	Shallow Gravel	UCL	12,125	--	--		
			11/11/80	16,000	--	--	11/13/80	
P-6S	PA4	Shallow Gravel	1/22/81	--	X	--	1/22/81	
P-7S	PA4	Shallow Gravel	UCL	8,600	6	--		
			3/04/81	11,800	382	--	3/04/81	
P-8S	PA4	Shallow Gravel	UCL	8,600	--	--		
			3/04/81	9,080	--	--	3/04/81	
WM-8S	PA4	Shallow Gravel	UCL	8,600	6	--		
			4/22/81	10,990	14	--	4/22/81	
P-6S	PA4	Shallow Gravel	UCL	8,600	6	--		
			4/30/81	8,640	26-40	--	4/30/81	
P-7S	PA4	Shallow Gravel	UCL	8,600	6	--		
			4/30/81	11,900	213	--	4/30/81	
P-8S	PA4	Shallow Gravel	UCL	8,600				
			4/30/81	9,200	--	--	4/30/81	
EM-1S	PA4	Shallow Gravel	UCL	8,600	6	--		
			4/30/81	8,750	47	--	4/30/81	
WM-8S	PA4	Shallow Gravel	7/17/81	--	X	--	7/17/81	
5MG-11	PA5	Shallow Gravel	7/29/81	X	--	--	7/29/81	
5MG-12	PA5	Shallow Gravel	8/06/81	X	--	--	8/06/81	
5MG-18	PA5	Shallow Gravel	UCL	12,125	--	--		
			11/19/81	13,900	--	--	11/20/81	2/11/82

[a]PA = Production Area.

[b]UCL = Upper Control Limits.

[c]Analyses presented where available; X = Analyses unavailable but excursion parameter indicated.

[d]Duration of excursions uncertain, however, at least nine excursions that occurred between 2/76 and 4/78 lasted for over 90 days.

Source: Texas Department of Water Resources files.

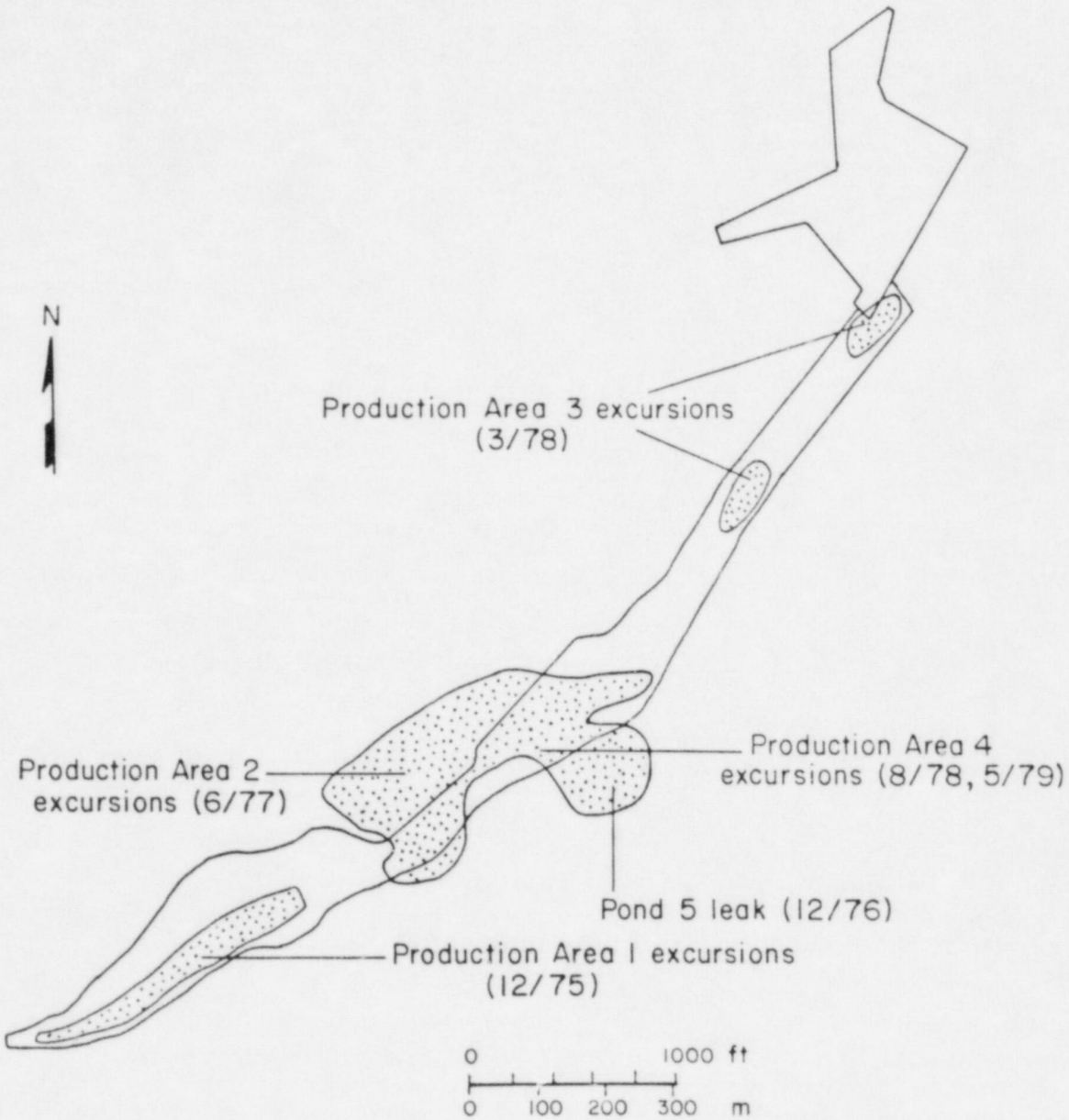


Fig. A.68. Location and dates of the shallow zone contamination at the Bruni mine. *Source:* WMC, 1981f.

sweep, reverse osmosis was utilized to treat the produced water which was then reinjected into the production aquifer.

The TDS concentration was reduced to acceptable levels at the termination of the clean water recycling phase of this test; however, the ammonium concentration remained at approximately 90 mg/L. Chemical treatment of the aquifer was initiated subsequently in an attempt to reduce the ammonium concentration further. A solution high in calcium and magnesium ions was circulated through the aquifer to effect exchange of ammonium ions from the clay-mineral portion of the aquifer matrix. Ammonium ions were stripped from produced solutions in an above-ground processing plant.

The results of this restoration test are summarized in Table A.56. WMC stated the following conclusions: 1) ground-water sweeping and clean water recycling were not effective in reducing concentrations of uranium and ammonium to baseline levels; 2) chemical restoration techniques were required to reduce ammonium and uranium to near-baseline levels; and 3) the high clay content of the production aquifer (15% to 25%) made removal of ammonium difficult.

A large-scale restoration demonstration project was undertaken in Well Field AB, an area of approximately 0.4 ha (0.95 acre) within Production Area 3. Although most details concerning this effort are unavailable, a brief account of the project follows.

Mining activities in Well Field AB ceased in April, 1980. Restoration commenced on April 29, 1980, when lixiviant was transferred from the well field to a new mining unit and clean water was injected into Well Field AB. Approximately 0.8 pore volumes of water were removed from the production aquifer. Conductivity was reduced from about 10,000 to 6,290 umhos/cm by this process. A ground-water sweep was conducted from May 9 to June 25, 1980. A reverse osmosis circuit was utilized subsequently to reduce the concentration of dissolved constituents remaining in the ground water. Treated water was reinjected into the production aquifer. By mid-October, 1980, approximately ten pore volumes of water had been treated, and conductivity was reduced to about 3,800 umhos/cm (baseline was about 2,600 umhos/cm). About 15 pore volumes were circulated through the reverse osmosis unit from October, 1980, to March 24, 1981, when restoration was halted. WMC concluded that the restoration attempt had been very successful in reducing all constituents, except uranium and ammonium, to acceptable levels. Table A.57 presents the available data on the Well Field AB restoration effort.

Several small-scale restoration tests are scheduled during the second half of 1984 at the Bruni mine (Texas Department of Water Resources, 1984). Four separate five-spot patterns within Production Area 4 will be utilized for these tests. In three of the patterns, ground water will be pumped from the production aquifer at about 0.13 L/s (2 gpm) and treated by reverse osmosis. Prior to reinjection, the treated water will be fortified with sodium and/or calcium chloride to induce cation exchange between ammonium on clays in the aquifer matrix and sodium and calcium ions in solution.

Table A.56. Summary of the 1977 restoration test at the Bruni Mine.[a]

Parameter	Average Baseline	Post-Leach	Post-Ground Water Sweep	Post-Clean Water Recycle	Post-Chemical Restoration
Carbonate	205	3640	600	191	47
Sodium	74	-	730	155	160
Ammonium	1	497	300	91	17
Chloride	622	1460	600	494	500
Calcium	105	-	80	27	127
Magnesium	23	-	-	10	-
Uranium (U_3O_8)	0.5	-	4	9	0.5
Radium-226 (pCi/L)	191	-	-	-	-
pH (units)	7.5	-	-	8.2	-
Conductivity (μmhos/cm)	2483	16,500	4500	2542	-

[a]Units are mg/L except as noted.

Source: Resource Engineering and Development, 1980.

Table A.57. Water quality[a] following the restoration demonstration project in Well Field AB at the Bruni mine.

Parameter	Average[b] Post-Mining 5/15/80	6/24/80[b]	9/22/80[b]	3/24/80[b]	11/5/81[b]	TDWR[c] Restoration Limit
Calcium	122.2	67.7	57.0	67.0	73.0	200.0
Magnesium	87.7	80.6	39.2	27.0	30.0	150.0
Sodium	454.5	446.0	265.0	152.0	205.0	465.0
Potassium	49.2	40.9	30.1	20.0	33.0	18.0
Bicarbonate	965.0	1430.0	837.0	412.0	488.0	500.0
Sulfate	495.0	480.0	252.0	108.0	138.0	125.0
Chloride	1342.0	1340.0	677.0	405.0	490.0	680.0
Fluoride	0.2	0.2	0.3	0.31	0.89	1.8
Nitrate	59.4	15.3	15.0	5.0	1.0	12.0
Ammonia	506.0	N/A	288.0	119.0	148.0	0.5
Arsenic	0.011	0.028	0.04	0.037	0.026	0.5
Barium	0.12	<0.10	0.06	0.19	0.09	1.0
Boron	0.7	0.17	0.07	0.78	0.72	4.0
Cadmium	0.003	<0.002	0.004	0.0001	<0.0001	0.01
Chromium	0.005	<0.005	0.005	0.005	0.001	0.05
Copper	0.030	0.017	<0.005	0.018	0.044	1.0
Iron	0.22	0.59	0.14	0.68	0.45	0.3
Lead	0.01	<0.005	0.040	0.004	0.002	0.05
Manganese	0.084	0.59	0.026	0.048	0.050	0.17
Mercury	0.0002	<0.0002	<0.0002	0.0053	0.0008	0.002
Molybdenum	1.90	2.10	1.15	0.70	0.71	1.0
Nickel	0.02	<0.01	0.01	<0.01	<0.01	1.0
Selenium	0.023	0.018	0.011	0.010	0.008	0.05
Silica	25.7	24.0	18.1	21.0	27.0	15.0
Silver	0.002	<0.002	<0.002	<0.01	<0.01	0.05
Uranium	17.5	22.0	11.0	4.75	4.24	2.0
Vanadium	0.005	<0.05	<0.05	<0.01	<0.01	5.0
Zinc	0.03	<0.01	<0.01	0.028	0.039	5.0
TDS	2270.0	1994.0	1187.0	943.0	1190.0	1590.0
Conductivity (µmhos/cm)	7190.0	7535.0	4398.0	2150.0	2790.0	2620.0
pH (units)	7.32	7.5	7.3	6.97	7.2	6-9

[a]Units are mg/L except as noted.

[b]Analyses of composite samples from all wells in the Well Field AB restoration project.

[c]TDWR = Texas Department of Water Resources.

Source: WMC, 1982b.

A multiple-phase restoration test will be conducted in the fourth five-spot pattern. The first phase of the test will involve pumping ground water from the production aquifer, storing the water in an evaporation pond where the pH will be raised to promote degassing of volatile ammonium, and reinjecting the water after the pH has been lowered. In the second phase of the test, injected solution will be at a higher pH to suppress mobilization of heavy metals within the aquifer. The final phase of the test will involve reverse osmosis treatment to reduce the TDS concentration.

The Bruni mine currently is in a standby mode due to economic considerations. Twenty-four wells throughout the mine area are pumped at a combined rate of approximately 1.6 L/s (25 gal/min) to contain mining solutions within the area. This in effect creates a slight ground-water sweep and should therefore have some beneficial effect on water quality over the long term. If WMC does not resume mining at Bruni, the production areas will have to be restored to baseline conditions according to the Texas Department of Water Resources mining permit.

APPENDIX A REFERENCES

Agarwal, R. G., Al-Hussainy, R., and Ramey, H. J., 1970. An Investigation of Wellbore Storage and Skin Effect in Unsteady Liquid Flow; I. Analytical Treatment, Soc. Pet. Eng. Jour. Trans. AIME 249, p. 279-290.

Boulton, N. S., 1963. Analysis of Data from Nonequilibrium Pumping Tests Allowing for Delayed Yield from Storage, Proceedings Institute of Civil Engineers.

Buma, G., Johnson, P. H., Branck, G. K., Watson, C. G., Noyes, H. J., and Capuano, R., 1980. Analysis of Groundwater Criteria and Recent Restoration Attempts After In Situ Leaching, Final Draft for U.S. Dept. of Interior, Bureau of Mines, Contract No. J0295019, with Resource Engineering and Development, Inc.

CCIC, 1981. U.S. Nuclear Regulatory Commission Source Material License Composite Application, SUA-1352, Docket No. 040-08714, Cleveland Cliffs Iron Company.

CCIC, 1982. Groundwater Restoration Report, Collins Draw In Situ Uranium Mining, Cleveland Cliffs Iron Company, May 21, 1982.

Chow, V. T., 1952. On the Determination of Transmissivity and Storage Coefficients from Pumping Test Data, Trans. Am. Geophys. Union, Vol. 33, p. 397-404.

Cooper, H. H., Jr., Bredehoeft, J. D., and Papadopulos, I. S., 1967. Response of a Finite-Diameter Well to an Instantaneous Charge of Water, Water-Resources Research, 3, p. 263-269.

Cooper, H. H., Jr. and Jacob, C. E., 1946. A Generalized Graphical Method for Evaluating Formation Constants and Summarizing Well Field History, Trans. Am. Geophys. Union, Vol. 27, No. 4, p. 526-534.

Curry, W. H., III, 1976. Late Cretaceous Teapot Delta of Southern Powder River Basin, Wyoming, 28th Annual Field Conference, Wyoming Geol. Assoc. Guidebook.

D'Appolonia Consulting Engineers, Inc., 1976. Determination of Hydrologic Conditions and Groundwater Chemistry, Test Site for In-Situ Leaching of Uranium, Nine Mile Lake Area, Wyoming, Unpublished Report. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

D'Appolonia Consulting Engineers, Inc., 1980. Hydrostratigraphy Study in Wyoming Mineral Corporation Report on Irigaray Mine Shallow Zone Evaluation, Vol. I, p. III-1 to III-57. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

Denson, N. M. and Pipiringos, G. N., 1974. Miscellaneous Investigations Map I-835, U.S. Geol. Survey.

DOI, 1974. Earth Manual, Bureau of Reclamation, U.S. Dept. of Interior.

EMC, 1978a. Application for Ammendment to Source Material License SUA-1064, Exxon Minerals Company, January 19, 1978. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1978b. Appendix I to Request for Modification to Mining Permit 218C, Uranium Solution Mine Test Program, Hydrological Test Data (Revised) Submitted to Wyoming Department of Environmental Quality, October 19, 1978. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1979a. Report for the Fourth Quarter 1978, Highland Solution Mine R and D Project, February 14, 1979. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1979b. Letter to Dr. Ray Cooperstein, U.S. Nuclear Regulatory Commission from Marvin Freeman, Exxon Minerals Company, April 19, 1979. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1979c. Letter to Dr. Ray Cooperstein, U.S. Nuclear Regulatory Commission, from Marvin Freeman, Exxon Minerals Company, March 21, 1979. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1980a. Letter to John J. Linehan, U.S. Nuclear Regulatory Commission, from H. H. Trussel, Jr., Exxon Minerals Company, October 16, 1980. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1980b. Application for Amendment to Mining Permit 218C, Submitted to Wyoming Department of Environmental Quality, March 27, 1980. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1981. Renewal Application for Source Material License SUA-1064, Highland Uranium Solution Mining R and D Program, Submitted to U.S. Nuclear Regulatory Commission, July 28, 1981. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1982a. Letter to Ken L. Kalman, U.S. Nuclear Regulatory Commission, from H. Paul Estey, Exxon Minerals Company, March 16, 1982. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1982b. Letter to John J. Linehan, U.S. Nuclear Regulatory Commission, from James Gilchrist, Exxon Minerals Company, September 29, 1982. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

EMC, 1984. Quarterly Report for Fourth Quarter 1983, Submitted to U.S. Nuclear Regulatory Commission, Docket No. 40-8064, January 25, 1984. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

Exxon Production Research Company, 1982. Highland Uranium Tailings Impoundment Seepage Study. Unpublished Report, February, 1982.

Ferris, J. G., Knowles, D. B., Browne, R. H., and Stallman, R. W., 1962. Theory of Aquifer Tests, U.S. Geol. Survey, Water Supply Paper 1536-E.

Galloway, W. E., Murphy, T. D., Belcher, R. C., Johnson, B. D., and Sutton S., 1977. Catahoula Formation of the Texas Coastal Plain: Depositional Systems, Composition, Structural Development, Ground Water Flow History and Uranium Distribution, Report of Investigations No. 87, Bureau of Economic Geology, University of Texas at Austin.

Hantush, M. S., 1960. Modifications of the Theory of Leaky Aquifers, J. Geophys. Res., Vol. 65, No. 11, p. 3713-3725.

Hantush, M. S., 1966. Analysis of Data from Pumping Tests in Anisotropic Aquifers, J. Geophys. Res., Vol. 71, No. 2, p. 421-426.

Hantush, M. S. and Jacob, C. E., 1955. Nonsteady Radial Flow in an Infinite Leaky Aquifer, Trans. Am. Geophys. Union, Vol. 36, No. 1, p. 93-100.

Hodson, W. G., Pearl, R. H., and Druse, S. A., 1973. Water Resources of the Powder River Basin and Adjacent Areas, Northeastern Wyoming, U.S. Geol. Surv. Hydrol. Inv. Atlas HA-465.

Humble Oil and Refining Company, 1970. Applicant's Environmental Report, Proposed Uranium In Situ Mining Test, Converse County, Wyoming, September, 1970. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

Hydro-Engineering, Inc. 1982. Ore-sand Aquifer Test in Bison Basin Mining Unit 2, Unpublished Report to Ogle Petroleum. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

Hydrology Associates, Inc., 1979. Irigaray Mine Excursion Potential, Unpublished Report to Wyoming Mineral Corporation.

Hydrology Associates, Inc., 1980. Baseline Hydrology Report, Irigaray Mine, Units Six and Seven, Unpublished Report to Wyoming Mineral Corporation.

In-Situ Consulting, Inc., 1978a. Hydrologic Evaluation of the Nine Mile Lake Property for In-situ Uranium Recovery, Unpublished Report, Laramie, Wyoming. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

In-Situ Consulting, Inc., 1978b. Hydrologic Analysis, Reno Ranch Site, Wyoming.

In-Situ Consulting, Inc., 1978c. Hydrologic Evaluation of the Collins Draw Research and Development Site.

Jones, P. H., 1968. Hydrology of Neogene Deposits in the Northern Gulf of Mexico Basin, U.S. Geol. Survey Open File Report.

Kasper, D. R., Martin, H. W., Munsey, L. D., Bhappu, R. B., and Chase, C. K., 1979. Environmental Assessment of In Situ Mining, Final Report by PRC Toups and Mountain States Research and Development under Contract No. J0265002 for U.S. Dept. of Interior, Bureau of Mines.

Langen, R. E. and Kidwell, A. L., 1974. Geology and Geochemistry of the Highland Uranium Deposits, Converse County, Wyoming, September, 1970.

Lonsdale, J. T. and Day, J. R., 1937. Geology and Ground Water Resources of Webb County, Texas, U.S. Geol. Survey, WSP 778.

Neuman, S. P., 1975. Analysis of Pumping Test Data from Anisotropic Unconfined Aquifers Considering Delayed Gravity Response, Water-Resources Research, Vol. 11, No. 2, p. 329-342.

Neuman, S. P. and Witherspoon, P. A., 1972. Field Determination of the Hydraulic Properties of Leaky Multiple Aquifer Systems, Water-Resources Research, Vol. 8, No. 5, pp. 1284-1298.

NRC, 1978a. Final Environmental Statement Related to Operation of Irigaray Uranium Solution Mining Project, Wyoming Mineral Corporation, Docket No. 40-8502, NUREG-0481. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado

NRC, 1978b. Final Environmental Statement Related to the Operation of the Highland Uranium Solution Mining Project, Exxon Minerals Company, Docket No. 40-8102, NUREG-0489, November 1978. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1980. Draft Environmental Statement, Nine Mile Lake Project, Rocky Mountain Energy Company, Docket No. 40-8380. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1981a. Final Environmental Statement Related to the Operation of Bison Basin Project, Ogle Petroleum, Inc., Docket No. 40-8745, NUREG-0687. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1981b. Memo from J. A. Pohle, U.S. Nuclear Regulatory Commission, to R. A. Scarano, U.S. Nuclear Regulatory Commission, June 10, 1981.

NRC, 1981c. Memo from J. A. Pohle, U.S. Nuclear Regulatory Commission, to R. A. Scarano, U.S. Nuclear Regulatory Commission, March 11, 1981.

NRC, 1981d. Groundwater Monitoring at Uranium In Situ Solution Mines, Staff Technical Position Paper WM-8102. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1982a. Amendment No. 12 to Source Material License SUA-1396, Ogle Petroleum, Inc., Bison Basin Project, Docket No. 40-8745. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1982b. Environmental Impact Appraisal by the Division of Waste Management in Consideration of the Renewal of Source Material License SUA-1064 for Exxon Minerals' R and D Highland Uranium In Situ Solution Mine, June, 1982. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1982c. Draft Environmental Statement Related to the Operation of the Teton Project, Teton Exploration Drilling, Inc., Docket No. 40-8781, NUREG-0925 (Draft). Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

NRC, 1982d. Memo from K. Kalman, U.S. Nuclear Regulatory Commission, to R. A. Scarano, U.S. Nuclear Regulatory Commission, June 1, 1982.

NRC, 1983a. Final Environment Statement Related to the Operation of the Teton Uranium ISL Project, NUREG-0925, Docket No. 40-8781, August, 1983.

NRC, 1983b. Draft Memorandum for Docket File No. 40-8714, from F.W. Ross, November 21, 1983.

OPI, 1979. Application for Permit to Mine for the Bison Basin Project In-Situ Uranium Mine, Docket No. 40-8745 (Revised 1980). Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

OPI, 1980. Final Restoration Report, Bison Basin Project Pilot In-Situ Uranium Mining Operation. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

OPI, 1981. Annual Report to Wyoming Department of Environmental Quality Covering August 28, 1980 to August 28, 1981. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

OPI, 1982a. Amendment Request to License SUA-1396, Condition No. 42, January 17, 1982, Docket No. 40-8745. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

OPI, 1982b. Quarterly Excursion Report, July 21, 1982, Docket No. 40-8745. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

Papadopulos, I. S., 1965. Nonsteady Flow to a Well in an Infinite Anisotropic Aquifer, Proceedings of Dubrovnik Symposium on Hydrology of Fractured Rocks, Internat. Assoc. of Sci. Hydrol., Vol. I, p. 21-31.

Papadopulos, I. S. and Cooper, H. H., Jr., 1967. Drawdown in a Well of Large Diameter, Water Resources Research, Vol. 3, No. 1, P. 241-244.

Payne, J. N., 1968. Hydrologic Significance of the Lithofacies of the Sparta Sand in Arkansas, Louisiana, and Texas and the Meridan Sand of Mississippi, U.S. Geol. Survey Prof. Pap. 569-A.

Resource Engineering & Development, Inc., 1980. Analysis of Groundwater Criteria and Recent Restoration Attempts After In Situ Uranium Leaching, Vol. 1, for U.S. Department of the Interior, Bureau of Mines, under Contract No. J0295019, October, 1980.

Riding, J. R. and Rosswog, F. J., 1979. Restoration of Ground Water Quality After In Situ Uranium Leaching, Final Report for U.S. Dept. of Interior, Bureau of Mines, Contract No. J0275028 with Ford, Bacon and Davis, Utah Inc.

RMEC, 1978a. Mine and Reclamation Plan, Rocky Mountain Energy Company.

RMEC, 1978b. Application for Research and Development Scale In Situ Leaching Operation Source Material License, May 4, 1978.

RMEC, 1979. Nine Mile Lake Environmental Report and Appendix, Unpublished Report, Docket No. 40-8380. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

RMEC, 1980a. Nine Mile Lake Review of Operations, Docket No. 40-8380, November 1980. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

RMEC, 1980b. Excursion Notification to Wyoming Department of Environmental Quality, Unpublished Report, November 1980. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

RMEC, 1980c. Amendment Request, Reno Creek, April, 1980.

RMEC, 1980d. Letter from R. Iwanicki, Rocky Mountain Energy Company, to J. E. Rothfleisch, U.S. Nuclear Regulatory Commission, July 25, 1980.

RMEC, 1980e. Letter from M. R. Newmann, Rocky Mountain Energy Company, to M. Hulburt, Wyoming Department of Environmental Quality, September 26, 1980.

RMEC, 1981a. Review of Restoration Activities, Reno Creek, March 9, 1981.

RMEC, 1981b. Reno Creek Demonstrated Restoration Report, November, 1981.

RMEC, 1981c. Letter from M. R. Newmann, Rocky Mountain Energy Company, to J. Brinkman, Wyoming Department of Environmental Quality, April 15, 1981.

RMEC, 1981d. Letter from M. R. Newmann, Rocky Mountain Energy Company, to U.S. Nuclear Regulatory Commission, April 3, 1981.

RMEC, 1982a. Nine Mile Lake Groundwater Restoration Report, Docket No. 40-8380. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

RMEC, 1982b. Quarterly Progress Reports for 1982, Nine Mile Lake Project, Unpublished Report. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

RMEC, 1983. Quarterly Report for First Quarter 1983.

Sharp, W. M. and Gibbon, A. B., 1954. Geology and Uranium Deposits of the Southern Part of the Powder River Basin, Wyoming, U.S. Geol. Survey Bull. 1147-D.

Sharp, W. M., McKay, E. J., McKeown, F. A., and White, A. M., 1964. Geology and Uranium Deposits of the Pumpkin Buttes Area of the Powder River Basin, Wyoming, U.S. Geol. Survey Bull. 1107-H.

Stallman, R. W., 1963. Type Curves for the Solution of Single Boundary Problems, in Shortcuts and Special Problems in Aquifer Tests, Compiled by R. Bentall, U.S. Geol. Surv. Water-Supply Pap. 1545.C, p. C45-C47.

TEDC, 1979. Environmental Report for Source Material License, Leuenberger Site, Converse County, Wyoming, Docket No. 40-8728, Teton Exploration Drilling Company.

TEDC, 1980. Source Material License Application, Leuenberger Site, Converse County, Wyoming, Docket No. 40-8781 (Revised 1981).

TEDC, 1982. M-Zone Restoration Report, Leuenberger In Situ Pilot Project, Converse County, Wyoming, Docket No. 40.8728.

Texas Department of Water Resources, 1977. In Situ Uranium Mining Permit No. 01942 (Ammendment), August 11, 1977.

Texas Department of Water Resources, 1979. Ammended Permit No. 01942, May 29, 1979.

Texas Department of Water Resources, 1980. Ammended Permit No. 01942, April 14, 1980.

Texas Department of Water Resources, 1984. Technical Summary Permit No. 01942-001, March 1, 1984.

Theim, G., 1906. Hydrologische Methoden, Leipzig, J.M. Gebhart.

Theis, C. V., 1935. The Relation Between the Lowering of the Piezometric Surface and the Rate and Duration of Discharge of a Well Using Ground-water Storage, Trans. Am. Geophys. Union, Vol. 16, No., 2, pp. 519-524.

Welder, G. E. and McGreevy, L. J., 1966. Ground Water Reconnaissance of the Great Divide and Washakie Basins and Some Adjacent Areas of Southwestern Wyoming, U.S. Geol. Survey Hydrol. Inv. Atlas HA-219.

WMC, 1977a. Environmental Report, Irigaray Project, Unpublished Report, Docket No. 40-8502, Wyoming Mineral Corporation. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1977b. Application for In Situ Mining Permit for the Bruni Mine, Vol. 1.

WMC, 1978a. Irigaray Restoration Demonstration Program, Final Report, Unpublished Report, March 13, 1978. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1978b. Production Area Authorization Application for Grid X (Production Area 4).

WMC, 1979a. Production Area Authorization Application for Grid II (Production Area 5), March 1, 1979.

WMC, 1979b. Impact Evaluation of Bruni Restoration Operations, March 7, 1979.

WMC, 1980. Report on Irigaray Mine Shallow Zone Evaluation, Vol I and II, Unpublished Report, Docket No. 40-8502. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981a, Application for In-Situ Permit to Mine for the Irigaray Site, Unpublished Report, Vol. I and IV, Docket No. 40-8502 (Revised 1982). Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981b, Application for In-Situ Permit to Mine for the Irigaray Site, Unpublished Report, Vol. III, Docket No. 40-8502 (Revised 1982). Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981c. Application for In-Situ Permit to Mine for the Irigaray Site, Unpublished Report, Vol II, Docket No. 40-8502 (Revised 1982). Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981d. Quarterly Progress Reports for 1981, Irigaray Project, Unpublished Report. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981e. Status Report on Unit 1 Sand Monitor Well SSM-5, June 29, 1981, Unpublished Report. Available at URFO, USNRC, 730 Simms, Lakewood, Colorado.

WMC, 1981f. Geohydrologic Information on Bruni Shallow Zone, Permit No. 019424m.

WMC, 1982a. Letter to Bill Klemt, Texas Department of Water Resources, from Wyoming Mineral Corporation, August 24, 1982.

WMC, 1982b. Letter to H. Davis, Texas Department of Water Resources from C. Rufledge, Jr., Wyoming Mineral Corporation, January 18, 1982.

NRC FORM 335 (2-84) NRCM 1102, 3201, 3202

U.S. NUCLEAR REGULATORY COMMISSION

BIBLIOGRAPHIC DATA SHEET

SEE INSTRUCTIONS ON THE REVERSE

1. REPORT NUMBER (Assigned by TIDC, add Vol. No., if any)
NUREG/CR-3967
ORNL/TM-9956

2. TITLE AND SUBTITLE
An Analysis of Excursions at Selected In Situ Uranium Mines in Wyoming and Texas

3. LEAVE BLANK

4. DATE REPORT COMPLETED
MONTH: May
YEAR: 1986

5. AUTHOR(S)
W.P. Staub, N.E. Hinkle/ORNL
R.E. Williams, F. Anastasi, J. Osiensky, D. Rogness/UI

6. DATE REPORT ISSUED
MONTH: July
YEAR: 1986

7. PERFORMING ORGANIZATION NAME AND MAILING ADDRESS (Include Zip Code)
Oak Ridge National Laboratory
Oak Ridge, TN 37831
Subcontractor: University of Idaho
Moscow, ID 83843

8. PROJECT/TASK/WORK UNIT NUMBER

9. FIN OR GRANT NUMBER
B0279

10. SPONSORING ORGANIZATION NAME AND MAILING ADDRESS (Include Zip Code)
Division of Waste Management
Office of Nuclear Material Safety and Safeguards
U.S. Nuclear Regulatory Commission
Washington, DC 20555

11a. TYPE OF REPORT
Technical

b. PERIOD COVERED (Inclusive dates)

12. SUPPLEMENTARY NOTES

13. ABSTRACT (200 words or less)

Intensive research and development of in situ uranium mining took place in the United States during the 1970s. By the end of the decade, nearly 10% of all uranium production came from in situ mines. Recent poor market conditions, however, forced the closure of all domestic in situ uranium mines.

When market conditions improve, domestic in situ mining is expected to make a modest recovery. Successful licensing of future in situ mines will depend to a large extent on the ability to contain the leach solutions (lixiviant) within the ore zone. Uncontrolled movement of lixiviant beyond the ore zone is called an excursion.

The industry has demonstrated the capability to reduce, monitor, and control lateral excursions in an ore zone aquifer. On the other hand, vertical excursions of lixiviant migrating into an overlying aquifer through poorly plugged exploration holes continue to plague the industry.

This paper summarizes the history of excursions at selected in situ mines in the United States and discusses methodologies for excursion identification and control. The incidence of vertical excursions can be significantly reduced by certifying well field units through field-wide testing of the ore zone aquifer prior to the injection of lixiviant. An overlying aquifer can be effectively monitored for excursions by periodically analyzing fluids in observation wells for nonreactive ionic species common to the lixiviant. These ions would necessarily have low probabilities of naturally exceeding a given concentration, referred to as an upper control limit. A mining unit would require recertification after each vertical excursion.

14. DOCUMENT ANALYSIS - a. KEYWORDS/DESCRIPTORS
in situ uranium mines
lixiviant excursions

b. IDENTIFIERS/OPEN-ENDED TERMS

15. AVAILABILITY STATEMENT
Unlimited

16. SECURITY CLASSIFICATION
(This page) Unclassified
(This report) Unclassified

17. NUMBER OF PAGES

18. PRICE

UNITED STATES
NUCLEAR REGULATORY COMMISSION
WASHINGTON, D.C. 20555

OFFICIAL BUSINESS
PENALTY FOR PRIVATE USE, $300

SPECIAL FOURTH-CLASS RATE
POSTAGE & FEES PAID
USNRC
WASH. D.C.
PERMIT No. G-67

120555078877 1 1AN1RU
US NRC
ADM-DIV OF TIDC
POLICY & PUB MGT BR-PDR NUREG
W-501
WASHINGTON DC 20555