Uranium in situ leaching

Proceedings of a Technical Committee Meeting
held in Vienna, 5–8 October 1992
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FOREWORD

The IAEA’s latest published estimates show that nuclear power production worldwide will maintain modest growth well into the next century. These estimates indicate that nuclear energy production will grow on average by 1.5% to 2.5% per year worldwide over the next two decades. It is for this reason that despite the continuing depressed uranium market, the question of uranium supply and demand remains an issue that needs to be addressed and monitored. New developments, due to the recent entry of the Commonwealth of Independent States and China into the uranium market, increase such a need.

Despite its limited applicability to specific types of uranium deposits, the in situ leaching (ISL) method of uranium production has grown in importance for its competitive cost and has proven to be an environmentally sound technology with very little disturbance to the environment. It was also recognized that there are two distinct approaches of ISL uranium production being practised in eastern Europe, in particular, and in the USA. Therefore, the IAEA convened a Technical Committee meeting with the purpose of bringing together experts in this particular field from all regions to share, exchange and disseminate information and to discuss and make recommendations on those topics on which additional work is needed.

The Technical Committee Meeting on Uranium In Situ Leaching was held in Vienna from 5 to 8 October 1992 and was attended by 38 participants from 15 countries. Seventeen papers covering the different methods of in situ leaching uranium production, its effect on the environment, groundwater restoration and related regulations were presented. Contributions from Bulgaria, China, Czechoslovakia, Germany, the Russian Federation and Uzbekistan represent new information in this field.

The IAEA is grateful to those participants who contributed papers and took part in the discussion. Special thanks are extended to W.M. Mays (USA), S.D. Simov (Bulgaria) and D.H. Underhill (USA), who chaired the sessions.

The IAEA staff member responsible for the organization and implementation of the meeting was M. Tauchid of the Division of Nuclear Fuel Cycle and Waste Management.
EDITORIAL NOTE

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SUMMARY OF THE TECHNICAL COMMITTEE MEETING

Despite the depressed situation that has affected the uranium industry during the past years, the second Technical Committee Meeting on Uranium In Situ Leaching, organized by the International Atomic Energy Agency and held in Vienna from 5 to 8 October 1992, has attracted a relatively large number of participants. A notable development since the first meeting was that the majority of the contributions came from the actual operators of in situ leaching uranium production. More importantly, the first meeting was mainly devoted to the experience and observations from operations in the USA, where the only commercial uranium production by this method is known. At the present meeting, presentations on operations in the USA were balanced by those of the eastern European and Asian countries. Contributions from Bulgaria, China, Czechoslovakia, Germany (from the operation in the former German Democratic Republic), the Russian Federation and Uzbekistan represent new information not commonly available.

In situ leach mining is defined in one of the papers presented as a "mining method where the ore mineral is preferentially leached from the host rock in place, or in situ, by the use of leach solutions, and the mineral value is recovered". At this meeting it was shown that there is a clear division between practices of in situ leaching (ISL) uranium production in the USA and those in the eastern European countries (which include operation in the former German Democratic Republic), Central Asia and China.

In situ leaching operations in the USA are basically a no-entry system using alkaline (carbonate) solution as lixiviant and oxygen as oxidant. This system can be applied only to a rather specific type of sandstone uranium deposit, in particular of the roll-front type. In the eastern European countries, both no-entry and underground entry methods (underground preparation of the ore to be leached) are the normal practice. Various methods of underground entry system are well illustrated in the papers of R. Hähne and G. Altmann (Germany) and of B.T. Tabakov (Bulgaria). Almost all of the latter operations use sulphuric acid as lixiviant and nitric acid or hydrogen peroxide as oxidant. Stringent regulatory requirements in the USA to restore the ore body aquifer quality to premining conditions limits the use of the more aggressive acid as leaching agent. As demonstrated by operations in Bulgaria, Germany and China, the advantage of the entry approach of in situ leaching of uranium production is the possibility of applying this technique to non-sandstone types of uranium deposits. One of the main questions faced by the ISL uranium production in the USA is the maximum depth in which such a method of commercial mining remains feasible. Past and present pilot and commercial productions have been limited to deposits of less than 300 m deep. As noted in some of the papers presented, there is basically no depth limitation to the underground entry approach.

There was unanimous agreement that the main advantage of the ISL method of uranium production, as compared to conventional, open pit and/or underground mining, is financial. However, there is still no unified opinion on what is considered the best process causing the least harm to the environment. As noted by D.H. Underhill, about 30 pilot and commercial production plants in the USA had been successfully restored by the end of 1987. W.M. Mays' paper on restoration of groundwater after ISL operations further confirms that it is an environmentally sound method of mining. While some successful tests of restoration were reported, there is still a lack of actual experience in the complete

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1 The proceedings of the first Technical Committee Meeting on In Situ Leaching of Uranium, held in Vienna, 3–6 November 1987, were published as IAEA-TECDOC-492 (1989).
restoration of the affected groundwater system after an acid leach operation. This is primarily due to the rather long period of time (20 to 25 years or more) in the operation of all of the reported cases. Work presently carried out in Czechoslovakia, Bulgaria and Germany may provide examples for such a restoration and remediation programme.

The following list, taken from D.H. Underhill's paper, shows the advantages of ISL of uranium production by alkali leaching (a similar list is provided by J.I. Skorovarov and M.I. Fazlullin with reference to the acid system): lower capital cost; shorter lead time to production; less energy intensive; less usage and maintenance of equipment; very low labour intensity per unit of product; substantially reduced personal radiation exposure; less surface disturbance and pollution; less waste generation; allowing the possibility to recover low grade ores; and the possibility of exploiting otherwise inaccessible deposits. Because of its lower cost of production, experience in the USA shows that only ISL operations can compete under the present depressed uranium market. However, it is doubtful that uranium production by the ISL method, or any existing production centres, can survive the continuity of the present low uranium price, which is actually lower than the cost to produce it.

One paper dealt with the regulatory requirements for the development of in situ mining. This is obviously a topic which is of general interest, particularly to those countries where such a formalized approach has not yet evolved. More uniform regulations for licensing and monitoring is definitely a subject that requires attention by countries with existing in situ uranium production as well as by countries where such a potential exists.

It may be noted that in 1991 about 16% (i.e. 7150 tonnes U) of the estimated total world uranium production of 44 500 tonnes U came from ISL operations. Around 14% of these came from eastern Europe, Central Asia and China. With the present closure of a number of conventional mines in the USA and Canada, and the suspension of the by-product operation from the phosphoric acid plant in Florida, USA, it is quite obvious that ISL will play a more important role in future world uranium production.
WORLD URANIUM PRODUCTION AND DEMAND: A REVIEW

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Abstract

Despite the growing public concern on the use of nuclear energy, nuclear power generation capacity in the world is expected to follow a modest, but positive growth at least during the next two decades. Uranium production needed to fuel these reactors has been below demand since 1985. The WOCA production figure for 1991 is in the order of 27 000 tonnes U which is 39% below the peak production of 1980. With the exception of Australia, all other countries produced less uranium than in the previous year. It is expected that the production figure for 1992 will shrink even further to about 23 000 tonnes U. In-situ leaching uranium production contributed about 16% to the 1991 world production figure, most of which came from Eastern Europe and Central Asia. With the closing of a number of production facilities the relative contribution of in-situ leaching to the world uranium production is expected to grow.

Only about 60% of WOCA’s reactor related uranium demand for 1991 was supplied from its own production. The remaining 40% was filled from existing inventories and imports from the Russian Federation and China. The estimated gap between the world uranium production and reactor related demand for 1991 is in the order 10 900 tonnes U or 19.7%. The cumulative requirement for the world reactor related demand to the year 2010 has been estimated to be about 1 270 000 tonnes U.

INTRODUCTION

With the exception of a relatively small group of people who are involved with the uranium industry, there seems to be a general feeling that there is unlimited amounts of uranium to fuel existing and future operating nuclear power reactors. A careful look into the situation will show that the picture is not as clear nor simple as it may seems. This feeling of complacency is being reinforced by the continuing depressed price of uranium in the spot market which has reached its lowest level during the past few months, lower than the cost to produce it. It is not a surprise that more and more production facilities around the world are closed, to be closed or kept on stand-by.

The purpose of this presentation is to provide this forum with a brief background information on current uranium production and its relation to future short and long-term requirements in accordance with the projected growth of installed nuclear power capacity. Much of this information is extracted from the NEA(OECD)/IAEA report "Uranium Resources, Production and Demand", popularly known as the "Red Book" [1], and more recent information available to the Agency.

Despite of the growing public concern related to nuclear power generation the world over, nuclear energy generation is expected to follow a modest, but continuing growth to the year 2010 at the rate of 1.5% to 2.5% annually. As shown in Figure 1, the Middle East and South Asia followed by Latin America are showing the higher rate of annual growth of 11.9% to 13.7% and 6.1% to 8.0% respectively [2].
URANIUM PRODUCTION

Of interest to the present forum is the world uranium production and how much of it is produced by in-situ leaching method. To have a better appreciation on the up and down of the uranium industry, at least in WOCA where more information is available, it is worthwhile to review the historical uranium production for the past two decades. As can be seen from Figure 2 [1], there was a steep increase in the uranium production after the energy crisis of 1973 which peaked in 1980 with a production of 44 200 tonnes U. Since then, a gradual but continued decline is observed to the point that in 1985, for the first time, production was actually below reactor related demand.

The production figure for 1991 is in the order of 27 000 tonnes U, which is 39% below the peak production of 1980. With the exception of Australia, all other countries produced less uranium than in the previous year. Most notable are the reductions in South Africa, Namibia and the U.S.A. with 33%, 24% and 18% respectively. With the present closures of a number of production centres and that existing facilities are producing at below capacity, it can be assumed that the production figures for 1992 will shrink even further. A figure in the order of 23 000 tonnes U has been suggested [3, 4, 5].

From the selected countries noted in Figure 2, in-situ leaching production is known only in the U.S.A. Figure 3 shows the 3.1% share of ISL in the 1991 WOCA uranium production. This includes the small amount produced in Germany. The contribution from the phosphoric acid byproduct is about 1 200 tonnes U or 4.4%.

Uranium production in Eastern Europe, Central Asia and China for 1991 is estimated to be around 17 500 tonnes U. As noted in Figure 4, quite a large percentage of this production comes
from in-situ leaching facilities which amounts to about 6 300 tonnes U or 36% of the total. A more accurate estimate can probably be obtained from the various papers to be presented at this meeting.

Combining the two above mentioned figures, one comes up with 44 500 tonnes U as the World uranium production for 1991, of which 7 150 tonnes U, or 16.1% was produced by in-situ leaching process (Figure 5). This year closures of the two remaining conventional mining in the U.S.A. (Rhode Ranch, Texas and Shirley Basin, Wyoming), the Canadian Denison Mines (Elliot Lake, Ontario) and the stoppage of production from the IMC phosphoric acid plant in Florida, will certainly increase the contribution from the in-situ leaching in the overall world production, at least in a relative sense.


FIG. 6 WOCA uranium production and demand.

FIG. 7 Calculated civil uranium inventories at end of year (thousands of tonnes of uranium) [5]
INVENTORIES

One can not talk about uranium supply situation without mentioning inventories. Figure 6 illustrates how in WOCA, the uranium overproduction through about 1985, has led to inventory build up. An accurate estimate on the actual amount of this inventory is difficult. However, it was estimated that it reached a maximum of about 200 000 tonnes U in the early 1980’s [5]. Most quoted figure indicates that, despite of a draw down since 1986, this inventory is still estimated to be in the order of 150 000 to 160 000 tonnes U (Figure 7).

A similar development can be assumed for the former Comecon area whose cumulative production from 1945 to 1990, mainly from the former GDR, CSFR and the former USSR, is being estimated to total about 600 000 tonnes U. The unused portion of these production, estimated to be in the range of 140 000 - 160 000 tonnes U, is being held by the Russian Federation.

Combining the two civil inventories noted above will give an estimated total of about 300 000 tonnes U. The amount that might be available on the market has been the subject of numerous speculations. Assuming that utilities will carry a stockpile for 2 - 3 years requirements would make approximately 2/3 of these inventories free for the market.

SUPPLY - DEMAND RELATION

History has shown that the most dangerous thing to do is to make a supply demand projection. It practically paralyzed the uranium industry and all activities that are connected to it, to the point that uranium become an unwanted word, even in places where it should still command some respect. With this precautionary note let us review the present situation and what might be in front of us for the next two decades.

<table>
<thead>
<tr>
<th>Country</th>
<th>Production</th>
<th>%</th>
</tr>
</thead>
<tbody>
<tr>
<td>CIS</td>
<td>13500</td>
<td>24.4%</td>
</tr>
<tr>
<td>South Africa</td>
<td>1750</td>
<td>3.2%</td>
</tr>
<tr>
<td>Niger</td>
<td>2960</td>
<td>5.4%</td>
</tr>
<tr>
<td>Namibia</td>
<td>2500</td>
<td>4.5%</td>
</tr>
<tr>
<td>Germany</td>
<td>1190</td>
<td>2.2%</td>
</tr>
<tr>
<td>France</td>
<td>2508</td>
<td>4.5%</td>
</tr>
<tr>
<td>Canada</td>
<td>8100</td>
<td>14.6%</td>
</tr>
<tr>
<td>Australia</td>
<td>3530</td>
<td>6.4%</td>
</tr>
<tr>
<td>USA</td>
<td>3500</td>
<td>6.3%</td>
</tr>
<tr>
<td>Rest of World</td>
<td>4870</td>
<td>8.8%</td>
</tr>
<tr>
<td>Production Gap</td>
<td>10892</td>
<td>19.7%</td>
</tr>
</tbody>
</table>

World Demand: 55 300 t U

FIG. 8 Estimated 1991 world uranium production vs reactor related demand.
FIG. 9. World nuclear generating capacity projection.

FIG. 10. World uranium requirement projection.
Only about 60% of WOCA's reactor related uranium demand for 1991 was supplied from its own production. The remaining 40% was filled from existing inventories and imports from the Russian Federation and China. Looking at the world as a whole, Figure 8 shows a production gap of about 10 900 tonnes or 19.7% of the total reactor related uranium requirement. From the foregoing discussion, as world mine output is shrinking, at least in the next coming 2-3 years, this production gap is certainly to grow even larger.

The introduction of the new recommendations of the International Commission of Radiological Protection (ICRP), known as ICRP 60, to reduce the occupational dose limit from 50 mSv/a to 20 mSv/a averaged over 5 years, will probably add a negative effect to future uranium production.

Figure 9 shows a projection of the expected growth in the Western and Total World nuclear generating capacity from 1992 to 2010. The projection for the years between 2005 and 2010 are uncertain, since in this period numerous nuclear power plants reach the end of their rated lifetime and there is uncertainty if these plants will be decommissioned or if their life will be extended by appropriate technical measures.

If this projection is translated into reactor related uranium demand, for the Western World, it means an increasing requirements from about 46 500 tonnes U in 1992 to 57 100 tonnes U in 2010, and for the Total World from 56 800 tonnes U and 76 500 tonnes U respectively. As shown in Figure 10, the cumulative requirement to the year 2010, for the Western World is estimated to be about 990 000 tonnes U and for the World 1 270 000 tonnes U, a large amount of materials that can not be ignored by the uranium industry and utilities alike.

If these figures are compared to the corresponding projected production capacities, it becomes apparent that the projected demand will not be filled with the newly produced uranium. This production gap will certainly be filled from excess inventories in East and West, but at the end of the century new production centres, over and above those which are presently operating and committed, will be needed to fill this gap.

Noritake [6], at the recent Uranium Institute Annual Symposium, presented an interesting paper on computer model to forecast uranium supply-demand relation to the year 2030. The model considered the availability of uranium resources at different price category, excess inventories, recycled uranium, as well as those coming from military high enriched uranium (HEU). His conclusion was that without the military source, after the year 2014 demand can not be met by presently operating and new mines expected to have gone into operation by that time. This crucial date is pushed back only by two years if military nuclear material comes into use.

CONCLUSIONS

It is to be acknowledged that the need of energy in many parts of the world is still growing. Effort to reduce environmental degradation and the much discussed "green house" effect may awaken the society on the positive benefit that nuclear power generation can offer.

Despite of the growing public concern on the use of nuclear energy, nuclear power generation capacity in the world is expected to follow a modest, but positive growth at least during the next two decades. So is the need for uranium to fuel them. From the foregoing discussion, it
is obvious that there is a need to analyze the uranium supply-demand situation very carefully if the world is not to face with a repeat of the mid 1970's event.

Coming back to the theme of the present meeting, for its competitive production cost, the role of in-situ leaching in uranium production, in established and new areas with similar deposit type, is expected to grow. There is a need to review the technology in terms of its cost effectiveness and environmental acceptance. An appropriate and more uniform regulations to minimize environmental damage caused by this activity is probably needed.

References

IN-SITU LEACH URANIUM MINING IN THE UNITED STATES OF AMERICA: PAST, PRESENT AND FUTURE

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Abstract

Commercial in-situ leach (ISL) uranium mining in the United States began in the mid-1970s. Both acid and alkaline leach systems were evaluated. The difficulty of restoring ground water following acid leaching, led to exclusive adoption of alkaline leach systems. The low production cost achievable from carefully selected, effectively run projects, was primarily responsible for the adoption and increased use of the unconventional production technology. Today ISL production continues, while production from all conventional uranium mines has been suspended. This paper describes the history of ISL uranium mining in the U.S. While 85 percent of ISL production has come from south Texas, today projects are also operating in Wyoming and Nebraska. Now that most of the ISL amenable reserves in Texas have been mined, most new projects will be in Wyoming, Nebraska and New Mexico. This paper discusses why ISL production costs are relatively low and can therefore compete with low cost, high grade conventional mines. It describes the ore reserves and production capacity of all of the installed, planned and potential U.S. ISL projects. It discusses why ISL mining is expected to be the predominant uranium production technology in the U.S. over the next 10 or 15 years.

1. INTRODUCTION TO IN SITU LEACH URANIUM MINING IN THE USA

Since the start of commercial in-situ leach (ISL) uranium mine production in the USA in 1975, ISL uranium mining has grown from an obscure, experimental technology to the dominant U.S. producer. U.S. uranium production reached its maximum level in 1980 when 16,809 MTU were produced. Of this amount about 1,600 MTU or nearly 10 percent was produced by ISL mining. The 1980 production came from 22 conventional mills processing ore from 303 underground and 52 open pit mines, in addition to 11 ISL projects, 6 phosphate by-product and 4 other producers [1].

At present (October 1992) in the U.S. there are four ISL projects, one phosphate by-product and one mine water recovery plant operating. All conventional uranium mining activity ended earlier this year, with the permanent closure of two open pit mines. Freeport Uranium Recovery Company’s phosphate by-product facilities operation produces at it’s capacity of about 385 MTU per year. Rio Algom Mining Company’s mine water recovery operation produces less than 77 MTU per year. It is estimated that about 795 MTU or over 40 percent of the total 1992 U.S. production will come from ISL facilities. In 1993, because of the recent shutdown of non-ISL production centers, it is expected that ISL production will make up an even higher proportion of the total U.S. production.

Since 1975 ISL technology has produced 14,852 MTU or about 10.2 percent of the U.S. production. With the addition of 577 MTU, produced at Shirley Basin, Wyoming between 1963 to 1970, about 15,428 MTU or 4.6 percent of the total U.S. production between 1947 and 1991 was by ISL mining.

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The four operating ISL production centers are producing at only 35 percent of their combined capacity. It is evident that even lowcost ISL operations are not immune to the present low priced uranium market. Although current ISL production is small when compared to the peak 1980 production it is remarkable that ISL production has continued at all in today’s market. ISL uranium mining production has survived because the technology is capable of producing at low cost (i.e. $26 to $52 per kgU) from carefully selected projects.

ISL production also continues because the project owners believe that these low cost production centers will be very competitive in the anticipated improving uranium market. Analysis of the U.S. uranium industry indicates that ISL mining will be the dominant production technology over the next 10 to 15 years. Identified reserves with a forward production cost of under $52 per kgU could support the planned development of an annual capacity of over 6,000 MTU.

ISL production will continue to increase in importance primarily because of the relative economic advantage of the lowcost technology. Production from ISL technology will also dominate the U.S. uranium industry because of the limited availability of uranium reserves suitable for lowcost conventional production. Conventional production capacity is also being negatively impacted by the ongoing decommissioning of conventional uranium mills and the increasingly restrictive U.S. regulatory environment.

Successful ISL uranium mining in the early 1990s represents a maturing technology, applied by experienced management and personnel in well defined geologic environments. The current growth in ISL production follows the experimental years of the 1970s and early 1980s, during which there were more unsuccessful than successful ISL projects. Today alkaline leaching systems are used to mine very specific types of orebodies. These are rollfront-type sandstone deposits with favorable geologic and hydrologic characteristics.

Commercial ISL production began in 1975 and had increased to an annual level of about 1,731 MTU by 1982. Following a slow contraction of ISL uranium production, which had decreased to about 538 MTU in 1987, production has stabilized at between 577 and 769 MTU per year over the last few years. Today a refined technology is being employed in the well established ISL uranium mining districts of Texas, as well as in new areas of Wyoming and Nebraska. In the past, projects produced from ore bodies at depths ranging from 75 to 230 meters. Now ISL mining at depths of 300 meters or more is planned in New Mexico.

The most successful operating projects involve selective mining of the better grade ore in favorable geologic environments. More favorable economics are being attained by developing larger projects, which benefit from economies of scale, and automated plants requiring smaller operating staffs. This new generation of projects has in many cases economically benefited by recycling plants and equipment from earlier projects. Some of the projects have extensive ore reserves delineated by former project owners. All of these factors contribute to making ISL uranium mining, with production costs in the $26 to $52 per kgU range, competitive with production from high grade conventional mines.

At a 1978 uranium supply conference, one of the speakers stated that ISL mining of uranium was still in the developmental stage [2]. He indicated that many problems remained to be solved, including questions regarding:
- underground fluid control
- well completion techniques
- selection of the leaching agent

20
- selection of the oxidizing agent
- non-selective oxidation and leaching
- co-precipitation of uranium after leaching

Although these uncertainties may not have been completely eliminated, experience gained during the 1980s has helped ISL project operators to substantially reduce the risk associated with these issues and/or problem areas. During the last 15 years there have been many changes and developments in all phases of the process. These changes involve property evaluation standards, geohydrology, wellfield development, leach chemistry, plant design and operation, and reclamation technology. Licensing and regulatory practice have also changed.

As a result, today the level of confidence in uranium ISL technology is much greater than it was in the late 1970s. During the early 1980s, not one ISL wellfield had been restored and there was some question whether restoration could be achieved by practical means. By the end of 1987 about 30 commercial and pilot uranium solution mine wellfields had been restored in Wyoming, Texas, Colorado and New Mexico.

In most cases today wellfield restoration is routine. This has helped to assure both federal and state regulators that ISL mining does not significantly impact the environment. As a result, in May 1989, a representative of the U.S. Nuclear Regulatory Commission (NRC) wrote that "Based upon the accumulation of operational data and information, it has become apparent that ISL operations pose no significant environmental impacts." [3]. As compared with the mid-1980s the requirements for permitting and licensing ISL operations have generally become more well defined and the average time required to permit projects has decreased. Today permitting and licensing of a new project takes from two, to more than four years, depending on the location and circumstances.

Probably the single greatest ongoing challenge is the application of ISL technology to economically mine deeper orebodies. Most of the earlier projects mined uranium at depths of 90 to 150 meters. Current ISL projects have extended into the 150 to 250 meter range. Some of the planned projects will operate at depths of 300 meters or more. The Mobil Crownpoint research and development project successfully produced uranium at a depth of 610 meters. While the Mobil pilot project was a technical success no information is available to indicate the level of economic success.

Additional challenges will involve the application of ISL technology in new areas, such as New Mexico, where geologic conditions differ from those of established producing districts. The continuing effort to streamline project design to minimize costs is an continuing challenge. However, with the exception of projects located in New Mexico, all of the installed and planned projects will generally operate within the range of previous ISL experience.

In the United States the record of ISL uranium mining speaks for itself. ISL mining has survived and there are plans for expanding use of the technology. Meanwhile conventional underground and open pit uranium mining has experienced severe cutbacks. While the principal advantages of ISL mining over conventional mining are financial, there are several other advantages. Given favorable conditions, the principal advantages include [4]:
- lower capital and operating costs, improved cash flow, and a generally greater return on investment;
- shorter lead time to production;
- less energy intensive;
- less equipment to maintain;
- very low labor intensity per unit of product;
- substantially reduced personal radiation exposure;
- less surface disturbance and pollution;
- less waste generation and fewer disposal problems;
- lower ore grades can be treated in some cases - recoverable reserves of uranium are therefore increased;
- potential for application to otherwise inaccessible deposits.

As noted above the most significant advantage relates to the favorable economics of ISL production. However, the flexibility of ISL operations is also very important [5]. For example, the short lead times to bring a project into production as compared with conventional projects is a real advantage. The ability to selectively mine and adjust the cutoff grade of deposits is a strategy that all of the current ISL operators are using. The potential for readily adjusting ISL production to match market requirements is another strength of the technology. These factors have all helped to make ISL uranium producers more competitive in today's market.

1.1 Overview of the ISL Mining Process

In-situ leach (ISL) mining is defined as that mining method where the ore mineral is preferentially leached from the host rock in place, or in-situ, by the use of leach solutions, and the mineral value is recovered. Although this definition can include the use of explosives or hydraulic fracturing techniques to fragment an ore body in preparation for in-situ leaching, these techniques are not employed in current U.S. ISL uranium mining. Uranium dump or heap leaching would not be included.

In general, ISL extraction consists of injecting a suitable leach solution (lixiviant) into the ore zone below the water table; oxidizing, complexing, and mobilizing the uranium; recovering the pregnant solution through production wells; and finally pumping the uranium-bearing solution to the surface for further processing.

In the past various types of injection-recovery well configurations or patterns have been used. A five-spot pattern (a production well surrounded by four injection wells at the vertices of a square or rectangle) is the most commonly used configuration. Other less regular or geometric patterns such as line drives or staggered line drives are used to take advantage of ore body configuration, site-specific hydrology, geology, etc. Seven-spot patterns are also used. Injector to production well spacing normally ranges between 15 and 30 meters.

Proper well construction and completion methods are among the most important aspects of successfully bringing an ISL uranium wellfield into production. If a completed well does not function as was intended and another well must be drilled, extraction costs attributable to that well are doubled. An inoperative production well could mean lower overall uranium recoveries. Various types of well integrity tests are now required to assure that well casings perform according to design specifications.

More specifically, carefully constructed injection wells are used to inject an appropriate lixiviant, usually dilute concentrations of a carbonate-bicarbonate, and a suitable oxidizing agent, usually dissolved oxygen, into the ore zone. The lixiviants are maintained at a near neutral pH of 7.0 to 7.4. The leach solution and oxidizing agent migrate through the permeable sandstone and contact the uranium minerals. The oxidant oxidizes the uranium, changing it from an insoluble
tetravalent to a soluble hexavalent state. When a carbonate leach solution is used the uranium is mobilized as a soluble uranium carbonate complex.

Once a uranium complex is formed and is mobilized in the leaching solution, it moves down the pressure gradient towards a production well. In the well, submersible pumps transfer the uranium-bearing solution to the surface for processing. Ion exchange technology is used to recover the uranium from the pregnant liquor. The residual uranium-barren solution from the ion exchange operation is recharged with leaching chemicals and recirculated into the wellfield through injection wells.

Ion exchange is a cyclic operation that consists of two steps, the loading or adsorption step and the elution step. During the loading step, the pregnant leach solution contacts the ion exchange resin and the uranium is selectively adsorbed. When a suitable uranium loading has been achieved,
the resin is cycled to the elution step. During elution, the resin is contacted with a chemical solution that strips the uranium from the resin. The uranium enriched solution is called the eluate. The eluted resin is then cycled back to the loading operation.

The uranium is recovered from the eluate by precipitation. The precipitate is usually ammonium diuranate or uranyl peroxide, both of which are normally termed yellowcake. The yellowcake precipitate is separated from the residual solution by thickening and filtration. The filter cake may be partially dewatered to form a yellowcake slurry, or it may be dried and packaged for shipment. An example of a flow diagram for an ISL uranium mining process is given in Figure 1.

2. HISTORICAL PROJECT ACTIVITIES

In the U.S., experimentation with ISL mining started in the early 1960s led to more systematic development of the technology in the 1970s. In Wyoming the first ISL uranium mine was operated by Utah Construction and Mining Co., (later to become Utah International Inc., and then Pathfinder Mines Corp.) at its Shirley Basin site. Uranium mining started at Shirley Basin in 1959 using conventional underground methods. Following recovery of about 460 MTU, underground mining was discontinued in 1963 because of adverse ground conditions. From 1963 until 1970 patented ISL mining techniques were used to recover about 577 MTU. A sulfuric acid leaching system was used.

Utah Construction reportedly [6] used many of the same principles and techniques currently used, including ion exchange (IX) systems, pattern drilling, and the use of leach solutions with an oxidizer. During 1961-63, Utah Construction experimented with many techniques, including well development procedures and leach solutions. By 1963 the company had experimented with and tried 5 generations of well field designs and had drilled over 100 well field patterns in the attempt to maximize recoveries. From 1963 to 1969 ISL mining was the only method used by this company for uranium production. Open pit mining was started nearby at the site in 1969 and in 1970 the ISL operation was replaced by open pit mining. The open pit mines continued to produce uranium until their final closure early 1992.

In New Mexico ISL mining for uranium was first reported in early 1970, when Anaconda experimented with the North Windup Project at the Jackpile-Puguate mine area near Laguna. Pump tests from two well fields recovered uranium from a depth of 61 to 73 meters in the Jackpile Sandstone Member of the Morrison Formation.

Initially each well field consisted of one injection well surrounded by nine production wells on 60-meter centers. To improve hydrologic control well spacing was adjusted resulting in 2 injection wells and 29 production wells. A sulfuric acid lixiviant was used with uranium recovery by IX resin on the surface. Loaded resin was trucked to the Anaconda Bluewater mill some 80 kilometers west of the site where elution, precipitation, clarification, decantation, drying and packaging were done. Production was not reported and the project was discontinued [7].

Uranium was first produced in south Texas by conventional open pit mining. As exploration progressed, uranium mineralization was discovered at increasingly greater depths and the limits of open pit mining were exceeded. The unconsolidated, saturated condition of the host sands made underground mining impractical. This was one of the reasons that ISL mining was first considered for the recover of uranium in Texas. As experience with ISL mining increased it became apparent that for some deposits ISL mining has economic advantages that can not be equalled by other
Table I

U.S. IN SITU LEACH URANIUM CONCENTRATES PRODUCTION (MTU)

<table>
<thead>
<tr>
<th>Year</th>
<th>Number of ISL Projects</th>
<th>U.S. Total</th>
<th>ISL</th>
<th>Percent ISL</th>
</tr>
</thead>
<tbody>
<tr>
<td>1975</td>
<td>NA</td>
<td>8,923</td>
<td>106</td>
<td>1.2%</td>
</tr>
<tr>
<td>1976</td>
<td>NA</td>
<td>9,805</td>
<td>192</td>
<td>2.0%</td>
</tr>
<tr>
<td>1977</td>
<td>NA</td>
<td>11,491</td>
<td>600</td>
<td>5.2%</td>
</tr>
<tr>
<td>1978</td>
<td>NA</td>
<td>14,220</td>
<td>1,223</td>
<td>8.6%</td>
</tr>
<tr>
<td>1979</td>
<td>NA</td>
<td>14,412</td>
<td>1,346</td>
<td>9.3%</td>
</tr>
<tr>
<td>1980</td>
<td>NA</td>
<td>16,809</td>
<td>1,602</td>
<td>9.5%</td>
</tr>
<tr>
<td>1981</td>
<td>14</td>
<td>14,798</td>
<td>1,708</td>
<td>11.5%</td>
</tr>
<tr>
<td>1982</td>
<td>18</td>
<td>10,334</td>
<td>1,298</td>
<td>12.5%</td>
</tr>
<tr>
<td>1983</td>
<td>10</td>
<td>8,138</td>
<td>908</td>
<td>11.2%</td>
</tr>
<tr>
<td>1984</td>
<td>14</td>
<td>5,724</td>
<td>700</td>
<td>12.2%</td>
</tr>
<tr>
<td>1985</td>
<td>10</td>
<td>4,351</td>
<td>758</td>
<td>17.4%</td>
</tr>
<tr>
<td>1986</td>
<td>12</td>
<td>5,195</td>
<td>525</td>
<td>10.1%</td>
</tr>
<tr>
<td>1987</td>
<td>15</td>
<td>4,997</td>
<td>560</td>
<td>11.2%</td>
</tr>
<tr>
<td>1988</td>
<td>11</td>
<td>5,050</td>
<td>956</td>
<td>18.9%</td>
</tr>
<tr>
<td>1989</td>
<td>9</td>
<td>5,322</td>
<td>976</td>
<td>18.3%</td>
</tr>
<tr>
<td>1990</td>
<td>5</td>
<td>3,418</td>
<td>560</td>
<td>16.4%</td>
</tr>
<tr>
<td>1991</td>
<td>6</td>
<td>3,038</td>
<td>834</td>
<td>27.5%</td>
</tr>
</tbody>
</table>

Total (1975-1991) 146,025 14,850 10.2%
Shirley Basin (1963-1970) 577
Total U.S.A. (1947-1991) 337,033 15,427 4.6%

Table 2

U.S. ISL URANIUM PRODUCTION THROUGH 1991 AND PROJECT STATUS BY STATE

<table>
<thead>
<tr>
<th>State</th>
<th>Production (MTU)</th>
<th>Percent</th>
<th>Commercial Projects</th>
<th>Pilot Projects</th>
<th>Planned Projects</th>
</tr>
</thead>
<tbody>
<tr>
<td>Texas</td>
<td>16,872</td>
<td>85.4%</td>
<td>20</td>
<td>30</td>
<td>4</td>
</tr>
<tr>
<td>Wyoming</td>
<td>3,483</td>
<td>17.6%</td>
<td>3</td>
<td>25</td>
<td>7</td>
</tr>
<tr>
<td>Nebraska</td>
<td>140</td>
<td>0.7%</td>
<td>1</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>New Mexico</td>
<td>0</td>
<td>0.0%</td>
<td>0</td>
<td>6</td>
<td>3</td>
</tr>
<tr>
<td>Total</td>
<td>19,745</td>
<td>100.0%</td>
<td>24</td>
<td>62</td>
<td>15</td>
</tr>
</tbody>
</table>

25
mining methods. The technology of in situ leaching of uranium deposits using chemicals was readily accepted in Texas, where many of the uranium industry personnel were familiar with designing and operating fluid handling systems used in the oil and gas industry.

Mays reports [8] that at least six pilot plants operated in the U.S. in the early 1970s. Most of these tests were small and involved only one or a few well patterns. The first large test involved the Dalco, Atlantic Richfield and U.S. Steel In Situ Uranium Project (Clay West), Texas in 1975. This group carried out extensive tests using 13 well patterns with a flow capacity of 25 liter/second. The Clay West commercial project involved scaling up the pilot to 150 liters/second of flow capacity and 96 MTU per year. Clay West operated at this capacity until 1978 when the expanded plant began producing at an annual rate of 385 MTU.

During the same period of time several other ISL projects were developed in South Texas. Several of these projects experimented with and tested various aspects of the technology including leaching systems, oxidants and ion exchange systems. The rising uranium price of the mid-1970s and the success of the Clay West project resulted in intense interest in ISL mining. Mays [9] reports that by 1984, 8 pilot plants and 21 commercial plants had been put into production in Texas, Wyoming and New Mexico.

A summary of ISL production from 1975 through 1991 is given in Tables 1 and 2. Over this period, production rose from a modest 106 MTU to a maximum of 1,708 MTU in 1981. ISL production then decreased to 525 MTU in 1986. Output again increased and has varied between 560 MTU and over 923 MTU per year since 1988. Total production for the period 1975 through 1991 was about 14,852 MTU, or 10.2 percent of total U.S. production for the period. Texas produced 85.4 percent of the total ISL production.

Certain observations are warranted. Mays reports that there have been more unsuccessful than successful ISL projects. While there were several pilot tests using acid leach systems there have been no commercial ISL operations in the U.S. using an acid leach system. There has been only one acid leach pilot test in south Texas: the Duderstadt project operated by Cities Service in 1969 to 1971.

Test results of both acid and alkaline leach systems showed that in geologic environments with low carbonate content (i.e. less than a few percent) acid systems frequently have advantages. These include high yield and efficient rapid recoveries. Acid systems have particular advantages where the ore mineral is contained or coated by other minerals, or is otherwise leach resistant. The problem with strong acid leach systems is that they solubilize large amounts of other chemical constituents. These must be removed in the surface recovery plant. Restoration of the orebody aquifer following mining is the major problem for acid leach systems. In the U.S., regulators routinely require that the orebody aquifer quality be restored to premining quality.

Table 3 shows the constituents in both acid and alkaline leach solutions tested at the Irigaray, Wyoming pilot plant [10].

Review of the table indicates that with the exception of Radium-226 and arsenic, the concentration of all constituents is higher in the acid than in the alkaline lixiviant. This difference illustrates the more aggressive behavior of the sulfuric acid leach solution.
Table 3
Partial Composition of Recirculated Acid and Alkaline Lixiviants

<table>
<thead>
<tr>
<th>Constituent</th>
<th>Acid System Concentration mg/liter</th>
<th>Alkaline System Concentration mg/liter</th>
</tr>
</thead>
<tbody>
<tr>
<td>Arsenic</td>
<td>&lt;0.05</td>
<td>&lt;0.05</td>
</tr>
<tr>
<td>Copper</td>
<td>1.00</td>
<td>0.04</td>
</tr>
<tr>
<td>Zinc</td>
<td>4.30</td>
<td>0.10</td>
</tr>
<tr>
<td>Lead</td>
<td>0.70</td>
<td>0.20</td>
</tr>
<tr>
<td>Iron</td>
<td>25.40</td>
<td>0.60</td>
</tr>
<tr>
<td>Nickel</td>
<td>0.60</td>
<td>0.06</td>
</tr>
<tr>
<td>Chromium</td>
<td>0.15</td>
<td>0.07</td>
</tr>
<tr>
<td>Strontium</td>
<td>3.70</td>
<td>1.50</td>
</tr>
<tr>
<td>Zirconium</td>
<td>3.30</td>
<td>0.90</td>
</tr>
<tr>
<td>Selenium</td>
<td></td>
<td>1.60</td>
</tr>
<tr>
<td>Manganese</td>
<td>1.20</td>
<td></td>
</tr>
<tr>
<td>Molybdenum</td>
<td></td>
<td>0.90</td>
</tr>
<tr>
<td>Radium-226</td>
<td>390 pCi/l</td>
<td>1750 pCi/l</td>
</tr>
<tr>
<td>Vanadium</td>
<td>1.00</td>
<td></td>
</tr>
<tr>
<td>Cobalt</td>
<td>0.20</td>
<td></td>
</tr>
</tbody>
</table>

3. CURRENT ISL PROJECT ACTIVITY

Nuclear Assurance Corporation (NAC) maintains a database including all installed, planned and potential uranium production facilities in its Uranium Supply Analysis (USA) System. The USA System includes technical and financial information for all database projects. The following analysis of installed and planned ISL projects was done using the USA System. A summary of the 20 identified U.S. ISL projects is given in Table 4.

At present there are four operating ISL uranium projects, four shut-in projects and five planned or developing projects. Of the four producing projects; one is in South Texas, two are in Wyoming and one in Nebraska. Three of these projects were started (or restarted) since late 1987. The Crow Butte, Nebraska project is the newest project and came into production in April 1991. The capacity of the operating projects is 1,720 MTU.

With the exception of Crow Butte, each project uses satellite ion exchange equipment to recover uranium and then trucks the resin to a central plant for elution and final concentrate final production.

Both the Smith Ranch and Ruth/North Butte projects in Wyoming, are licensed for operation. Smith Ranch is scheduled to come into production in 1995, but the startup is dependent on market conditions. The startup date for Ruth/North Butte has not been announced. Startup dates for the Alta Mesa, Churchrock/Crown Point and Gas Hills projects range from 1994 to 1997, but will most probably be market dependent. There are no announced startup dates for development of the potential projects.
### Table 4

**U.S. IN SITU LEACH URANIUM MINING PROJECTS**

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Christensen/Irigaray - Malapai Resources</td>
<td>WY</td>
<td>14,800</td>
<td>770</td>
</tr>
<tr>
<td>Holiday/Ei Mequite - Malapai Resources</td>
<td>TX</td>
<td>2,540</td>
<td>300</td>
</tr>
<tr>
<td>Highland - Power Resources, Inc.</td>
<td>WY</td>
<td>6,500</td>
<td>770</td>
</tr>
<tr>
<td>Crow Butte - Ferret Exploration of Nebraska</td>
<td>NB</td>
<td>13,600</td>
<td>785</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>37,440</strong></td>
<td><strong>2,225</strong></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Installed/Not Operating mid-1992</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Kingsville/Rosita - Uranium Resources, Inc.</td>
<td>TX</td>
<td>2,390</td>
<td>770</td>
</tr>
<tr>
<td>West Cole - Total Minerals Corp.</td>
<td>TX</td>
<td>850</td>
<td>77</td>
</tr>
<tr>
<td>Hobson/Gruy - Everest Minerals Corp.</td>
<td>TX</td>
<td>230</td>
<td>77</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>3,470</strong></td>
<td><strong>924</strong></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Licensed(1)and/or with Announced Plans</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Smith Ranch(1) - Rio Algom Mining Corp.</td>
<td>WY</td>
<td>9,600</td>
<td>770</td>
</tr>
<tr>
<td>Ruth/North Butte(1) - Pathfinder Mines Corp.</td>
<td>WY</td>
<td>4,600</td>
<td>615</td>
</tr>
<tr>
<td>Alta Mesa - Total Minerals Corp.</td>
<td>TX</td>
<td>3,500</td>
<td>425</td>
</tr>
<tr>
<td>Churchrock/Crown Point - Uranium Resources, Inc.</td>
<td>NM</td>
<td>8,500</td>
<td>1,150</td>
</tr>
<tr>
<td>Gas Hills - Power Resources/UG Mining, Inc.</td>
<td>WY</td>
<td>7,700</td>
<td>385</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>33,900</strong></td>
<td><strong>3,345</strong></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Potential Projects with Producer Owners</th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Ruby Ranch - Power Resources, Inc.</td>
<td>WY</td>
<td>1,770</td>
<td>115</td>
</tr>
<tr>
<td>Leuenberger - Power Resources, Inc.</td>
<td>WY</td>
<td>1,550</td>
<td>115</td>
</tr>
<tr>
<td>Powder River - Pathfinder Minerals Corp.</td>
<td>WY</td>
<td>5,800</td>
<td>NA</td>
</tr>
<tr>
<td>Reno Creek - Energy Fuels Nuclear</td>
<td>WY</td>
<td>1,900</td>
<td>150</td>
</tr>
<tr>
<td>Benham - Albuquerque Uranium Corp.</td>
<td>TX</td>
<td>390</td>
<td>100</td>
</tr>
<tr>
<td>Vasquez - Uranium Resources, Inc.</td>
<td>TX</td>
<td>1,400</td>
<td>NA</td>
</tr>
<tr>
<td>Churchrock Option - Uranium Resources, Inc.</td>
<td>NM</td>
<td>6,900</td>
<td>NA</td>
</tr>
<tr>
<td>Big Red - Ferret Exploration of Nebraska</td>
<td>NB</td>
<td>7,300</td>
<td>NA</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>27,010</strong></td>
<td><strong>480</strong></td>
</tr>
</tbody>
</table>

**Grand Total**                                         |        | **101,820**   | **6,974**     |

---

*a. TX = Texas; WY = Wyoming; NB = Nebraska and NM = New Mexico*

### 3.1 ISL Plant Capacity and Labor Efficiency

The labor efficiency of ISL operations compares favorably with most conventional uranium production centers. In addition, labor efficiency in current ISL operations, is significantly higher than it was in 1980. Two factors are responsible for these changes. Today's ISL operations have larger capacity’s and fewer employees. Both factors have helped reduce ISL production costs.
Payroll costs of U.S. ISL projects make up about 30 percent of operating costs and also account for about 15 to 20 percent of total production costs, including capital. Payroll costs are a major cost center in all types of uranium mining activities. However the labor efficiency of ISL mining is high as compared with most conventional uranium mining operations. For example Stover [11] reports that employee productivity for the Rosita and Highland ISL projects was 6.5 and 7.5 MTU/worker-year in 1989. In comparison, employee productivity in conventional uranium mines ranged from 1.13 MTU/worker-year at Elliot Lake, to 6.5 MTU/worker-year at Ranger. Only Key Lake with 13.0 MTU/worker-year had a significantly higher productivity.

Over the last 15 years the ISL uranium mining industry has achieved higher productivity per worker by using more efficient project design and automation, as well as through economy’s of scale achieved in larger projects. The increase in the size of present day installed and planned projects is readily apparent compared to earlier projects. In 1980 there were 15 commercial ISL projects with an average annual capacity of 148 MTU. Individual annual project capacities ranged from 38 to 385 MTU. In 1992 the 12 installed and planned projects have an average annual capacity of 615 MTU. The annual capacity of these 12 projects ranges from 77 to 1,154 MTU.

During the same period the productivity of ISL project workers has also increased. In 1980 it was reported [12] that a typical ISL operation with a capacity of 193 MTU per year required 60 to 100 people. Today the Highland project produces 385 MTU per year with a staff of about 50. Smith Ranch plans to operate at up to 769 MTU per year with a staff of 65 to 75. The 1980 personnel level equates to a productivity of between 1.9 to 3.2 MTU per worker-year. Productivity at the Highland project is 7.7 MTU per worker-year, while Smith Ranch is expected to achieve a productivity of between 10.2 and 11.5 MTU per worker-year.

4. GEOLOGY AND HYDROLOGY

All of the installed and planned U.S. ISL uranium projects are located in Wyoming, Nebraska, Texas and New Mexico. Figure 2 shows the location of these areas. All of the installed and planned ISL projects will mine sandstone hosted uranium deposits. With the exception of New Mexico, the ore forming mineralization consists of uraninite and/or coffinite. A description of the regional geology of the ISL uranium mining districts follows.

4.1 Wyoming and Nebraska

The Wyoming and Nebraska uranium mineralization occurs in major rollfront-type deposits in sandstone of Tertiary age in intermontane basins of the Rocky Mountain foldbelt. The basins are filled with clastic sedimentary rocks and lie between (or as in Nebraska, adjacent) to mountain ranges, with granitic cores of Precambrian age. The basins are products of Laramide orogeny of late Cretaceous and Paleocene time. Tectonic forces were responsible for basin formation and sedimentary filling. The host basins for installed or planned ISL projects include the Wind River Basin (i.e. Gas Hills project), the Denver Basin (i.e. Crow Butte project) and the Powder River Basin for the other Wyoming projects.

The Powder River Basin is a structural basin open to the north, bounded on the south by the Laramie Range and Hartville uplift, on the east by the Black Hills, and on the west by the Big Horn Mountains and the Casper Arch. The Basin includes an area of nearly 31,000 square
kilometers and has been a prolific uranium producer. In the past several open pit and a few underground mines produced uranium, while today only the Highland and Christensen-Irigaray ISL mines are in production.

4.2 Texas

The south Texas uranium province consists of rollfront-type deposits located in sandstones that occur in a mixed fluvial-shallow marine sedimentary sequence. The province occurs on a broad flat coastal plain located along the northwest margin of the Gulf of Mexico. The sedimentary basin is located on the margin of a continental plate adjacent to a spreading ocean basin. It is located to the east of a volcanic field occupying the Big Bend region and adjacent areas.

The coastal plain is underlain by more than 15,200 meters of interbedded Tertiary marine and non-marine sediments. The depositional history of these rocks reflects inter-relationships between migrating shorelines, relative and eustatic changes of sea level, and structural deformation.
In response to sea level changes during the Tertiary, the position of shorelines in South Texas have fluctuated, with deposition gradually extending out into the subsiding Gulf. This pattern has been even more well developed since the Oligocene.

The regional geology is characterized by a series of easterly dipping continental sediments that gradually increase in thickness toward the east. The sediments are composed of major sand systems that grade laterally into clay and siltstone. Post-depositional tilting toward the Gulf of Mexico has resulted in truncation of the sediments from Eocene through Oligocene in age.

4.3 New Mexico

In New Mexico the planned ISL uranium operations are located in the northwest corner of the state in the Grants Uranium Region on the south flank of the San Juan Basin. The San Juan Basin is a large basin, about 160 kilometers by 110 kilometers, that has been the site of recurrent differential vertical tectonic movement since late Paleozoic time. The basin contains up to 3,350 meters of sedimentary rocks ranging in age from Pennsylvanian to Late Cretaceous (some volcanic intrusive and extrusive rocks are also present).

Most of the uranium deposits occur within the main sandstone bodies of the Westwater Canyon and Brushy Basin Members of the Morrison Formation of Upper Jurassic age. They are generally localized near the thickest part and in the most permeable parts of the sandstones. These fluvial sandstones were formed as alluvial fan deposits and most commonly consist of medium to fine grained feldspathic sandstones. The Westwater Canyon reaches a maximum thickness of about 90 meters.

The Grants Uranium Region has been a prolific source for uranium production by conventional methods. In this area the Morrison Formation has been the source for 98 percent of the 130,800 MTU produced in the state. More than 99 percent of New Mexico's remaining reserves of about 282,300 MTU are sandstone-type uranium deposits that occur in the San Juan Basin. There are, however, characteristics of these remaining resources that may prevent extensive production using ISL technology. Two principal factors will be critical in determining to what degree uranium deposits in the San Juan Basin may be amenable to ISL mining: ore deposit type, and depth.

While some of the uranium ore deposits are of the roll-front type, a large portion of the deposits are uniquely classified as tabular, uraniferous humate deposits. The uranium minerals are intimately associated with humate (i.e. a carbonaceous precipitate from humic acid) which coats and is intergrown with the uranium minerals. In addition most of the uranium resources of the San Juan Basin occur at depths greater than 300 meters.

4.4 Geology of Current ISL Operations

All significant ISL uranium production in the U.S. has been from paleochannel sands of Tertiary age. Planned New Mexico ISL production will be from paleochannel sands of Upper Jurassic age. While all of the planned ISL production is from rollfront-type sandstone deposits, geologic characteristics are somewhat different in each district. See Tables 5 and 6.

With the start-up of six commercial ISL uranium operations during the mid-1970s, South Texas was the only significant ISL uranium producer. The principal ISL uranium production units in Texas are the Goliad sand, Oakville sandstone, Catahoula and the Jackson Group. All current and
Table 5

<table>
<thead>
<tr>
<th>State</th>
<th>Recoverable Reserves (MTU)</th>
<th>Formation</th>
<th>Epoch</th>
<th>Age (Host rock) (10⁶ years)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wyoming</td>
<td>54,200</td>
<td>Fort Union and Wasatch, Wind River</td>
<td>Paleocene</td>
<td>38-63</td>
</tr>
<tr>
<td>Texas</td>
<td>11,200</td>
<td>Catahoula and Goliad</td>
<td>Eocene</td>
<td>5-54</td>
</tr>
<tr>
<td>Nebraska</td>
<td>20,900</td>
<td>Chadron</td>
<td>Oligocene</td>
<td>24-38</td>
</tr>
<tr>
<td>New Mexico</td>
<td>15,400</td>
<td>Morrison</td>
<td>Upper Jurassic</td>
<td>138-205</td>
</tr>
</tbody>
</table>

Table 6

<table>
<thead>
<tr>
<th>State</th>
<th>Depth Range, Meters</th>
<th>Characteristics</th>
<th>Special Problems</th>
</tr>
</thead>
<tbody>
<tr>
<td>S. Texas</td>
<td>60-245</td>
<td>- High permeability</td>
<td>- Low levels of Mo in some ores</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Thicker ore intercepts</td>
<td>- Faulting</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Clean sands</td>
<td>- Saline water in some areas</td>
</tr>
<tr>
<td>Wyoming &amp; Nebraska</td>
<td>60-300</td>
<td>- Lower permeability</td>
<td>- Varying levels of vanadium</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Thin, high grade ore intercepts</td>
<td>- Insufficient groundwater levels in some cases</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Good ground water quality</td>
<td>- Winterization required and remote locations</td>
</tr>
<tr>
<td>New Mexico</td>
<td>245-760</td>
<td>- Intermediate to high permeability</td>
<td>- Deep ore</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Very thick host sand (30m plus)</td>
<td>- High Mo concentration in some areas</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Multiple mineralized intervals in one sand</td>
<td>- High humate concentrations in some areas</td>
</tr>
</tbody>
</table>

planned ISL production will be from the Goliad and Catahoula Formations. They are characterized by clean, very well sorted sands with high permeability. Rollfronts are characterized by thick ore intercepts. South Texas has been the source of about 85 percent of the uranium produced by ISL mining. Geologic favorability has been a major factor contributing to this. However, identified south Texas reserves of 11,231 MTU are limited and producers have gradually sought new districts in Wyoming, Nebraska and New Mexico where large ISL amenable uranium reserves are known to occur.

With the exception of one planned project in the Gas Hills District, all installed and planned Wyoming ISL projects are located in the Powder River Basin. Here the principal uranium bearing units are the Wasatch and Fort Union Formations, respectively of Eocene and Paleocene age. Results of ISL test work indicate that, while some areas of the Powder River Basin may not be amenable for ISL operations, much of the Basin has favorable geologic properties.

The Gas Hills uranium deposits are rollfront-type and are hosted by fluvial sandstones of the Eocene Age Wind River Formation. Information regarding the ISL amenability of these deposits is very limited as the one ISL project in the Gas Hills District is in the early planning stage.

In general, the Wyoming uranium deposits exhibit lower permeability than do South Texas deposits. This characteristic may restrict economic ISL production of some Wyoming uranium deposits. Most of the identified uranium resources occur at depths that should be amenable for ISL operations. As compared with Texas deposits, the ore is characteristically thick and high grade. Rollfronts frequently occur in more than one sand unit, distributed one above the other, named “stacked” ore. The rollfronts may be narrow and rapidly change direction.

The opportunity for successful ISL uranium mining in Wyoming is best illustrated by the operation of the Highland project. The project has produced over 1,540 MTU since its start-up in January 1988. Wyoming has identified ISL amenable reserves of 54,231 MTU, or 53 percent of the U.S. total.

Substantial reserves of ISL minable uranium have been identified near Crawford, in northwestern Nebraska. In the combined Crow Butte and Big Red projects, up to 20,923 MTU reserves are reported to occur in the Tertiary Basal Chadron member of the White River Group. This is 21 percent of the total U.S. ISL amenable reserves.

In the Crow Butte project area, 13,615 MTU are reported. The ore averages 0.9 to 4.6 meters in thickness and occurs at a depth between 185 and 256 meters. Permeability is high and the principal geologic and hydrologic factors that effect ISL operations are reported to be favorable [13]. Results of two pilot tests and over 18 months of Crow Butte production demonstrate the ISL amenability of these deposits.

Holen and Hatchell have evaluated the ISL potential of New Mexico’s uranium deposits and report the following conclusions:

"The Morrison Formation, and to a lesser extent, the Dakota Sandstone, account for the bulk of the reserves amenable to exploitation by ISL. The Todilto Limestone, which has accounted for about two percent of New Mexico’s uranium production, is probably unsuitable for ISL production."
Two general types of deposits occur in the Morrison Formation: primary and redistributed. Uranium in primary deposits is coextensive with an amorphous high-carbon organic material commonly called humate. Although specific leach effectiveness data are lacking, the association with humate results in a reduction in host rock permeability and in uranium mobilization that would be expected to have an adverse effect on recovery. In contrast, the ratio of humate to uranium in redistributed deposits is highly variable and in some deposits humate is virtually absent. In many respects redistributed deposits are similar to the roll-type deposits that have been exploited successfully by ISL in Texas and Wyoming.

About 83 percent of the remaining reserves in New Mexico are at depths exceeding 1,000 feet (305 meters) and extend to depths over 4,000 feet (1,220 meters), but most of the more amenable redistributed deposits are at depths of 2,000 feet (610 meters) or less. Primary ore is the dominant ore type in most areas except Church Rock where redistributed ore is dominant. Subequal mixtures occur in deposits at Crownpoint. There has been no commercial-scale ISL production in New Mexico and recovery factors at several pilot operations are largely unknown. Mobil’s South Trend Development Area project at Crownpoint has been the most extensively tested and is reportedly considered to be successful from the standpoint of recovery as well as groundwater restoration. The Crownpoint deposits are at a depth of 2,000 feet (610 meters) compared to depths of less than 800 feet (244 meters) for Texas and Wyoming deposits." [14]

Uranium Resources, Inc. plans to bring New Mexico’s first commercial ISL uranium mine into production near Churchrock and Crownpoint. The area hosts large reserves of relatively shallow ore where ISL pilot tests have been completed. Reserves associated with the planned Churchrock/Crownpoint and potential Churchrock Option projects are 15,385 MTU, or 15 percent of the total identified U.S. reserves. Reserves at the initial Churchrock site are at a depth of 245 meters. Additional reserves are at a depth of 300 meters or more.

4.5 Factors of Geologic and Hydrologic Favorability

The two most important factors in determining the economic feasibility of any ISL project are the flow rate per well and the concentration of uranium in fluid produced from the well. Given an optimal design of the ISL mine system, the geology then becomes the fundamental control of what flow rates and uranium head grades are achievable.

The success of the current U.S. ISL uranium industry depends on the ability of the project operators to define favorable geologic environments that consistently provide physical and chemical conditions amenable to economic ISL recovery using alkaline leaching systems. The industry currently mines only rollfront-type sandstone uranium deposits. These deposits are uniquely amenable to ISL exploitation since ISL mining relies on physical and chemical processes similar to those that originally deposited the uranium orebodies.

Geological requirements for ISL mining are:
- Orebody located below water table
- Uranium mineral amenable to oxidative dissolution with proposed leaching system
- Permeability that will permit required flow, with a minimum in the .15 to .37 meters per day range
- High correlation of permeability to uranium ore to allow intimate contact of leachant to ore
- Overlying and underlying continuous permeability barriers for fluid confinement
- Groundwater with chloride content less than 2.5 grams/liter
These elements define the minimum requirements for ISL operation. The more successful ISL operations exploit orebodies that have the most favorable ISL amenable characteristics. Several characteristics that improve the amenability for ISL mining are:

- Rollfront-type deposit, continuous and wide (width not less than 30 meters)
- High average grade with minimum thickness of 1 meter or more and high Grade times Thickness (GT) product
- High permeability ranging up to 3.7 meters per day or more
- Artisan water table with minimum hydrostatic head of about 15 meters
- Depths from 60 to 180 meters
- No by-product metals

Geologic characteristics that may prevent economic development include:

- Excess presence of unfavorable gangue minerals (metal sulfides, calcite, organics, clay, etc.)
- Uranium mineralization encapsulation in clays or silts
- High molybdenum or vanadium concentration
- Thin, sinuous, and deep mineralization
- Poor vertical solution confinement
- Highly faulted or dipping formation

The presence of groundwater around the orebody is critically important to ISL mining of uranium. The aquifer water has four functions in the leaching/restoration process:

- Forms leach solution
- Moves the leach solution within the deposit
- Allows control of the leach solution, including control of possible solution excursion outside of the operating wellfields.
- Natural flow of the aquifer helps restore the chemical properties of the aquifer and host formation once leaching is terminated.

The development of methods to control wellfield fluids is one of the advances that has made current ISL operation successful. Today greater care is taken to assure that wells are carefully constructed to prevent vertical migration of solutions along the well bore. Regulators now require that the integrity of each well be tested prior to operation, thereby substantially reducing the risk of leakage. The risk of lateral excursions from the wellfield has also been reduced. This is done by pumping between one and three percent more fluid than is injected, thereby inducing a net flow of ground water into the wellfield.

Some of the early ISL operations were attempted where the groundwater level extended only a few meters over the orebody. Today all operators are mining orebodies with a head of 15 meters or more over the orebody. Selection of orebodies located well below the water table provides several benefits. Less control is required to assure that pumping does not excessively lower water levels. Higher flow rates can be achieved under conditions of high water pressure. Higher water pressures permit an increased concentration of dissolved oxygen, which is used as the oxidant in all current operations.

Formation permeability is the most significant control of wellfield flow rate. While ISL mining may be carried out with low permeability, successful operations require higher permeabilities that permit rapid movement of fluids through the aquifer. The minimum required permeability is in the .15 to .45 meters per day range. Several of the Texas ISL operations have been carried out in aquifers where permeabilities are .75 meters per day or greater.
Table 7
Hydrological Criteria for ISL Extraction

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Character/or Reported Values</th>
<th>Effect of ISL Extraction</th>
</tr>
</thead>
<tbody>
<tr>
<td>Permeability of production zone (meters/day)</td>
<td>0.22-7.5</td>
<td>High production rate requires high permeability</td>
</tr>
<tr>
<td>Type of groundwater occurrence</td>
<td>Below water table: artisan; some water table</td>
<td>Feasibility</td>
</tr>
<tr>
<td>Position of ore zone below water level</td>
<td>Normally 15 to 75 meters or more</td>
<td>Oxidation/production rate increases with water pressure</td>
</tr>
<tr>
<td>Confining strata present above and below ore aquifer</td>
<td>Normally required</td>
<td>Feasibility; restrict solution to production zone</td>
</tr>
<tr>
<td>Horizontal continuity or ore aquifer</td>
<td>Poor to excellent</td>
<td>Feasibility/production rate</td>
</tr>
<tr>
<td>Geologic structure of strata</td>
<td>Gently dipping</td>
<td>Simple structure enhances feasibility</td>
</tr>
<tr>
<td>Baseline water quality</td>
<td>Water quality standards</td>
<td>Restoration guidelines less difficult for low quality; very poor quality will interfere with leaching</td>
</tr>
<tr>
<td>Well efficiency</td>
<td>Low to 90%</td>
<td>Flow/production rate improves with high efficiency</td>
</tr>
<tr>
<td>Status of drill hole plugging or status of other man-made conduits</td>
<td>Holes should be plugged. Other openings should not enter ore zones</td>
<td>Open holes or mine workings may result in excursions or make solution control difficult or impossible</td>
</tr>
<tr>
<td>Regional influence on groundwater operations use</td>
<td>Normally located in remote areas but conflicting water use may occur</td>
<td>Water use conflicts may restrict or prevent operation</td>
</tr>
</tbody>
</table>

Most of the present day operations are developed in orebodies with high flow rates. If projects with lower permeabilities are to be successful, it is necessary to have consistently high solution grades. High solution grade can make up for low flow rates.

Most Tertiary age sandstones that host rollfront-type uranium ores exhibit high permeability. These sandstones are characteristically moderately compacted and have relatively little cement between the sand grains. Permeability decreases as compaction occurs or as intergranular cements have been introduced. These features are common in older formations that have sustained longer periods of burial, often at increased depth. Well compacted and/or cemented formations may therefore not be amenable to ISL mining. See Table 7 for a summary of hydrogeologic criteria for ISL mining.
Orebody depth is another factor that has a significant impact on ISL amenability. In the past most ISL uranium mines have operated at depths of 60 to 150 meters. Although most production is still coming from moderate depths several current projects have some portion of their orebodies located as deep as 210 to 290 meters.

As operators turn to new projects, such as the extensive New Mexico reserves, ore depths will increase to between 300 and 760 meters. Exploration, development and construction costs increase rapidly with depth. Operating costs will be higher because of increased pumping costs. It is known however, that the efficiency of uranium dissolution rises with increasing pressure because of the increasing solubility of oxygen in water. It is also argued that it is possible to increase pressure differentials between injection and production when operating at greater water pressures. This is believed to improve fluid velocities and to allow greater distances between wells.

As early as 1980, Hunkin argued that the high cost of well completion at deeper levels is more than offset by greater efficiency of leaching and greater separation of injection and production wells [15]. The technical feasibility of ISL uranium mining at depths of 610 meters has been demonstrated by Mobil’s pilot test on Section 9 at Crownpoint, New Mexico. There is however, insufficient information about the tests to indicate the economic viability of ISL mining at these depths [16]. The fact that Mobil has abandoned the project and returned the properties to the original owners suggests that a high uranium price will be necessary to justify this project.

4.6 Ore Reserves

The ore reserves of the 20 identified ISL projects are in most cases in the proven and probable (RAR) category and have been adjusted to account for a 75 percent recovery. Recoverable reserves for the currently operating and shut-in ISL projects are about 40,925 MTU. See Table 4. Licensed and planned projects add an additional 33,845 MTU. Potential projects add another 27,000 MTU. It should be noted that all of the potential projects are owned by companies still active in the uranium industry. Total ISL recoverable reserves are 101,770 MTU.

Ore reserves in currently operating or shut-in projects have the following characteristics:
- Average uranium Grade times Thickness Product (GT) ranges from 0.55% ft. (0.14% m) to greater than 2.0% ft. U₃O₈ (0.5% m U).
- Average thickness ranges from 0.9 to 4.5 m
- Average grade ranges from 0.04 to 0.26% U
- Individual project reserves vary from 230 to more than 14,800 MTU
- Average ore depths of less than 300 m

These parameters are also generally typical of the planned and potential projects. However, the average grades are in nearly all cases closer to .08% U than to .04% U. The average ore thickness is typically 3 to 4.5 m. With the exception of the New Mexico deposits the average ore body depths are less than 300 m.

The distribution of the ISL recoverable reserves by state is given in Table 8. The distribution of recoverable reserves is 53, 11, 15 and 21 percent respectively, for the states of Wyoming, Texas, New Mexico and Nebraska. Table 9 provides an insight into the relative significance of ISL amenable reserves in the U.S. For comparison the table gives the U.S. Energy Information Administration’s 1990 estimate of all reserves with a forward production cost of $30/pound and
### Table 8

**PRODUCTION CAPACITY AND RECOVERABLE RESERVES BY STATE**  
(MTU)

<table>
<thead>
<tr>
<th>State</th>
<th>Capacity</th>
<th>Percent</th>
<th>Reserves</th>
<th>Percent</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wyoming</td>
<td>3,692</td>
<td>53%</td>
<td>54,229</td>
<td>53%</td>
</tr>
<tr>
<td>Texas</td>
<td>1,750</td>
<td>25%</td>
<td>11,230</td>
<td>11%</td>
</tr>
<tr>
<td>New Mexico</td>
<td>1,154</td>
<td>17%</td>
<td>15,384</td>
<td>15%</td>
</tr>
<tr>
<td>Nebraska</td>
<td>385</td>
<td>6%</td>
<td>20,922</td>
<td>21%</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>6,980</strong></td>
<td><strong>100%</strong></td>
<td><strong>101,765</strong></td>
<td><strong>100%</strong></td>
</tr>
</tbody>
</table>

### Table 9

**ISL RECOVERABLE RESERVES AND TOTAL RESERVES BY STATE**  
(MTU)

<table>
<thead>
<tr>
<th>State</th>
<th>Forward Cost Reserves¹</th>
<th>ISL Recoverable² (Installed and Planned)</th>
<th>Percent</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>$78/kgU ($30/Lb)</td>
<td>$130/kgU ($50/Lb)</td>
<td></td>
</tr>
<tr>
<td>Wyoming</td>
<td>27,307</td>
<td>102,050</td>
<td>54,230</td>
</tr>
<tr>
<td>Texas</td>
<td>8,846</td>
<td>18,077</td>
<td>11,220</td>
</tr>
<tr>
<td>Nebraska</td>
<td>NA</td>
<td>NA</td>
<td>20,923</td>
</tr>
<tr>
<td>New Mexico</td>
<td>32,692</td>
<td>135,000</td>
<td>15,385</td>
</tr>
<tr>
<td>Others</td>
<td>33,077</td>
<td>76,154</td>
<td>0</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>101,922</strong></td>
<td><strong>255,203</strong></td>
<td><strong>101,768</strong></td>
</tr>
</tbody>
</table>

2. This report
$50/pound U3O8 ($78 and $130/KgU) for each state[17]. ISL amenable reserves of the 20 identified projects described above account for 29 percent of the total reserves with a forward production cost of $50/pound U3O8 ($130/KgU).

4.7 Resource Estimation for ISL Mining

As indicated above ISL amenable uranium ore reserves must meet several well defined criteria. ISL recoverable ore reserves estimates must take into account the highly selective nature of the ISL process. ISL mining can only extract those resources that lie directly beneath or within a few tens of feet from each well pattern, and within or immediately above and below the screened well interval. Only those resources exposed to direct contact with the leaching fluid can be recovered.

Ore mineral grains occurring in zones of low permeability are not leached during normal operations. This includes uranium minerals in very fine grained rocks, such as clay and siltstones, or zones where clays or mineral cements surround the ore minerals. Ore reserve calculations should discount resources occurring in these environments.

Initial ore reserve calculations for ISL projects have been made using a variety of estimation techniques. However, in the U.S., detailed well pattern, or wellfield, estimates are usually calculated using some type of Grade x Thickness (GT) contour method.

Today most ISL projects are designed using a recovery factor of 65 to 75 percent. The recovery factor is defined as the amount of uranium recovered compared to the amount of uranium in the reserve estimate. Over the years, the recovery factor for ISL uranium mining has been subject to much discussion. The low recovery achieved in some of the early projects were partly to blame for this uncertainty.

Actual recoveries are difficult to document. However, various studies reported in the literature provide some insight. Everest Minerals Corporation reported an overall recovery of only 27 percent from their first project completed in 1982 (Hobson Project in Karnes County, Texas). They indicate that several geological, hydrological and geochemical factors were responsible for this low value [17]. In particular, the uranium was restricted to low permeability sands, channelization of leaching solutions occurred and an appreciable amount of uranium occurred in a mineral phase that is difficult to dissolve.

More recently, Everest and other operators have been able to achieve much higher average recoveries. In 1991, Power Resources, Inc., (PRI) operator of the Highland project, reported that they were recovering an average of 86 percent of the calculated reserves at the Highland Project [18]. In a report entitled "In Situ Leaching of South Texas Uranium Ores Part 3 - Post Leach Assessment of Recovery and Sweep Efficiency", Mobil indicated that overall recovery from a 12 m thick mineralized zone was 70 percent based on analysis of a core drilled after leaching was complete [19].

It should be noted that under conservative estimation practice only those reserves are included that fall within the boundaries defined by the 5-spot well patterns. No allowance is given for mineralization swept by leaching fluids but located outside of limits defined by straight lines connecting the injection wells. Other estimation methodologies assign some reserves to this area.
Based on a review of current and past operations, it is concluded that a properly designed wellfield will recover a minimum of 65 to 75 percent of estimated reserves. This assumes that the reserve estimate is made using appropriate considerations for ISL amenability of the ore. Recovery may be higher if a conservative approach is used in estimating reserves. This would include deducting all reserves in zones of low permeability and restricting the ore inventory to the area within the boundaries of the well patterns.

5. FUTURE OUTLOOK FOR THE U.S. ISL URANIUM INDUSTRY

There are twelve licensed and planned ISL uranium projects in the U.S. See Table 4. The estimated forward cost of production for these projects ranges from about $26 to $52 per KgU. With a total planned annual capacity of 6,346 MTU and recoverable reserves of about 74,615 MTU, ISL mining should be the dominant U.S. uranium production technology during the next 10 to 15 years. This capacity could be increased by 20 to 30 percent through the development of 8 additional potential projects with identified uranium reserves of about 26,923 MTU. The total reserves associated with NAC's identified ISL projects include 101,769 MTU.

NAC's reserve estimate may be compared with the total estimated ISL amenable reserve base of the U.S. Based on its 1990 annual survey of all industry participants [20], the U.S. Energy Information Agency (EIA) reported an estimated 127,308 MTU producible by ISL technology at a Forward Cost of up to $50 per pound U3O8 ($130/KgU). This includes 32,308 MTU producible at a forward cost of up to $30 per pound U3O8 ($78/KgU). NAC's inventory of ISL projects equals 80 percent of the U.S. ISL reserves producible at a Forward Cost of up to $50 per pound ($130/KgU) and therefore includes most of the identified ISL reserves in this cost category.

While there are substantial uranium reserves amenable to ISL mining in the U.S., there is an apparent practical limit to ISL uranium mine development because of the finite limit of these reserves. As shown in Table 9, based on its industry wide survey the EIA reports an estimated 356,154 MTU of reserves recoverable at a Forward Cost of up to $50 per pound U3O8 ($130/KgU) using all mining methods. ISL recoverable reserves make up about 36 percent of this total. While some additional portion of the national reserve base will be minable using ISL technology there is a practical limit because of the stringent geologic and hydrologic requirements for economic production.

The probable future dominance of ISL mining of the U.S. uranium production industry becomes even more clear when considered in perspective with the decreasing capacity of conventional projects. This year's permanent closure of the Shirley Basin and Rhode Ranch open pit mines and the start of decommissioning of the Shirley Basin and Panna Maria mills marks the end, at least for the time being, of the U.S. open pit uranium mining industry. The closures remove 992 MTU per year of U.S. uranium production capacity. At present there are no plans for developing new open pit mines in the U.S.

The future of uranium production from underground mining is somewhat more positive. However, with the exception of the relatively high grade breccia pipe hosted deposits of the Arizona Strip there is relatively little potential for development of underground mines amenable to lowcost production. The Green Mountain project may be one possible exception to this case. While NAC's database includes eight existing or planned projects with a cumulative annual capacity of 6,769 MTU, most of these projects will not be able to compete in a market with uranium selling for $52 per KgU or under.
The ongoing decommissioning of uranium mills motivated by the increasingly restrictive regulatory requirements of the U.S. Environmental Protection Agency (EPA), as implemented by the U.S. Nuclear Regulatory Commission (NRC), will further diminish the potential for reactivation of shut down and development of new conventional uranium mining projects in the U.S. In addition, the added cost of meeting new, lower radiation exposure limits for workers, particularly in mines with medium to low average grade, may make it uneconomic to develop new underground mines.

The future of U.S. ISL uranium production therefore depends not only on the economic advantages of the technology, but also on the relatively small amount of uranium resources amenable to lowcost conventional production.

References


1. INTRODUCTION

In 1969 the then SDAG Wismut started underground uranium leaching in East Germany. Among the reasons for this venture were rising costs of conventional mining and milling procedures, low-grade ore reserves as well as increasingly complicated geological and mining conditions.

Since the end of world war 2, some 250 000 t of uranium were mined in East Germany from different deposits (production amounted to 216 500 t):

- Hydrothermal deposits in Ore Mountains/Vogtland 103 000 t
- Sedimentary metamorphose Paleozoic deposit Ronneburg 113 000 t
- Carbonate Zechstein deposit Culmitzsch 12 000 t
- Lower Permian coal deposit Dresden-Freital 4 000 t
- Cretaceous sedimentary deposit Königstein 19 000 t

Since 1968 leach mining by Wismut produced a total of 8 930 t:

5 526 t by in-situ underground leach mining (including slime leaching) at Königstein
596 t by heap leaching at Königstein
2 716 t by heap and waste pile leaching at Ronneburg
approx 90 t by in-situ underground leach mining at Ronneburg.

In 1990, the year in which uranium production was shut down in East Germany, leaching produced 384 t, accounting for 13% of total annual production.

2. IN SITU LEACHING OF SEDIMENTARY ORES AT KÖNIGSTEIN

2.1 Characteristics of the deposit

The deposit is located in the Elbsandsteingebirge mountains (Fig.1) near the town of Dresden. It had a surface of 25 km² of which only 5 km² were developed. Mineralization occurs in
FIG. 1. Mining and milling sites operated by Wismut Corporation.

FIG. 2. Geology of the Königstein deposit.
FIG. 3. Mineralization structure in the Königstein deposit.

FIG. 4. Königstein deposit: hydrogeology.
depths between 150 and 250 m in Cretaceous sediments superposing granites and generally trending S-N at approx 3°. Pitchblende occurs in paragenesis with pyrite, marcasite, and hematite in terrestrial and marine Cenomanian and Turonian sandstones (Fig. 2, 3).

There are three seam-like ore horizons:

i Marine Cenomanian sandstones
ii Cenomanian lagoonal argillaceous sandstones
iii Cenomanian terrestrial sandstones and siltstones

Uranium was mined in the dewatered fourth aquifer. The hanging aquiclude separating the mine from the third aquifer was intersected by mine work. This third aquifer provides water supply in the Elbe valley. Its water table was lowered as well (Fig. 4). The Elbe river to which part of the groundwaters discharge is at a distance of 600 m.

The deposit is marked by intense tectonic disturbance. Minable horizons vary from 2 - 10 m forming a roll-front in their central part. The U-Ra ratio is very irregular due to secondary redeposition by ground waters. Average uranium content in geological reserves was 840 g/t. A total of 19 258 t were mined.

2.2 Leaching technology

Underground leaching of low-grade uranium ores of < 200 g/t and subsequently of even lower-grade ores of 50 - 100 g U/t and conventional mining were running parallel from 1971 to
FIG. 6. In situ leaching.

FIG. 7. Leaching block in Königstein.
1984. By then, leaching had supplanted conventional mining. It was a block-type leaching using the following processes:

1) Pillar pressure leaching
2) Block pressure leaching
(Fig. 5)
3) Hydrodynamic infiltration leaching
4) ISL (Fig. 6)

Hydrodynamic infiltration was the most widely used process in block leaching. Its initial phase was the stoping of a free space for the blasting of the rock. The mined rock (17% of the volume of the leaching block) was leached above ground.

Following the blasting operations, injection drifts were driven in the hanging wall and a number of sealed boreholes were drilled for the injection of leach liquor at a pressure of 0.6 MPa. An extraction system comprising dams, mine openings, and wells was established at the bottom. Dispersion of solution beyond the block was checked by a specific monitoring system. Individually fed boreholes facilitated selective leaching of different sections of the block in accordance with uranium content and leaching progress (Fig. 7).

Average concentrations of the leach liquor were 2 - 3 g H$_2$SO$_4$/L, pH 1.5 - 1.8, salinity 10 - 14 g/L. Maintaining a liquid-solid ratio of 3 - 8 m$^3$/t, an irrigation rate of 0.003 - 0.1 m$^3$/t . d allowed to recover 65 - 75% or even more of the uranium reserves within three years (Fig. 8).

### Table: Parameters of in situ leaching Königstein

<table>
<thead>
<tr>
<th>Technology</th>
<th>Ore layer</th>
<th>Boreholes</th>
<th>Blasted Rock</th>
<th>Explosives</th>
<th>Output C</th>
<th>Liquid-solid Ratio</th>
<th>Time</th>
<th>Irrigation Rate</th>
<th>Permeability</th>
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<tbody>
<tr>
<td>Pillar Pressure Leaching</td>
<td>I</td>
<td>90</td>
<td>0</td>
<td>0</td>
<td>70</td>
<td>7</td>
<td>700</td>
<td>0.1</td>
<td>&gt; 5 x 10$^{-4}$</td>
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<td>Block Pressure Leaching</td>
<td>I</td>
<td>300</td>
<td>810</td>
<td>1.7</td>
<td>70</td>
<td>8</td>
<td>1000</td>
<td>0.008</td>
<td>&lt; 5 x 10$^{-5}$</td>
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<td>Hydrodynamic Infiltration Leaching</td>
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<td>200, 400, 320</td>
<td>0, 810, 470</td>
<td>0, 1.7, 1.8</td>
<td>70, 70, 65</td>
<td>1,5, 4,5</td>
<td>375, 750</td>
<td>0.004, 0.006</td>
<td>&lt; 5 x 10$^{-4}$, &lt; 5 x 10$^{-6}$</td>
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<tr>
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<td>0</td>
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<td>2,6</td>
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<td>121</td>
<td>0</td>
<td>0</td>
<td>75</td>
<td>6</td>
<td>1000</td>
<td>0.006</td>
<td>&gt; 10$^{-5}$</td>
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<tr>
<td></td>
<td>III</td>
<td>160</td>
<td>0</td>
<td>0</td>
<td>35</td>
<td>6</td>
<td>600</td>
<td>0.01</td>
<td>&gt; 5 x 10$^{-6}$</td>
</tr>
</tbody>
</table>

FIG. 8. Parameters of in situ leaching Königstein.

2.3 Processing

Dissolved uranium in the pregnant liquor was 10 - 150 mg/L. The liquor passed through an intermediate processing stage to increase diuranate concentrations to 60 - 150 g/L U in the solids. Uranium and competing ions like iron were stripped by ion exchange resins. The barren liquor was fortified with sulfuric acid and recirculated (Fig. 9). After multistage resin recovery, iron hydroxide was precipitated by primary precipitation and then piped to slime leaching. Concentrate pulp was
produced by secondary precipitation followed by mechanical thickening. This intermediate product was then shipped for further processing to yellow cake to the Seelingstädt processing facility 160 km away.

2.4 Problems related to decommissioning and rehabilitation

At present, 800 000 m$^3$ of sulfuric acid liquor of 1.2 - 1.7 g/L are still circulating in the developed part of the deposit. Only 30% of that part is still accessible from underground workings.
FIG. 10. Schematic geological cross-section through the Ronneburg area.

FIG. 11. Technological variations of leaching in the Ronneburg uranium mining district.
The rock pore volume contains an additional 1 million m$^3$ of liquor. The mine is to be flooded. The following preventive measures are imperative to prevent discharge of contaminated ground waters into aquifers and nearby receiving streams:

- preservation and sealing of aquiclude
- neutralization of leach liquor
- removal or underground inerting of already dissolved or leachable uranium
- rinsing, squeezing or neutralization of pore waters
- treatment of contaminated ground water.

Only then will the mine be gradually flooded; flooding will be controlled and comprehensive-ly monitored. Treatment and disposal facilities will be available as required.

FIG. 12. Combined leach and conventional mining.
3. IN-SITU LEACHING OF SEDIMENTARY METAMORPHOSE ORES AT RONNEBURG

3.1 Characteristics of the deposit

The Ronneburg ore field covers some 160 km². Mineralization is locked to metamorphosed Ordovician, Silurian and Devonian sediments and eruptives. Magmatic gangue complexes are intruded into heavily disturbed argillaceous, sandy, and carbonate strata series and are superimposed by wedges and overfolding tectonics (Fig. 10). Fifty one square km of the ore field were developed.

FIG. 13. Liquor circulation system in Ronneburg.
Uranium mineralization was primarily locked to sapropelites and coalified sediments. Geochemical rock alterations have occurred since the Permian age. For that reasons epigenetic mineralizations are occurring in association with pyrite, carbonate minerals and organic matter at fault disturbances, joint systems and along boundaries of the bed. Total reserves of the deposit amounted to > 157 000 t U₃O₈ at an average grade of 970 g/t.

Major quantities of residual reserves and ore losses were left behind in the partly fissured mine workings. This holds true in particular of 1 billion m³ of fissured zones stemming from initial roof fall exploitation.

3.2 Leaching technology

Since 1970 there were repeated attempts at increasing U recovery from the Ronneburg ore field. In addition to ongoing heap and waste pile leaching, underground leach mining was to be used (Fig. 11).
What then was called combined recovery started with stope development in high-grade ore. The developed cavity served as compensation storage for the blasting of low-grade ore from the hanging wall (Fig. 12).

Leach liquor was injected either from a top heading or from surface or underground boreholes. Depending on the geochemical composition of rock and geological and mining conditions, the process used sulfuric acid or alkaline liquor and infiltration, hydrodynamic or hydrostatic leaching using a barrier of pressure in the hanging wall (Fig. 13). From 40 - 70% or uranium reserves were recovered at concentration of 3 - 10 g/L $\text{H}_2\text{SO}_4$, pH 1.5 - 2.5 and approx. 7 - 20 kg $\text{H}_2\text{SO}_4$/t of ore. Average uranium content in the pregnant liquor was 20 - 100 mg/L.

Attempts were also made at leaching oxidized ore losses by borehole infiltration from no longer accessible former stopes. The liquor collected at lower level had partly considerable uranium concentrations (Fig. 14).

3.3 Processing

The pregnant liquor was piped to an above ground sorption unit. The enriched ion exchange resin was then shipped to Seelingstädt for further processing. At some remote leaching blocks underground sorption tests were carried out using modified ore carts filled with resin.

As a precaution and for environmental reasons, the blocks were washed with water at the end of the leaching process in order to limit the discharge of contaminants at the time when the water gable would be restored.
IN SITU LEACHING OF URANIUM IN NORTH BOHEMIA

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Abstract

The paper describes the history of the in situ leaching (ISL) technology for the exploitation of uranium ores in Czechoslovakia, in Northern Bohemia. The ISL exploitation started in 1969 and today covers an area of ca. 600 ha. Geological and hydrogeological conditions are described below. The development of the technology and the present state of ISL is also described and the aspects of the following topics are discussed: the well completion and types of pattern, the consumption of chemicals, the composition of leach solutions, the solution processing and the effectiveness of the leaching process. The technological research is the first step to wellfield and technology planning. The results of laboratory and field research are applied in the fluid flow and geochemical modeling. The results of the exploitation of uranium deposit and further prospects of the uranium mining are presented.

1. INTRODUCTION

The "in situ leaching" (ISL) technology of the uranium recovery has been performed in Czechoslovakia for 25 years. The pilot plant experiments with in situ leaching were performed in 1967, and in the following year the first leaching field was put into operation. Afterwards the rapid progress of this mining method followed in parallel with research work and the gradual solution of technical and technological problems. During this period extensive experience with the planning and operating of the ISL technology was obtained. In the course of this application it was necessary to develop a theory of the relevant underground processes, to work out the methods of the geotechnological evaluation of the deposit and to solve a great number of specific technological problems. This led to a certain degree of stabilisation of the drilling technology and also of the methods of processing the recovery solutions on the surface. At present the leaching fields occupy an area of about 6 000 000 m² and there are about 7 000 technological wells and numerous monitoring wells within that area. The importance of environmental aspects has intensified in recent years.

2. GEOLOGICAL CONDITIONS

The ISL is applied in the Stráž deposit in North Bohemia. The deposit is situated in sedimentary rocks of the so-called Stráž block of the Bohemian Cretaceous basin which contains a significant reservoir of drinking water in the European range. The Stráž block is a tectonically limited unit inside of the Cretaceous basin in which several uranium deposits are situated. The Hamr deposit, which is exploited by classic underground mining, is situated in the neighbourhood of the Stráž deposit.

The thickness of the Cretaceous formation in the Stráž block is from 200 to 250 meters. The basement consists mainly of grained rocks. The Cretaceous sedimentary complex is formed by two stratigraphic units. The lower, the Cenomanian formation contains the uranium ore mineralization. On its basis the freshwater sediments are locally developed (in depressions of the paleorelief). They are overlain by the washout sediments with a thickness of 0.5 to 5.0 m. These sediments are formed by changing layers of different composition, grain size and permeability (breccia, sandstones, siltstones, often with admixtures of organic matters and pyrite). They often
contain uranium ore mineralization. Above these there is a stratum of the friable sandstones (thickness 20 - 25 m). These sandstones are very good permeable and uranium mineralization is located in their basal part, too. The upper part of the Cenomanian sequence is formed by more compact and less permeable fucoid sandstones (thickness 40 m). The basis of the Turonian sequence is formed by a 60 m thick formation of poorly impervious siltstones and marly limestones. Above them there are permeable thick-bedded sandstones which often reach to the surface (see Fig. 1).
The most economically important uranium minerals and uranium-containing minerals of these ore deposits are uranium oxides (uraninite $UO_{2+x}$), ningyoite $(CaU(PO_4)_{2}.nH_2O)$, and hydrozircon $(Zr(Si_{4-x}O_{4-x})(OH)_{4x}.nH_2O)$.

The size of the individual grains of uranium minerals usually does not exceed 1 to 2 $\mu$m. The mutual ratios of these minerals change in the area. The likability of uranium is directly connected with this. The technological properties of the ore are unfavourably effected especially by the content of hydrozircon and often by accompanying metacolloidal baddeleyite.

In the Strâz block there are two divided groundwater levels. The upper aquifer is bound by good permeable sandstones of the middle Turonian and is a significant reservoir and source of drinking water. These waters are of the bicarbonate calciferous type. The groundwater level of the Cenomanian aquifer is characterized as artesian. The confining layer is formed by siltstones of the lower Turonian. From the hydrochemical point the Cenomanian waters in their natural state are not affected by mining interventions. They differ from the waters of the Turonian sequence by containing of $^{226}$Ra. The safety standard for drinking water is exceeded about one hundred times.

The most permeable rocks of the Cenomanian are the friable, poorly cohesive sandstones with the coefficient of filtration $k_p = 2$ to 8 meters per day. Due to the influence of their detailed stratification these rocks show a characteristic anisotropy of permeability. The petrographically more varied washout sediments with alternating small layers of rocks with different grain sizes have even higher anisotropy but lower permeability. The value of the filtration coefficient here is in tens of meters per day similar to fucoid sandstones.

3. TECHNOLOGY OF ISL

The results of laboratory research and the parallel knowledge obtained from the operation showed very quickly, that the conditions for underground leaching in the Strâz deposit were extremely difficult. The part of uranium mineralization occurs in poorly permeable rocks, often in contact with permeable layers. There are usually good leachable ores which are leached unfortunately only under diffusion conditions. Another part of uranium mineralization in permeable rocks is very difficult to leach. The reaction takes place very slowly here and it can be accelerated practically only by higher concentrations of the leaching reagent.

Under these conditions only acid leaching can be applied with any success. The main component of the leaching solution is sulphuric acid in an average concentration of about 5 %. The leaching solution is also enriched by an oxidant. The oxidant consists of nitric acid ($HNO_3$) and $NO_3^-$ ions. The filtration transport of the dissolved components is relatively fast, so the comparatively slow chemical reactions and the diffusion processes play the leading role in the whole process. For this reason the borehole network of new leaching fields was gradually adapted to the present optimal density. The total leaching period will take 15 - 25 years with gradually decreasing doses of chemicals and uranium concentrations in the leachant. The oldest leaching fields are still giving an economically workable solution at the present time.

The composition of the leaching agent and the batching of sulphuric acid were essentially modified during ISL technology application. The original technology didn’t differentiate the technological properties of ores in different parts of the deposit. The improvement of the technological evaluation methodology together with higher knowledge about the leaching process enabled the optimization of leaching technology by means of mathematical models. At the present
time different batches of sulphurid acid are applied to individual wellfields. The total batches are from 2,000 t to 14,000 t per 10,000 m$^2$. They are different also in the time evolution and the dependence on technological and economical properties of each deposit part.

During the adjustment of the well networks and the changes of the pumping technology a number of different networks have been tested, e.g. polygonal and linear ones. The first well fields were built with a square pattern and the distance between wells was 28 m. This well net was reduced to 20x20 m and even to 14x14 m on areas with the rich uranium mineralization. The injection and recovery wells alternated regularly. The newer well fields were built with a parallel pattern 20x50 m based on the higher quality of boring and casing and the higher knowledge of the leaching process. The last step in the development of well networks was connected with the application of submersible pumps. The present well networks are formed by lines of recovery wells with a diameter of 200 mm and by double lines of injection wells with a diameter of 90 mm. The distance between lines is about 100 m, the distance between recovery wells is from 60 m to 80 m and the distance between injection wells is about 12 m. The present network is also optimized with respect to the topographic inequalities in some parts of the deposit. Under favourable conditions it is possible to farm on the agricultural soil between individual well lines during the leaching process.

The long time activity of the individual wellfields also made it necessary to develop very safe well constructions. They must guarantee an almost perfect safeguard against the possible contamination of the upper Turonian aquifer by technological solutions. At the present time the injection wells are passed through the Turonian sequence with a double casing - on the outside by steel, from within by an acid-resistant casing pipe. Until 1984 pumping was performed exclusively by air-lifts and the construction of both the recovery and injection wells was similar. Since 1985 submersible pumps with a diameter of 4 - 6" have gradually been put into operation. These pumps require wells with a diameter of 200 mm cased by stainless steel. The well completion consists of underreaming, screening and gravel packing.

The difficult conditions made it necessary to work out an original methodology for the technological evaluation of the ores. It is based on special laboratory tests made on core samples from exploratory wells. They involve both short term and long term (up to 1000 days) leaching and filtration tests which are used to determine the kinetics of the chemical reactions. Similarly the rate of diffusion’s transport of the reagent and dissolved components is fixed. The filtration parameters of rocks are determined indirectly by means of regression analysis from the chemical composition. The resulting values serve as input data for the mathematical models.

The mathematical models are used for designing and optimization of the operation. They describe the geotechnological structure of the rocks, the underground flow of the solutions, the transport of components, the chemical reactions taking place underground and the economical relations also. The models also utilize the data collected in extensive databases referring to geological and technological features and the operation data of individual wells and leaching fields.

The pipelines on the surface are mostly made of polyethylene tubes. Stainless steel is used only in sections with higher pressure. The local pumping stations are built for solution transport from greater distances or at super elevations. The pipelines are equipped with a control system to signal any failures.

For the uranium separation from the solutions at the surface the classical ion exchange technology is used. Strongly basic anion exchange resins are used with respect to the composition
TABLE 1. THE IN SITU LEACHING FIELDS — DEPOSIT STRÁŽ

<table>
<thead>
<tr>
<th>Leaching field</th>
<th>Area [m²]</th>
<th>Thickness of U mineralization [m]</th>
<th>Average content of U [%]</th>
<th>NUMBER OF WELLS</th>
<th>TYPE OF THE WELL NETWORK</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Injection recovery recovery</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>[o 90 mm]</td>
<td>[o 90 mm]</td>
</tr>
<tr>
<td>VP-4</td>
<td>90120</td>
<td>7.55</td>
<td>0.073</td>
<td>182</td>
<td>44</td>
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<tr>
<td>VP-7A</td>
<td>288200</td>
<td>6.26</td>
<td>0.044</td>
<td>210</td>
<td>113</td>
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<tr>
<td>VP-7B</td>
<td>128600</td>
<td>7.93</td>
<td>0.064</td>
<td>93</td>
<td>63</td>
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<td>VP-8A</td>
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<td>0.041</td>
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<td>VP-8E</td>
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<td>2.77</td>
<td>0.067</td>
<td>177</td>
<td>18</td>
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</tbody>
</table>


of recovery solutions. The original technology with ion exchange in fixed bed columns was gradually replaced by the continuous countercurrent adsorption and elution technology. The proper development of the surface technology, of adsorption and elution apparatus, complementary technologies and the development of the most suitable eluant solution increased the efficiency of the surface processes and decreased the total consumption of chemicals. The resin loading strongly depends on the acidity of the recovery solution and in these conditions is low, amounting to approximately 10 g.dm\(^{-3}\)U. The solution based on HNO\(_3\) and NH\(_4\)NO\(_3\) is used for ionex regeneration. The nitrate ion which passes from the sorption partly to the leaching cycle is used in the leaching technology at the same time as oxidizing reagent. The uranium concentration in the eluate is about 5 g.dm\(^{-3}\). The eluate is neutralized by ammonium. The ammonium diuranate is the final product of the hydrochemical uranium extraction after the filtration and drying.

4. THE RESULTS OF ISL

About 30 wellfields are operated at the present time, the area each of them is from 100,000 m\(^2\) to 350,000 m\(^2\) (Table 1). The whole area of the wellfields is nearly 6 000,000 m\(^2\). The ISL facilities have a flow capacity of 90,000 m\(^3\) per day and a production capability of 800 t U per year.

<table>
<thead>
<tr>
<th>YEAR</th>
<th>Growth of the area [m(^2)]</th>
<th>Consumption of chemicals [t]</th>
<th>Production of U [t]</th>
</tr>
</thead>
<tbody>
<tr>
<td>1968 to 1970</td>
<td>90129</td>
<td>27011 2617 2186</td>
<td>56.8</td>
</tr>
<tr>
<td>1971</td>
<td>18861 2171 2855</td>
<td>149.8</td>
<td></td>
</tr>
<tr>
<td>1972</td>
<td>128599</td>
<td>41645 4103 1841</td>
<td>226.3</td>
</tr>
<tr>
<td>1973</td>
<td>630206</td>
<td>99394 11574 4158</td>
<td>477.4</td>
</tr>
<tr>
<td>1974</td>
<td>544530</td>
<td>184687 16062 6782</td>
<td>714.5</td>
</tr>
<tr>
<td>1975</td>
<td>167558</td>
<td>180657 17663 5544</td>
<td>775.2</td>
</tr>
<tr>
<td>1976</td>
<td>342637</td>
<td>186386 16182 6145</td>
<td>852.8</td>
</tr>
<tr>
<td>1977</td>
<td>229348</td>
<td>207555 16597 5905</td>
<td>859.5</td>
</tr>
<tr>
<td>1978</td>
<td>93825</td>
<td>188609 16765 6169</td>
<td>754.1</td>
</tr>
<tr>
<td>1979</td>
<td>290381</td>
<td>234938 15627 6155</td>
<td>785.8</td>
</tr>
<tr>
<td>1980</td>
<td>206201</td>
<td>231407 18915 7643</td>
<td>705.1</td>
</tr>
<tr>
<td>1981</td>
<td>279359</td>
<td>280030 17259 6998</td>
<td>682.8</td>
</tr>
<tr>
<td>1982</td>
<td>313015</td>
<td>209910 17159 6760</td>
<td>710.0</td>
</tr>
<tr>
<td>1983</td>
<td>222126</td>
<td>203640 16165 6153</td>
<td>702.5</td>
</tr>
<tr>
<td>1984</td>
<td>303619</td>
<td>198470 15764 5765</td>
<td>708.9</td>
</tr>
<tr>
<td>1985</td>
<td>231569</td>
<td>158460 14262 4647</td>
<td>712.6</td>
</tr>
<tr>
<td>1986</td>
<td>580275</td>
<td>217931 12807 3907</td>
<td>702.5</td>
</tr>
<tr>
<td>1987</td>
<td>90169</td>
<td>254066 11342 3677</td>
<td>705.8</td>
</tr>
<tr>
<td>1988</td>
<td>274611</td>
<td>275603 10050 3705</td>
<td>705.0</td>
</tr>
<tr>
<td>1989</td>
<td>246680</td>
<td>192612 9879 3548</td>
<td>700.2</td>
</tr>
<tr>
<td>1990</td>
<td>208220</td>
<td>149938 7871 2720</td>
<td>652.8</td>
</tr>
<tr>
<td>1991</td>
<td>201021</td>
<td>108300 8000 2600</td>
<td>627.7</td>
</tr>
</tbody>
</table>

TOTAL 5674078 3840000 270000 102600 13968.2
Only one wellfield was discarded and liquidated, the others are still in different phases of the leaching process. The first group, the oldest wellfields, is working without the addition of sulphuric acid, the lixiviant is only circulated after the separation of the uranium. The second group (from 5 to 10 years old) is operated with small batches of sulphuric acid, the acidity of the leaching solution is from 25 to 35 g.dm\(^{-3}\) and the circulated volume is low. The newest wellfields are operated with greater circulation and the concentration of sulphuric acid in injected solutions is from 40 to 100 g.dm\(^{-3}\).

The amount of chemicals injected into the wellfields during the period 1968-1991 are:

<table>
<thead>
<tr>
<th>Chemical</th>
<th>Amount (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>H(_2)SO(_4)</td>
<td>3,800,000</td>
</tr>
<tr>
<td>HNO(_3)</td>
<td>270,000</td>
</tr>
<tr>
<td>NH(_3)</td>
<td>100,000</td>
</tr>
<tr>
<td>HF</td>
<td>25,000</td>
</tr>
</tbody>
</table>

The growth of the area of ISL claims and consumption of chemicals in individual years of the operation is summarized in Table 2.

The sulphuric acid is consumed by the reaction with carbonate, aluminosilicate and iron minerals. The resultant reactions produce slightly soluble to soluble metallic sulfates which contribute to a buildup of SO\(_4^{2-}\) anions and polyvalent metal cations in the lixiviant. The nitric acid is consumed by oxidation reactions with uranium minerals, sulphide minerals and organic matter. The ultimate products of the oxidation reduction reactions are not quite well defined.

Nitrous and other intermediate nitrogen oxides products as well as nitrogen could be present in the solution. The ammonia is not consumed in the process. It is hydrolyzed and ionized to NH\(_4^+\) cations and is adsorbed on the clay minerals. The NH\(_4^+\) cations will desorb slowly during the restoration. The hydrofluoric acid is used for cleaning of injection wells. All of the HF has probably consumed by reaction with gangue minerals. The average chemical composition of the recovery stream resulting from the reaction of all the chemicals in the wellfields is:

<table>
<thead>
<tr>
<th>Major constituents</th>
<th>g.dm(^{-3})</th>
</tr>
</thead>
<tbody>
<tr>
<td>H(_2)SO(_4)</td>
<td>15 - 30</td>
</tr>
<tr>
<td>SO(_4^{2-})</td>
<td>40 - 65</td>
</tr>
<tr>
<td>NH(_4^+)</td>
<td>1 - 2</td>
</tr>
<tr>
<td>Al</td>
<td>4 - 6</td>
</tr>
<tr>
<td>Fe</td>
<td>0.5 - 1.5</td>
</tr>
<tr>
<td>NO(_3^-)</td>
<td>0.2 - 0.8</td>
</tr>
<tr>
<td>F(^-)</td>
<td>0.1 - 0.3</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Minor constituents</th>
<th>ppm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ca</td>
<td>200 - 300</td>
</tr>
<tr>
<td>Si</td>
<td>100 - 200</td>
</tr>
<tr>
<td>P</td>
<td>50 - 150</td>
</tr>
<tr>
<td>K</td>
<td>40 - 70</td>
</tr>
<tr>
<td>Zn</td>
<td>30 - 50</td>
</tr>
<tr>
<td>Mg</td>
<td>20 - 30</td>
</tr>
<tr>
<td>Ni</td>
<td>20 - 30</td>
</tr>
<tr>
<td>Na</td>
<td>10 - 15</td>
</tr>
<tr>
<td>V</td>
<td>10 - 15</td>
</tr>
<tr>
<td>Cr</td>
<td>5 - 15</td>
</tr>
</tbody>
</table>
TABLE 3. THE IN SITU LEACHING FIELDS — DEPOSIT STRĂZ

<table>
<thead>
<tr>
<th>Leaching field</th>
<th>Area [m²]</th>
<th>Time of operation [years]</th>
<th>The batch of sulphuric acid [kg/m²]</th>
<th>Recovery of U [%]</th>
</tr>
</thead>
<tbody>
<tr>
<td>VP-4</td>
<td>90120</td>
<td>24</td>
<td>1400</td>
<td>49.82</td>
</tr>
<tr>
<td>VP-7A</td>
<td>288200</td>
<td>19</td>
<td>834</td>
<td>45.93</td>
</tr>
<tr>
<td>VP-7B</td>
<td>128600</td>
<td>20</td>
<td>1685</td>
<td>47.59</td>
</tr>
<tr>
<td>VP-8A</td>
<td>235180</td>
<td>18</td>
<td>969</td>
<td>42.41</td>
</tr>
<tr>
<td>VP-8CD</td>
<td>342020</td>
<td>18</td>
<td>693</td>
<td>45.88</td>
</tr>
<tr>
<td>VP-8E</td>
<td>176630</td>
<td>17</td>
<td>1082</td>
<td>45.85</td>
</tr>
<tr>
<td>VP-8F</td>
<td>167560</td>
<td>17</td>
<td>1052</td>
<td>37.81</td>
</tr>
<tr>
<td>VP-9A</td>
<td>152450</td>
<td>15</td>
<td>1167</td>
<td>39.44</td>
</tr>
<tr>
<td>VP-9B</td>
<td>229350</td>
<td>15</td>
<td>772</td>
<td>59.14</td>
</tr>
<tr>
<td>VP-9C</td>
<td>76560</td>
<td>16</td>
<td>1045</td>
<td>57.25</td>
</tr>
<tr>
<td>VP-9D</td>
<td>113000</td>
<td>16</td>
<td>775</td>
<td>33.54</td>
</tr>
<tr>
<td>VP-10A</td>
<td>93820</td>
<td>14</td>
<td>879</td>
<td>46.23</td>
</tr>
<tr>
<td>VP-10B</td>
<td>90660</td>
<td>13</td>
<td>307</td>
<td>32.65</td>
</tr>
<tr>
<td>VP-10C</td>
<td>101040</td>
<td>13</td>
<td>1052</td>
<td>76.03</td>
</tr>
<tr>
<td>VP-11</td>
<td>68940</td>
<td>10</td>
<td>1403</td>
<td>46.73</td>
</tr>
<tr>
<td>VP-12A</td>
<td>96880</td>
<td>13</td>
<td>486</td>
<td>37.17</td>
</tr>
<tr>
<td>VP-12B</td>
<td>210420</td>
<td>11</td>
<td>429</td>
<td>44.75</td>
</tr>
<tr>
<td>VP-12C</td>
<td>50960</td>
<td>12</td>
<td>131</td>
<td>34.18</td>
</tr>
<tr>
<td>VP-12D</td>
<td>155250</td>
<td>12</td>
<td>412</td>
<td>34.64</td>
</tr>
<tr>
<td>VP-13A</td>
<td>102920</td>
<td>10</td>
<td>408</td>
<td>66.07</td>
</tr>
<tr>
<td>VP-13B</td>
<td>222130</td>
<td>8</td>
<td>421</td>
<td>18.23</td>
</tr>
<tr>
<td>VP-14</td>
<td>305090</td>
<td>9</td>
<td>780</td>
<td>49.23</td>
</tr>
<tr>
<td>VP-15</td>
<td>208620</td>
<td>8</td>
<td>737</td>
<td>74.07</td>
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<tr>
<td>VP-16</td>
<td>231570</td>
<td>7</td>
<td>475</td>
<td>43.28</td>
</tr>
<tr>
<td>VP-17</td>
<td>263230</td>
<td>5</td>
<td>318</td>
<td>9.46</td>
</tr>
<tr>
<td>VP-18</td>
<td>317040</td>
<td>6</td>
<td>421</td>
<td>29.48</td>
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<tr>
<td>VP-19</td>
<td>90170</td>
<td>5</td>
<td>480</td>
<td>16.15</td>
</tr>
<tr>
<td>VP-20</td>
<td>208220</td>
<td>2</td>
<td>362</td>
<td>17.95</td>
</tr>
<tr>
<td>VP-21</td>
<td>201020</td>
<td>1</td>
<td>39</td>
<td>6.48</td>
</tr>
<tr>
<td>VP-23</td>
<td>274610</td>
<td>4</td>
<td>313</td>
<td>41.16</td>
</tr>
<tr>
<td>VP-24</td>
<td>246680</td>
<td>3</td>
<td>359</td>
<td>57.53</td>
</tr>
</tbody>
</table>

The average uranium concentration in the recovery stream is about 20 ppm on the oldest fields and about 100 ppm on new fields. A high uranium concentration, about 500 ppm, was reached on some wellfields in the first year of production.

The uranium recovery in each wellfield is significantly different. It depends on the technological type of uranium mineralization and on the selected technological mode. The technological modes are designed on the basis of technological and economical modelling. On wellfields with poor and difficult leachable resources of uranium there are technological modes with small batches of sulphuric acid, a short production period and a low predicted recovery of uranium. The opposite modes are applied on wellfields with relatively rich and more easily leachable uranium resources. In this case economical conditions enable us to apply bigger batches of sulphuric acid and the recovery of uranium can reach up to 80 per cent of the geological resources (see Table 3).
5. SUMMARY

The plant for the chemical extraction of uranium in Stráž is the largest in Europe and is comparable to other similar plants in the world. During the last 25 years a great number of original technological and technical devices for the proper production and the later remediation of the affected area have been developed and we are continuing to develop them for the future. Vast experience was gained in the methods of control for underground hydraulics over large areas (designing and operation of hydraulic barriers, pumping centers, plants for the purification of contaminated water etc.)

The development of new wellfields has stopped in the present time. The main problems of the next utilization of ISL technology are to solve the underground restoration after finishing the extraction of uranium and to design environmentally safer technology. There are some other deposits in the north Bohemia, which are suitable for the application of ISL technology and some of them are suitable for alcalic leaching too.

This concise information cannot reflect the whole extent of the problems regarding underground leaching. If you would be interested in using our experience we will be happy to provide you with more detailed information.
IN-SITU LEACHING OF URANIUM IN BULGARIA: GEOLOGICAL, TECHNOLOGICAL AND ECOLOGICAL CONSIDERATIONS

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Sofia

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Abstract

In-situ leaching share reached 70% of the total uranium production in Bulgaria during 1990. The development well diameter is 150 mm, and that of the casing applied plastic pipes of 63 mm at pressure 10 kg/cm². The filters are of slit type with width 0.6–1.2 mm depending on rock grain size. Well logging gives data for location and thickness of ore bodies, filtration capability of the host rocks and other data of interest.

In-situ leaching is mainly done by sulfuric acid technology and only where the host rock carbonate content is high is applied base leaching solutions. The pregnant solutions are transported by airlift technic. The well distance (space) is 12.5–30 m and the form of the well sets depends on: ore, body shape, depth and filtration capability of host rocks. The well productivity is 30–150 m³ per day pregnant solutions at one leaching field. The resin applied is made in Hungary (Varion) or Russian made (AMP). The productivity of sorption column is 160 m³ per hour at resin volume 30 m³. Several improvements have been done to increase the leaching productivity.

There is very small influence of uranium in-situ leaching on soils in the solution mines and close to them. The uranium content in the mine soils is 6 to 8 ppm and that in the roots, stems, grains and fruits of the plants which grow on mine ground is 2–30 ppb.

The first uranium in-situ leaching started in 1967 at Orlov Dol deposit in South-East Bulgaria. The positive results achieved were followed by applying this technic in other sandstone type deposits which were not economic for underground mining. This was a technological revolution in uranium production in Bulgaria giving a chance to develop deposits which have been considered to be not economic. For a short period many deposits were discovered and developed and uranium in-situ leaching shares reached 70% of the total production in 1990.

1. GEOLOGICAL ENVIRONMENT

Sandstone type uranium deposits in Bulgaria were formed in small intra mountainous basins during Tertiary and Quaternery. The basins are surrounded by high mountainous blocks built up by granites and gneisses enriched in uranium from 6 to 10 - 30 ppm leachable uranium. The sediments are flat molasses strata containing carbonaceous material. The young tectonic movement from Tertiary until now helped development of favourable ore forming conditions and depositions of alteration of permeable with nonpermeable strata formed by material eroded from surrounded mountains.
Uranium in-situ leaching has been applied to sandstone deposits located in south Bulgaria. The most important are deposits distributed within Upper Thracian Depression where three ore bearing structures are known - Momino, Maritza, and the east periphery of Maritza basin - Sokol.

1.1 Momino structure occupies the northern part of Plovdiv graben and extends towards the east for 35 km with a width of 4 km. The syncline basement is built up by igneous and metamorphic rocks. The structure is constructed by marine and continental sediments with thickness of 500 m. The strata are dipping in east direction under 15 degrees. The upper strata are almost flat and host the river net in the region. The ore bearing strata are hosted by not sorted medium to fine grained sandstones with total thickness of 30 - 50 m. Ore host rocks are alternating sandstones and siltstones. The geochemical environment is characterised by the development of primary and secondary epigenetic oxidizing conditions, which pushed uranium leaching and its deposition under reduction conditions. The ore morphology is flat irregular roll-type bodies connected to oxidized and non oxidized border. The ore bodies are found at 100 to 260 m depth. Ningyoite is the main ore mineral associated with uranophan [1]. Momino and Zeretelevo deposits are located in this structure having reserves of more than 5000 tU and 1000 tU, respectively.

1.2 Maritza palaeochanel hosts nine deposits - Troian, Navasen, Trilistnik, Manole, Belosem, Pravoslaven, and Haskovo. The basement of the channel is built by Palaeozoic igneous and metamorphic rocks and is covered by marine and fluvial sediments. The latter hosts ore bodies which are distributed in all thicknesses of sandstones varying from 40 to 160 m. Uranium is concentrated in coffinite and phosphorus bearing coffinite. The ore body configuration is similar to lenses, and sickles with 2 to 8 m thickness. They are hosted by low permeable grey sediments containing organic material.

### CHEMICAL COMPOSITION OF ORE AND HOST ROCKS IN MARITZA DEPOSIT

<table>
<thead>
<tr>
<th></th>
<th>GREY HOST ROCKS</th>
<th>MEAN</th>
<th>GREEN HOST ROCKS</th>
<th>MEAN</th>
</tr>
</thead>
<tbody>
<tr>
<td>U</td>
<td>0.02 - 5.09%</td>
<td>0.068</td>
<td>0.02 - 0.068</td>
<td>0.032</td>
</tr>
<tr>
<td>Fe</td>
<td>1.02 - 8.40%</td>
<td>3.14</td>
<td>1.69 - 5.06</td>
<td>2.82</td>
</tr>
<tr>
<td>C org.</td>
<td>0.08 - 10.8%</td>
<td>1.03</td>
<td>0.03 - 0.32</td>
<td>0.15</td>
</tr>
<tr>
<td>S sulph</td>
<td>0.04 - 17.63%</td>
<td>1.20</td>
<td>0.02 - 0.25</td>
<td>0.14</td>
</tr>
<tr>
<td>Sm</td>
<td>10 - 83 ppm</td>
<td>53 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ce</td>
<td>104 - 592 ppm</td>
<td>272</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Yb</td>
<td>2.3 - 4 ppm</td>
<td>2.3 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ln</td>
<td>0 - 51 ppm</td>
<td>26 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ca</td>
<td>10 - 50 ppm</td>
<td>35 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ni</td>
<td>6 - 30 ppm</td>
<td>18 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>V</td>
<td>20 ppm</td>
<td>20 ppm</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ge</td>
<td>30 - 60 ppm</td>
<td>45 ppm</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

1.3 Sokol structure is developed in the foot hill of Sakar anticline, extended 20 km in east direction. Pre-Cambrian gneisses and Mesozoic sedimentary rocks are found in the basement. They are covered by ore bearing coarse grained sandstones and siltstones with thickness of 40 - 50 m.
The area of ore bodies varies from hundred to several thousand square metres. Uranium is distributed within the phosphorus bearing clay cement of the sandstones. It was also recognised in fluorine bearing apatite. Three small deposits are distributed in this structure: Orlov Dol where for the first time uranium in-situ leaching technic was applied, Madretz and Vladimirovo (Fig. 1).

2. HYDROGEOLOGICAL ENVIRONMENT

The productive ore bearing sequence is permeable and it is deposited above and covered by non permeable rocks. The permeability of the ore host rocks varies from 0.3 to 8 m per day. The thickness of non permeable seal is about 300 m at Momino deposit and no vertical movement of fossil water from productive sequence through seal clay strata has been recognized. Above the seal sequence are deposited Quaternary sands, saturated by fresh water which is used for drinking by the town of Plovdiv (Fig. 2).

The fossil water of productive sequence at Momino deposit contains sulphate up to 600 mg/l. This water is also radioactive and can not be used for drinking nor for industrial and irrigation purposes.

The Quaternary water reserves are 15 billion m$^3$ and they saturate all 80 m thickness of the horizon.
Sometime ore body permeability is less than that of host rocks. This is mainly due to the well cemented grains of the sandstones by clay material. These types of ore bodies are not developed.

The water of ore bearing sequence in some deposits is naturally pressured and it can reach 2 - 9 m above the surface. The natural speed of water movement is very low and it should not be considered. When the productivity of the strata water is very low (small debit) the lowering of its static level leads to difference of the levels and also to considerable outgo of air (energy).

3. URANIUM IN-SITU LEACHING TECHNIC

3.1 Ore quality

It is low grade ore which contains 0.02 - 0.07% and only in one deposit to 0.1% uranium. Ore minerals are presented by coffinite, phosphorus bearing coffinite, uranophane, and ingyio-water phosphate of uranium. They are associated by montmorillonite, barite, pyrite, marcasite, vanadium and molybdenum oxides, and Co-Ni sulphides. The colour of ore is grey or dark green if host rocks are tuffs and tuff containing sandstones. When the ore is completely oxidised its colour is yellow-braun.
Fig. 3  A. Horizontal projection  B. Cross-section of band-shaped ore body

1. Sandstone  2. Ore body  3.1 Productive wells  3.2 Controlling wells

Fig. 4  Horizontal projections of productive fields

A. Hexagonal scheme of well distribution in a production field
B. Orthogonal chess scheme of well distribution

1. Ore body  2. Upstream wells  3. Downstream wells
3.2 Well distribution

The production wells depending on ore body configuration, permeability of host rocks and ore depth are distributed by different ways as follows:

- combined scheme for irregular roll type ore body (Fig. 2);
- in one to three lines for band-shaped ore body (Fig. 3);
- hexagonal scheme in shallow ore body (Fig. 4 A);
- orthogonal chess scheme for deep located with normal geometric shape ore body (Fig. 4 B).

Well spacing depending on permeability of host rocks, and ore depth is 12 to 30 m.

3.3 Well construction

The well diameter is 150 mm and that of casing 63 mm. Plastic pipes are constructed from polyvinylchlorid which has a density of 1400 kg/m$^3$. The pipes are designed for 10 kg/cm$^2$ working pressure, the filter is of slit type with 0.6 - 1.2 mm width (Fig. 5). The latter depends on the grain size of the rocks and that of gravels applied to fill up the space between the filter and well walls. The casing pipes are assembled by using gum made in Germany (Tängit).
In-situ leaching is mainly done by sulfuric acid technology and only in few cases where the host rock carbonate content is high soda leaching is applied. The pregnant solutions are transported by air lift which is simple and easy to operate and maintain. The production of up-stream wells is 30 to 150 m³ per day. In order to balance the system the ratio down-stream : up-stream wells is 1:2 to 1:3 and even more.

Uranium concentration in pregnant solutions is 8 - 50 and only in few cases 100 mg/l at acid concentrations 1-3 g/l (about 1,4 of pH) and that of residual fluids 1 to 2 g/l.

Three periods of acid dissolving exist during development and production in a field, as follows:

- Starting, when acid concentration reaches 10 g/l. This concentration is kept for few months till ore bearing sequences are completely penetrated by the solutions.

- Base period during uranium leaching when the acid concentration is almost stable at about 4 to 6 g/l. It is the longest and its duration can reach 2 and more years.

- End period when the uranium reserves are almost depleted and acid concentration lowered to 0,5 - 1 g/l.

Duration of these three periods is 3 - 5 years when 60 to 80% of uranium reserves are leached. The chemical composition of pregnant solution is given in Table 1.

The solutions are transported by pipelines to the sorption plant. It consists of several sorption columns with cylinder shape made from chromnickel steel and filled up with anion resin of Varion AP type made in Hungary and AMP type made in Russia. Productivity of one sorption column is 160 m³/hour. The volume of the resin is about 25 m³. Uranium content in the solution after sorption is 1 to 2 mg/l. Uranium concentration in the resin is 15 - 40 kg/m³.

This uranium enriched resin is treated in the processing plant by convectional technic. Regenerated resin is transported back to the productive field.

Only three deposits are completely depleted by applying in-situ leaching technic (Orlov Dol, Madrets, and Vladimirovo). 82% of the uranium reserves are leached. The rest productive fields are still in production and uranium recovery is expected to reach 65 - 75%.

### Table 1. Chemical composition of pregnant solutions [2]

<table>
<thead>
<tr>
<th>DEPOSIT</th>
<th>pH</th>
<th>Al</th>
<th>Fe</th>
<th>V</th>
<th>Mn</th>
<th>Mg</th>
<th>Ca</th>
<th>Zn</th>
<th>K</th>
<th>Na</th>
<th>HSiO</th>
<th>Total mineralization</th>
</tr>
</thead>
<tbody>
<tr>
<td>Momino</td>
<td>1,4</td>
<td>520</td>
<td>1000</td>
<td>2,1</td>
<td>33</td>
<td>330</td>
<td>550</td>
<td>6,7</td>
<td>90</td>
<td>210</td>
<td>290</td>
<td>19,0</td>
</tr>
<tr>
<td>Rakovski</td>
<td>1,7</td>
<td>770</td>
<td>1200</td>
<td>4,1</td>
<td>37</td>
<td>300</td>
<td>440</td>
<td>7,3</td>
<td>96</td>
<td>130</td>
<td>300</td>
<td>16,3</td>
</tr>
<tr>
<td>Belozem</td>
<td>1,9</td>
<td>640</td>
<td>1200</td>
<td>3,5</td>
<td>61</td>
<td>320</td>
<td>420</td>
<td>6,3</td>
<td>100</td>
<td>90</td>
<td>320</td>
<td>16,2</td>
</tr>
<tr>
<td>Trilistnik</td>
<td>2,0</td>
<td>350</td>
<td>1000</td>
<td>3,0</td>
<td>31</td>
<td>310</td>
<td>460</td>
<td>5,1</td>
<td>150</td>
<td>240</td>
<td>300</td>
<td>12,1</td>
</tr>
<tr>
<td>Tsarimir</td>
<td>1,8</td>
<td>350</td>
<td>1200</td>
<td>5,8</td>
<td>12</td>
<td>290</td>
<td>320</td>
<td>5,1</td>
<td>50</td>
<td>150</td>
<td>250</td>
<td>15,8</td>
</tr>
<tr>
<td>Tzeretelovo</td>
<td>1,7</td>
<td>570</td>
<td>1300</td>
<td>18</td>
<td>17</td>
<td>270</td>
<td>250</td>
<td>6,8</td>
<td>150</td>
<td>110</td>
<td>350</td>
<td>14,8</td>
</tr>
<tr>
<td>Selishte</td>
<td>1,6</td>
<td>840</td>
<td>700</td>
<td>1,0</td>
<td>8</td>
<td>100</td>
<td>210</td>
<td>3,2</td>
<td>30</td>
<td>30</td>
<td>300</td>
<td>13,2</td>
</tr>
<tr>
<td>Cheshmata</td>
<td>1,5</td>
<td>450</td>
<td>1000</td>
<td>1,2</td>
<td>13</td>
<td>220</td>
<td>600</td>
<td>2,1</td>
<td>50</td>
<td>90</td>
<td>330</td>
<td>18,1</td>
</tr>
<tr>
<td>Navasen</td>
<td>1,8</td>
<td>320</td>
<td>900</td>
<td>1,1</td>
<td>11</td>
<td>140</td>
<td>400</td>
<td>3,1</td>
<td>200</td>
<td>900</td>
<td>330</td>
<td>15,4</td>
</tr>
<tr>
<td>Debar</td>
<td>1,9</td>
<td>310</td>
<td>1600</td>
<td>4,6</td>
<td>8</td>
<td>150</td>
<td>140</td>
<td>2,9</td>
<td>60</td>
<td>130</td>
<td>280</td>
<td>10,8</td>
</tr>
<tr>
<td>Okop</td>
<td>1,7</td>
<td>420</td>
<td>900</td>
<td>5,4</td>
<td>9</td>
<td>180</td>
<td>240</td>
<td>2,1</td>
<td>95</td>
<td>110</td>
<td>210</td>
<td>18,4</td>
</tr>
<tr>
<td>Chukarevo</td>
<td>1,7</td>
<td>490</td>
<td>2200</td>
<td>6,2</td>
<td>17</td>
<td>280</td>
<td>550</td>
<td>6,6</td>
<td>150</td>
<td>340</td>
<td>280</td>
<td>16,9</td>
</tr>
<tr>
<td>Tenevo</td>
<td>1,9</td>
<td>470</td>
<td>1100</td>
<td>5,2</td>
<td>6</td>
<td>270</td>
<td>410</td>
<td>4,0</td>
<td>160</td>
<td>240</td>
<td>270</td>
<td>14,4</td>
</tr>
</tbody>
</table>
4. URANIUM IN-SITU LEACHING IMPACTS ON THE ENVIRONMENT

A study was carried out during the last 24 years to find out the impact of uranium in-situ leaching on air, soils and waters in Bulgaria.

Radioactive air pollution is possible if there is radium in pregnant solution. In fact, only traces of it were recognized. It was measured 480, 550 MeV/l in the solution mine air compared to 4000 MeV/l permissible value. 1500 m far from the mine it is recognized 371 MeV/l compared to the accepted maximum standard of 1330 MeV/l.

Radium content in the pregnant solutions is very low and the total alpha activity of radon daughter products is also very low - 180 to 550 MeV/l. Gamma activity in the solution mine is 20 - 22 µR/h(cps) and outside it is 18 - 20 cps.

There is very small influence of uranium in-situ leaching on soils in the mines and close to them. The total alpha activity in the mine soils is 1850 - 7000 Bq/kg compared to 1850 - 2960 Bq/kg outside the mines. The uranium content in the mine soils is also very low - 6 to 8 ppm and that of radium 7,4 - 34 Bq/kg. Uranium concentration in the roots, stems, grains, and fruits of the plants which grow on solution mine soils is 2 - 30 ppb and 7,4 - 51,8 Bq/kg. We consider these concentrations as too low to have any impressive influence on the environment.

Uranium ore bodies mined by solution technology are hosted by sandstone layers which are isolated by clay sequences from the rest strata. Therefore, there is no flow of pregnant solution outside the mined productive strata.

The waters which penetrate uranium sandstone deposits have the total mineralisation of 0,5 - 2 g/l, pH 6,3 - 8,8, Fe 0,1 - 648 mg/l and SO₄ 23,5 - 758 mg/l. These concentrations are increased during mining and reached to: pH - 1,8, total mineralisation 12 - 20 g/l, SO₄ 10 - 18 g/l, Fe 0,5 - 1,5 g/l.

Two technics of mine water cleaning have been studied [3]. The first is based on chemical processes such as neutralization and the second uses natural processes which start up when the mining stops.

A study was carried out to detect the pollution at Orlov Dol deposit. Pollution of soils close to the wells at 1 m depth every 0,2 m and also outside the production field for comparison. The soils in the production field up to 0,4 m depth is acid with pH 5,5 to 4,0. Deeper than 0,4 m the sulphate content and that of Ca, K, Na and Mg ions are normal. To neutralise this acidity the soil is treated with lime. There is no radioactive pollution in the soil of that deposit. Only close to the sorption plant there is a slight increase of its radioactivity.

After treatment of the soil with lime the land is biological recultivated through growing wheat, pumpkin, watermelon and peanut. The analyses made did not prove higher radioactive metal concentration above the normal value in the roots, stems, grains and fruits. 840 water samples taken during 6 years from controlling wells in Orlov Dol depleted deposit showed that uranium concentration has been changed but there is a very sensitive decrease of sulphate acid concentration from 1,3 g/l to 0,01 g/l for 6 years.

The total mineralisation is decreased as follows: [3]

<table>
<thead>
<tr>
<th>No of wells</th>
<th>1982</th>
<th>1987</th>
</tr>
</thead>
<tbody>
<tr>
<td>932</td>
<td>3082 mg/l</td>
<td>334 mg/l</td>
</tr>
<tr>
<td>1550</td>
<td>12358 mg/l</td>
<td>1620 mg/l</td>
</tr>
<tr>
<td>2048</td>
<td>10760 mg/l</td>
<td>574 mg/l</td>
</tr>
<tr>
<td>2641</td>
<td>11670 mg/l</td>
<td>1460 mg/l</td>
</tr>
</tbody>
</table>
The water level is recovered as before uranium in-situ production. It could be concluded that 6 years are not enough for natural neutralization of underground water. The decrease of total mineralisation is mainly due to pH increase and deposition of hydroxides.

5. COMPARISON BETWEEN URANIUM IN-SITU LEACHING AND CONVENTIONAL UNDERGROUND MINING TECHNICS APPLIED IN BULGARIA

A period of 25 years of application of uranium in-situ leaching in Bulgaria is long enough to come to the conclusion that this technic is very effective to develop Bulgarian sandstone uranium deposits.

5.1 If we to compare this technic with underground mining we would find that deposits mined by in-situ leaching are twice lower grade than that mined by underground technic. But the cost of in-situ leached uranium is two times cheeper than that recovered by underground mining and related processing plant.

5.2 From an ecological point of view in-situ leaching is more harmless than underground mining. Solutions do not leach radium and it is left in the host rocks. Therefore, the potential hazards related to the operating uranium in-situ leaching mine is smaller than that of underground mine. The latter brings radium together with ore on the surface and after ore processing radium is accumulated together with tailings and it can pollute waters, soils, and air.

5.3 The cost of land re-cultivation of solution mine is smaller than that of underground mine and associated processing plant and tailing storage.

References

[1] SIMOV S.D., BOJKOV I.B.; "Case histories and new areas for uranium exploration in Bulgaria" New development in uranium exploration resources, production and demand Proceedings of technical Committee Meeting jointly organised by the IAEA and the NEA of the OECD, held in Vienna 26 - 29 August 1991.


UNDERGROUND LEACHING OF URANIUM IN
THE COMMONWEALTH OF INDEPENDENT STATES

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Moscow, Russian Federation

Abstract

Underground (in-situ) leaching of uranium has been used widely in the Commonwealth of Independent States since the early 1960's. Both acid and alkali are used as lixiviant. However, the majority of operations used acid solvent. Experience has shown the benefit of this method of uranium recovery from the point of view of economics and its impact to the environment. The paper discusses advantages and disadvantages of in-situ leaching of uranium production as compared to conventional mining operation. Brief descriptions of the various processes are also presented.

The underground (in-situ) leaching of uranium has been widely used in the Commonwealth since the early 1960's.

The in-situ leaching could be a success under favourable geological-hydrogeological and mineral-geochemical conditions only. The most important of them are the flooding of the deposit, permeability of the ore body and ability of uranium to transit from the solid phase into solution, when standard solvents and oxidants are used.

The deposit is to be worked by a linear system of injection and production boreholes in general; only in single cases a cell network can be used. The distance between the boreholes to be drilled depends on a multitude of natural and technological conditions, the potential design and operation of the boreholes, the process economics, and so on. Unlike the well fields design in the USA presenting networks of 15 by 15 meters or 30 by 30 meters, the Commonwealth uses wellfields with networks of 10 by 10, 10 by 20 and up to 25 by 50 meters as well as 10 by 100 meters. The injection boreholes array should be more tight for the deposits with smaller capacities.

The injection boreholes are usually drilled 90 to 161 millimeters in diameter, the production boreholes diameter often equals that of the injection wells.

The in-situ leaching practice in the Commonwealth requires both acid and carbonate-bicarbonate solvents. Still, the major operation method implements the sulfuric acid leaching circuit. This technology has been now well developed and tested. Unlike the alkaline leach scheme widely used in the USA, application of the sulfuric acid circuit is determined by a more favourable material composition of the ores to be leached, which is mainly the low carbonate content at the majority of the deposits worked on. Any acid concentration raise in the solutions directed to leaching will increase the mean content of uranium in the production solutions and decrease the process duration. Still, the acid consumption per unit of ore mass will increase. On the other hand, the leaching solution must have such concentration of sulfuric acid in the feed, so that the commercial solution from the production wells has a surplus acidity of 2 to 5 gram per liter.

The aggressive character of the acid determines its raised consumption per one ton of the ore mass, its value comprising 5 to 6 and up to 10 to 15 kilograms. The average acid consumption per 1 kilogram of uranium recovered into the end product varies from 18 to 100 and up to from 120 to 150 kilograms.
The carbonate-bicarbonate leaching circuits are applied for working of the ores containing acid-absorbing components, for example, more than 1.5 to 2.5% of carbonates. In practice one usually takes a solution of ammonium bicarbonate with the concentration from 0.5 up to 3 to 5 gram per liter as solvent. In some cases it is possible to use sodium bicarbonate solutions.

The carbonate in-situ leaching scheme uses atmospheric oxygen or commercial oxygen as oxidant.

An important feature in the in-situ uranium recovery is the metal recovery degree. It is evaluated by calculating the difference between the deposit reserves and uranium recovered. The reasons for the uranium losses are given in the written report.

Depending on the single constituents change, the uranium losses can vary within a wide range. Thus, if the reserves shared in impermeable ores would be some 5 to 10%, the sulfuric acid leach recovery will comprise 65 to 80% and with the carbonate scheme 55 to 75%. The uranium recovery degree at the in-situ sites is lower than at a hydrometallurgical plant, by 15 to 20%. But according to uranium losses in mining and obtaining some additional quantities from the off-grade (0.01 to 0.03%) and low-grade (0.01%) the total recovery approximates and sometimes exceeds its level of the traditional mining and following hydro-metallurgical treatment. Stripping of uranium from the in-situ leach production solutions at the surface plants is the same procedure in the Commonwealth as in other countries, which is the adsorption on ion-exchange resins.

Development of deposits according to the in-situ leach scheme, as compared to the conventional mining methods can considerably reduce the environmental impact. Nevertheless, when planning an in-situ leach project one has to consider some contamination of the soils and water, especially with an acid process.

In conclusion I would like to illustrate the report with six tables.

Table 1: Main Advantages of the In-situ Leach Method - All seven points are quite important.

Table 2: Comparison of Results Obtained in Working Uranium Ores with Hydrometallurgical and In-situ Leach Methods - Out of the 8 indications I would like you to pay attention to the points 3., 5., 6., and 8.

Table 3: Evaluation of Sulfuric Acid In-situ Leach Method - The main advantages are shown in points 1, 3, 4, and 5. The main drawback is the total mineralisation raise up to fifteen to twenty-five grams per litre, point 3.

Table 4: Evaluation of Carbonate In-Situ Leach Method - The main advantages are shown in points 1., 3., 4. The drawbacks are especially related to points 1., 3., and 6.

Table 5: Recovery of Metals as By-products by Sulfuric Acid In-Situ Leach Method - Practice and calculations verify the in-situ recovery feasibility for valuable metals with their relatively low contents in solutions. The main problem is the choice of a selective adsorbent: anionite, ampholyte or cationite.

Table 6: Main Methods for Restoration of the Underground Waters Quality in Sulfuric Acid and Carbonate In-Situ Leach Process - There have been at least six methods developed and tested. Their application depends on factual conditions and technical feasibilities.
Table 1

**MAIN ADVANTAGES OF THE IN-SITU LEACH METHOD**

1. Potential to work deposits confined to heavily flooded ores and rocks.
2. Feasibility to involve into mining poor and off-grade ores.
3. Shorter terms of putting deposits into operation.
5. Mining personnel working at the surface.
6. Elimination of dumps and tailings stores polluting the environment.

Table 2

**COMPARISON OF RESULTS OBTAINED IN WORKING URANIUM ORES WITH HYDROMETALLURGICAL (HM) AND IN-SITU LEACH (ISL) METHOD**

<table>
<thead>
<tr>
<th>Indices</th>
<th>HM</th>
<th>ICL</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Board content</td>
<td>0.03%</td>
<td>0.01%</td>
</tr>
<tr>
<td>2. Recovery from the ore:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>a) with the sulphuric acid method</td>
<td>63—66%</td>
<td>70—90%</td>
</tr>
<tr>
<td>b) with the biocarbonate method</td>
<td>—</td>
<td>50—60%</td>
</tr>
<tr>
<td>3. Block treatment period</td>
<td>10—15 years</td>
<td>3—5 years</td>
</tr>
<tr>
<td>4. Reagent consumption (relative)</td>
<td>100%</td>
<td>30—35%</td>
</tr>
<tr>
<td>5. Production cost of the product</td>
<td>100%</td>
<td>40—45%</td>
</tr>
<tr>
<td>6. Capital costs</td>
<td>100%</td>
<td>25—50%</td>
</tr>
<tr>
<td>7. Energy consumption</td>
<td>100%</td>
<td>30—40%</td>
</tr>
<tr>
<td>8. Labour productivity</td>
<td>100%</td>
<td>250—300%</td>
</tr>
</tbody>
</table>
### Table 3

**EVALUATION OF SULPHURIC ACID IN-SITU METHOD**

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Drawbacks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. High recoverability of uranium from ores (E=70—90%)</td>
<td>1. Low profits when working ores with high carbonates content (CO₂ — 2%)</td>
</tr>
<tr>
<td>2. Better kinetics of solutions interreacting with the ore (at E=80 L:S=1—3)</td>
<td>2. Deteriorating of the bed permeability due to chemical and gaseous plugging</td>
</tr>
<tr>
<td>3. Relatively shorter blocks working time (3—5 years)</td>
<td>3. Raised total mineralization of the underground waters up to 15—25 g/l</td>
</tr>
<tr>
<td>4. High uranium concentration in solutions (80—100 mg/l)</td>
<td>4. Necessity to use non-corrosive materials and equipment</td>
</tr>
<tr>
<td>5. Potential for complex recovery of some associates</td>
<td></td>
</tr>
<tr>
<td>6. Minimal spreadings of the solutions beyond the working contour due to the colmatage effect</td>
<td></td>
</tr>
<tr>
<td>7. Successful working of the ores without using any oxidant</td>
<td></td>
</tr>
<tr>
<td>8. Wide range of the indices values pertaining to the technological process</td>
<td></td>
</tr>
</tbody>
</table>

### Table 4

**EVALUATION OF CARBONATE IN-SITU LEACH METHOD**

<table>
<thead>
<tr>
<th>Advantages</th>
<th>Drawbacks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Potential to work uranium ores with raised carbonate contents (CO₂—2%)</td>
<td>1. Insignificant recovery degree of uranium mined (E = 50—60%)</td>
</tr>
<tr>
<td>2. Selective recovery of certain number of elements from ores, facilitating successful treatment of solutions in adsorption units</td>
<td>2. Lower kinetics of the uranium minerals dissolution (at E = 60% L:S = 4—6)</td>
</tr>
<tr>
<td>3. Using of common materials and equipment</td>
<td>3. Compulsory use of an oxidant</td>
</tr>
<tr>
<td>4. Insignificant raise of the underground water total mineralization</td>
<td>4. Lower concentration of uranium in the production solutions</td>
</tr>
<tr>
<td></td>
<td>5. High degree of solutions spreading beyond contours of the blocks under treatment</td>
</tr>
<tr>
<td></td>
<td>6. Compulsory preliminary softening of underground waters</td>
</tr>
</tbody>
</table>
Table 5

RECOVERY OF METALS AS BY-PRODUCTS BY SULPHURIC ACID IN-SITU LEACH METHOD

<table>
<thead>
<tr>
<th>Elements recovered</th>
<th>Mean contents in the production solutions, mg/l</th>
<th>Contents necessary for profitable recovery, mg/l*</th>
<th>Adsorbent used</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rhenium</td>
<td>0.2 — 0.5</td>
<td>0.2</td>
<td>AMP (AM; AMp) co-adsorption</td>
</tr>
<tr>
<td>Selenium</td>
<td>50 — 60</td>
<td>10</td>
<td>selective adsorption on thiovirol</td>
</tr>
<tr>
<td>Molybdenum</td>
<td>10 — 40</td>
<td>10</td>
<td>AMP (AM; AMp) co-adsorption</td>
</tr>
<tr>
<td>Vanadium</td>
<td>10 — 40</td>
<td>40 — 50</td>
<td>selective on VPS</td>
</tr>
<tr>
<td>Scandium</td>
<td>0.15 — 0.6</td>
<td>0.3</td>
<td>Ampholyte</td>
</tr>
<tr>
<td>Yttrium</td>
<td>up to 15</td>
<td>10</td>
<td>AFI-21, AFI-22</td>
</tr>
<tr>
<td>Rare earths</td>
<td>10 — 40</td>
<td>10</td>
<td>— “ —</td>
</tr>
</tbody>
</table>

* Arbitrary solution flow rate 1000 m³/hr. Product prices of 1990.

Table 6

MAIN METHODS FOR RESTORATION OF THE UNDERGROUND WATERS QUALITY IN SULPHURIC ACID AND CARBONATE IN-SITU LEACH PROCESS

<table>
<thead>
<tr>
<th>Recultivation principles</th>
<th>Recultivation site</th>
<th>Methods</th>
<th>Development stage</th>
</tr>
</thead>
<tbody>
<tr>
<td>Physical-chemical cleaning of the residual solutions</td>
<td>At the surface</td>
<td>Precipitation using lime pulp</td>
<td>Contractor design</td>
</tr>
<tr>
<td></td>
<td>Within the bed</td>
<td>Electrodialysis</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Hyperfiltration</td>
<td></td>
</tr>
<tr>
<td>Neutralization of the residual solutions</td>
<td>Within the bed</td>
<td>Electrosorption technology</td>
<td>Pilot plant</td>
</tr>
<tr>
<td>Washing with stratal waters</td>
<td></td>
<td>Natural demineralization</td>
<td>Experiments</td>
</tr>
<tr>
<td>Demineralization intensified by accelerated underground currents rate («pulling» method)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
The development and implementation of the in-situ leach process in mining or uranium is a considerable achievement in science and engineering of mining industry. Experience accumulated in working of the deposits testifies a promising future for the methods. The method as irrefutable potential for future development, since the results achieved should be considered as the first step only.
THE CROW BUTTE ISL PROJECT — A CASE HISTORY

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Abstract

The Crow Butte project, located in the northwest corner of Nebraska, is one of less than half a dozen commercial in-situ uranium leach mines currently operating in the United States. The facility began commercial production in April 1991 some 12 years after the discovery of the sandstone, roll front uranium deposit. The ore body lies about 600 feet (183 meters) below the surface in a highly permeable aquifer with excellent shale/clay confinement above and below. The exploration and development period included ore body delineation, pilot testing with successful demonstration of aquifer restoration, feasibility study, environmental licensing, engineering design, and facilities construction. The commercial start-up is taking a phased approach with current annual production of some one-half million pounds U₃O₈ increasing to a planned one million pounds by 1995. Operations during the first year and a half of commercial production, using alkaline leach chemistry, have gone smoothly with no major technical or regulatory problems. Capital costs for construction of the 3,500 gallon per minute (221 liters per second) capacity plant were minimized by acquiring used processing equipment from closed in-situ leach uranium mines located in Texas and Wyoming. The total manpower requirement of only 25 on-site employees contributes to the low production costs.

1. INTRODUCTION

The Crow Butte in-situ leach (ISL) uranium mine, located in northwest Nebraska, commenced commercial production of yellowcake in April 1991. The well field is presently operating at 2,400 gallons per minute (151 liters per second) with this flow to be increased to the 3,500 gallons per minute (221 liters per second) capacity of the processing plant. The mine uses an alkaline leach chemistry (sodium bicarbonate) with oxygen as the oxidant. The ore body, discovered in 1980, is some 600 feet (183 meters) below the land surface in the present mining unit. The mineralization occurs in a typical sandstone, roll front deposit. The deposit has an overall ore grade of about 0.25% U₃O₈. Pilot scale mining that took place in the 1986-87 time period demonstrated the technical and economic feasibility of both the mining and aquifer restoration phases of the project.

The environmental licensing of the project with the state and federal regulatory authorities has been somewhat slower and more difficult in comparison with other ISL mines in the United States primarily because the Crow Butte project is the very first uranium mine in the state of Nebraska. Nebraska did not have any regulations covering uranium mining prior to the discovery of the Crow Butte deposit. It was therefore necessary for the state to research, prepare, and approve a set of ISL mining regulations. Now that the state officials have regulations and have become familiar with ISL mining, the regulatory process is much smoother and efficient, and license modifications are usually obtained in a reasonable length of time.

The Crow Butte project is located about 4 miles (6.5 kilometers) southeast of the small town of Crawford, Nebraska (see Figure 1) at an elevation of 3,900 feet (1,190 meters). The rolling plains topography, with local relief of less than 100 feet (30 meters), is typical of western
FIG. 1. Crow Butte project index map.
Nebraska and the land surface is used primarily for livestock grazing, and the growing of hay and wheat. The annual average precipitation is 15.5 inches (39.5 centimeters), and the temperature extremes are +110 degrees Fahrenheit (43 degrees Centigrade) summer and -31 degrees Fahrenheit (-35 degrees Centigrade) winter.

The Crow Butte project is held and operated by Ferret Exploration Company of Nebraska, Inc. (FEN) under a Joint Venture and Operating Agreement to which FEN is a party. The owner of the underlying mineral rights is the Crow Butte Land Company, a subsidiary of FEN. The beneficial owners of the project, and the owners of all production, are Uranerz U.S.A., Inc., Kepco Resources of America, Inc., Ferret Exploration Company, Inc., First Holding Company and Geomex Minerals, Inc.

2. GEOLOGY

The ore body is located in the basal sandstone of the Chadron Member of the White River (Oligocene) Formation (see stratigraphic column, Figure 2). Below the basal Chadron sandstone is
the Pierre Shale, a dark gray to black marine shale some 1,500 feet (457 meters) thick locally, which very adequately serves as the lower confining unit for ISL operations [1]. The unconsolidated basal Chadron sandstone containing the uranium ore is about 40 feet (12 meters) thick, being the depositional product of a large, braided stream system which formed during the early Oligocene (36 to 40 million years before present). The middle Chadron unit, immediately above the basal Chadron sandstone, consists of clays and siltstone, and serves as an excellent upper confining layer for ISL mining operations. This upper confining layer is 200 to 300 feet (61 to 91 meters) thick over the project area.

The origin of the mineralization is considered to be secondary deposition (precipitation) of uranium leached from volcanic ash. The ore body is a typical roll front deposit with the oxidation/reduction front about 6 miles (10 kilometers) long and up to 3,000 feet (914 meters) wide. The ore, at any location, is located in from one to three horizons. The ore thickness ranges from 5 to 15 feet (1.5 to 4.5 meters) and the depth to the ore body from the surface ranges from 275 to 820 feet (84 to 250 meters). Mineralization occurs within the sandstone unit as thin coating on the sand grains. The average grade of the ore in the Crow Butte deposit is 0.25% $U_3O_8$. The in-place uranium reserves for the project are estimated at over 30 million pounds $U_3O_8$.

A study of the regional geology, including the interpretation of oil and gas logs, defined an extensive fluvial sandstone system at the base of the Tertiary just above the Pierre Shale. Gamma kicks in some of the oil and gas wells, coupled with the presence of methane gas shows, provided the basis for an exploration drilling program. In 1979 a total of 95 exploration holes were drilled in the Crawford area with some weak uranium mineralization encountered. The 100th exploration hole drilled in 1980 intersected ore grade uranium mineralization establishing the discovery of the Crow Butte ore body. Additional drilling in 1980 defined the mineralized trend [2].

3. HYDROGEOLOGY

The basal Chadron sandstone, the uranium host sand, is water saturated and forms a regionally extensive confined aquifer that produces artesian (flowing) conditions where the local topography is depressed. The basal Chadron is a coarse-grained arkosic sandstone with an average thickness in the project area of 40 feet (12 meters). The ore body aquifer, referred to locally as the Chadron aquifer, has a piezometric surface some 495 feet (151 meters) above the top of the sand dipping gently to the north-northwest. The overlying aquifer nearest the Chadron aquifer is the Brule sand (aquifer) which is separated from the Chadron aquifer by some 200 to 300 feet (61 to 91 meters) of clay, claystone and siltstone. The difference in elevation of the piezometric surface of the Chadron and Brule aquifers in the project area is about 59 feet (18 meters).

A total of two major aquifer pump tests were performed in the project area to establish the hydrogeologic parameters for mining and environmental purposes. The aquifer pump tests were conducted for 51 and 72 hours, and produced quality data that were used to determine the hydrologic properties of the Chadron aquifer, and the upper and lower confining layers. The results of the aquifer pump tests show that the Chadron (ore body) sandstone is a non-leaky, confined, slightly anisotropic aquifer with an average hydraulic conductivity (permeability) of about 9.10 feet per day (2.8 meters/day). The average storativity was calculated to be $1.04 \times 10^{-4}$. The pump tests demonstrated, to the satisfaction of the regulatory agencies, that the upper and lower confining units sufficiently restrict the vertical movement of water out of the Chadron aquifer [3][4].
4. PILOT TESTING

After extensive exploration drilling and evaluation in 1981 and 1982, the decision was made to license and operate a pilot test mine to establish both the amenability of the ore deposit to ISL mining, and the technical/economic feasibility of successfully restoring the aquifer contaminated by mining. This latter aspect was a requirement of the regulatory agencies and a prerequisite for issuance of a license for commercial mining operations. In 1982 the location for the pilot test well field was selected, and environmental baseline data were collected for preparation of the pilot test license application. The license application was submitted to the regulatory agencies in early 1983 and all necessary approvals were obtained in 1985. Construction of the 100 gallon per minute (6.3 liters per second) pilot processing plant began in mid 1985 and was completed in early 1986.

Pilot test mining started in mid 1986 and lasted until February 1987 using an alkaline based (sodium bicarbonate) lixiviant operating at a pH in the 7.1 to 7.4 range. Oxygen was used as the oxidant and ion exchange columns using anion resin were used to recover the uranium. The restoration (recultivation) of the affected aquifer started shortly after the completion of mining and lasted until August 1987 (about a six month period). Aquifer restoration methodology included groundwater sweep, water treatment and reductant addition, and finally aquifer recirculation.

The pilot test results successfully demonstrated the technical and economic viability of the project in terms of both uranium extraction and aquifer restoration. The test also demonstrated, to the satisfaction of the regulatory agencies, that the mining solutions can be contained in both the horizontal and vertical dimensions. There were no excursions of mining solution during the pilot test. The project did not experience any significant regulatory or environmental problems during the pilot test, and there were no problems with regard to radiation exposure to workers.

5. COMMERCIAL OPERATIONS

5.1 Environmental Licensing

In order to construct and operate an ISL uranium mine in Nebraska it was necessary for the operator, Ferret Exploration of Nebraska, to obtain project approvals from both the state and federal governments. The principle state regulatory agencies responsible for ISL mining are the Nebraska Department of Environmental Quality and the Nebraska Department of Water Resources. At the federal level, the main agency responsible for uranium mining is the U.S. Nuclear Regulatory Commission (NRC). License application documents had to be submitted separately to each of the three agencies mentioned above. Preparation of the commercial licensing applications, containing extensive information on hydrology, vegetation, soils, wildlife and radiological conditions, started in 1986 and submittal of the voluminous application documents took place in October 1987.

As mentioned in the introduction section of this paper, environmental licensing was complicated and delayed by the fact that the Crow Butte project was the first uranium mine ever licensed in the state of Nebraska. The appropriate state agencies had to first write, and then get approved, regulations to cover ISL uranium mining. After much effort and considerable negotiations, the necessary environmental license and approvals were issued in April 1990.
Opposition from a small but vocal environmental group lengthened the licensing by increasing the number of public meetings and hearings. The local residents are overwhelmingly in favor of the project, and the environmental group was not successful in any of its many appeals and protests.

5.2 Commercial Construction

The construction of the commercial ISL mining facilities started in May 1990 and was completed in March 1991. The facilities can be divided into three major components. The first component is the well field which consists of the injection and recovery (production) wells, monitor wells, downhole equipment and surface equipment. The second component is the processing plant which includes the building and the process equipment. The third component is the waste water disposal system which, at Crow Butte, consists of plastic lined (double liner) evaporation ponds. Each of these components will be discussed separately.

5.2.1 Well Field Design and Construction

The wells needed to produce uranium at the Crow Butte ISL mine are installed by contract drillers using rotary drilling rigs typically found in the water well drilling business. At Crow Butte three contract drilling rigs are employed by the operator to install injection, recovery and monitor wells. The project uses PVC Yelomine casing with 4.5 inch (11.4 centimeters) internal diameter. The well annulus is completely cemented from bottom to top, and the screen is stainless steel with a slot size of .020 inch (.051 centimeter). Well completion is accomplished using the underreaming method where only the mineralized interval is exposed by cutting away the casing and cement. Downhole pumps sized at either 5 or 7-1/2 horsepower (3.7 to 5.6 kilowatts) are hung on 2 inch (5 centimeter) PVC pipe in each recovery well. Recovery wells and injection wells are drilled and completed in an identical fashion using the same diameter casing. Following the completion of each injection, recovery and monitor well, an integrity test must be performed to ensure that there are no leaks in the casing. The results of the integrity test must be documented in writing and submitted to the regulatory agencies for review.

From 15 to 20 recovery wells, with associated injection wells, are piped to a well field manifold house using buried plastic pipe. The well field manifold house includes a flow meter, flow control valve and sample port for each well piped to the house. The electrical power on/off control for each recovery well is also located in the well field manifold house. From the well field manifold house the solutions are routed to and from the main processing plant in buried 10 inch (25.4 centimeter) PVC Yelomine pipe. The manifold houses are skid mounted so that they can be taken to a new well field when they are no longer needed in an old well field. The project typically uses 5-spot and 7-spot patterns with a nominal distance between recovery and injection wells of about 70 feet (21 meters).

The leaching solution (or lixiviant), which causes the dissolution of the uranium, consists of Chadron Formation groundwater fortified with soda ash (Na$_2$CO$_3$) and carbon dioxide (CO$_2$) to form a sodium bicarbonate alkaline leach chemistry. The pH of the lixiviant is kept in the 7.2 to 7.6 range. Oxygen is added to the injection solution at each well field manifold house. Oxygen is delivered to the mine and stored on site as liquid oxygen. A chemical description of the well field and process chemistry is presented in Figure 3 [5]. The carbonate strength of the lixiviant is kept at about 1,500 mg/l CO$_3$ and the target oxygen concentration is 200 to 400 parts per million.
Fortification of Lixiviant

Barren Lixiviant + HCO_3^- + CO_2 + O_2

Injection Wells

Ore Body

Uranium: \( \text{UO}_2 + \frac{1}{2} \text{O}_2 \rightarrow \text{UO}_3 \)

\( \text{UO}_3 + 2\text{HCO}_3^- \rightarrow \text{UO}_2(\text{CO}_3)_2^- + \text{H}_2\text{O} \)

\( \text{UO}_3 + \text{CO}_3^- + 2\text{HCO}_3^- \rightarrow \text{UO}_2(\text{CO}_3)_3^- + \text{H}_2\text{O} \)

Vanadium: \( \text{V}_2\text{O}_3 + \text{O}_2 + 4\text{HCO}_3^- \rightarrow 2\text{VO}(\text{CO}_3)_2^- + 2\text{H}_2\text{O} \)

Pyrites: \( 2\text{FeS}_2 + 70_2 + 8\text{Na}_2\text{CO}_3 + 6\text{H}_2\text{O} \rightarrow 2\text{Fe(OH)}_2 + 4\text{Na}_2\text{SO}_4 + 8\text{NaHCO}_3 \)

Calcite: \( \text{CaCO}_3 + \text{CO}_2 + \text{H}_2\text{O} \rightarrow \text{Ca}^{+2} + 2\text{HCO}_3^- \)

Clay: \( \text{CaClay} + 2\text{Na}^{+1} \rightarrow \text{Na}_2\text{Clay} + \text{Ca}^{+2} \)

Production Wells

Processing Pregnant Lixiviant

Strong Base Anion IX Load: \( 2\text{R}\text{HCO}_3 + \text{UO}_2(\text{CO}_3)_2^- \rightarrow \text{R}_2\text{UO}_2(\text{CO}_3)_3^- + 2\text{HCO}_3^- \)

Note: Chloride or Sulfate substitute for carbonate

Elution: \( \text{R}_2\text{UO}_2(\text{CO}_3)_3^- + 2\text{Cl}^- + \text{CO}_3^- \rightarrow 2\text{RCI} + \text{UO}_2(\text{CO}_3)_3^- \)

Precipitation: \( \text{UO}_2(\text{CO}_3)_3^- + 6\text{H}^{+1} \rightarrow \text{UO}_2^{+2} + 3\text{CO}_2(\text{g}) + 3\text{H}_2\text{O} \)

\( \text{UO}_2^{+2} + \text{H}_2\text{O}_2 + 2\text{H}_2\text{O} \rightarrow \text{UO}_4(\text{H}_2\text{O})_2 + 2\text{H}^{+1} \)

\( \text{NaOH} + \text{H}^{+1} \rightarrow \text{Na}^{+1} + \text{H}_2\text{O} \)

RESTORATION CHEMISTRY - Lower Eh

\( \text{Na}_2\text{S} + \text{H}_2\text{O} \rightarrow 2\text{Na}^{+1} + \text{HS}^{-1} + \text{OH}^{-1} \)

\( 4\text{UO}_2(\text{CO}_3)_3^- + \text{HS}^{-1} + 15\text{H}^{+1} \rightarrow 4\text{UO}_2^+ + \text{SO}_4^{2^-} + 12\text{CO}_2 + 8\text{H}_2\text{O} \)

Prepared by: Charles E. Miller [5]
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FIG. 3. Crow Butte mine production chemistry.
FIG. 4. Process flowsheet.
5.2.2 Processing Plant Design and Construction

The processing plant building, which also includes the laboratory, maintenance shop, warehouse and staff offices, is a standard metal building some 300 feet (91 meters) long, 115 feet (35 meters) wide and 52 feet (16 meters) tall at its highest point. The major process components such as ion exchange columns, elution and precipitation tanks, product belt filter, chemical make up tanks, pumps, piping and some electrical gear were purchased used from closed ISL uranium mines in Texas and Wyoming. The used equipment included the supporting structural steel. The purchase of used equipment resulted in substantial capital cost savings for the project. Construction of the building and process plant was completed on time and under budget, and no significant problems were encountered during the plant construction phase.

The plant uses 8 up flow ion exchange (IX) columns with a diameter of 14 feet (4.3 meters) and a height of 35 feet (10.7 meters). The strong based anion resin is eluted (stripped) in-place eliminating the need to transfer resin. The uranium loaded resin is stripped with a sodium chloride/sodium bicarbonate eluant. Downstream of the IX columns are three precipitation tanks with agitators. The carbon dioxide in the concentrated eluant is removed by the addition of acid and precipitation of the uranium is accomplished with hydrogen peroxide (H₂O₂) followed by pH adjustment with caustic (NaOH). The precipitated uranium is then sent to a thickener. From the thickener the product is sent to a belt filter to remove excess water from the yellowcake slurry. The final slurry product is trucked from the mine to the converter in a special stainless steel tanker trailer. A simplified process flow sheet is included as Figure 4 [3]. The only product recovered at the Crow Butte mine is uranium; there are no by-products. Monitoring and recording of plant processes and well field operating conditions are accomplished with a computer. The computer also controls some of the processing plant operations.

5.2.3 Waste Water Disposal

The disposal of waste water is accomplished by plastic lined evaporation ponds using a double liner system. The sources of the waste water are the well field over production, the plant bleed and the aquifer restoration liquid effluents (e.g. reverse osmosis brine). Waste water is routed from the plant to the evaporation ponds in buried PVC pipe. Two ponds were constructed prior to start-up of commercial mining and additional ponds will be added as needed. The ponds are constructed with earthen embankments and covered with a 20 mil thickness PVC bottom liner and a 60 mil thickness HDPE top liner separated by a geonet. A leak detection system located between the two liners reports any liquid coming through the top liner. In the future, a deep disposal well and a land application waste water disposal system may be added at the mine.

5.3 Commercial Mining Operations

The construction of the commercial processing plant and ancillary facilities, and the installation of the first mining unit were completed in March 1991. Mining operations commenced in April 1991 with the continuous circulation of groundwater, addition of mining chemicals (carbon dioxide and bicarbonate), and the addition of the oxidant (oxygen). The first commercial well field, designated Mine Unit 1, consists of some 38 recovery and 72 injection wells. The target flow rate of 1,200 gallons per minute (76 liters per second) was achieved as the recovery wells averaged 31 gallons per minute (2 liters per second) each. A strong positive response to oxidant addition in the form of increasing uranium head grade occurred in three to four days after the oxygen system was activated.
There were only two minor problems experienced during start-up that are worthy of note. One problem was the frequent plugging of the injection solution filter system (bag filters) caused by the initial surge of sediment from the 38 recovery wells turned on over a short period of time. This problem was temporary and essentially corrected itself as the production wells soon stopped producing significant amounts of sediment. The other problem was the presence of leaks in the plastic lined evaporation ponds. The ponds have double liners and the leak detection system is located between the two liners. The regulatory agencies were properly notified of the leaks and the small holes in the liner causing the leaks were quickly located and repaired. Because of the double liner system, none of the leaked waste water was released to the environment.

In March of 1992, start-up operations for Mine Unit 2 commenced using a procedure of turning on only a few patterns at a time. By the end of July 1992 all the production and injection wells in Mine Unit 2 were in operation. Most of Mine Unit 1 remained in production as Mine Unit 2 was phased into the operation. Some of the patterns in Mine Unit 1 with very low uranium head grade were turned-off. Mine Unit 2 has 52 recovery wells and 89 injection wells. At this time all the patterns in Mine Unit 2 and about 80% of the patterns in Mine Unit 1 are in production giving a total well field flow rate of about 2,400 gallons per minute (151 liters per second). The uranium head grade since start-up has been lower than forecasted averaging about 66 mg/l. The problem may be related to dilution of the recovery solution. To reduce the amount of dilution the operator has instituted a new procedure for installing wells. This new procedure, which utilizes underreaming, is currently being used in the construction of Mine Unit 3. It is felt that the underreaming well completion procedure will allow the project to achieve the planned average head grade of 80 mg/l.

The mine operates with a compliment of 24 employees including site management. An additional 9 to 12 people operate the drilling rigs installing new well fields. The drillers are contractors and not employees of the mine operator. The plant is operated by two people working 8-hour shifts. The on-site laboratory is operated by one person using state of the art automated analytical equipment.

6. ENVIRONMENTAL AND RADIATION SAFETY CONSIDERATIONS

The project operates with an approximate 1% over production or well field bleed. This bleed accomplishes two things. First, a hydrologic cone of depression is maintained in the well field to ensure that contaminated water does not migrate out of the mining area and cause a legally defined excursion. The second reason for the bleed is to prevent the unwanted build-up of certain chemical species in the lixiviant. The bleed stream is taken out of the circuit in the processing plant and is piped to the plastic lined evaporation ponds. As of this time there have been no well field excursions at the Crow Butte mine.

The main component of the environmental monitoring program during the mining phase of the Crow Butte operation consists of the sampling of horizontal and vertical excursion monitor wells. The horizontal monitor wells are located 300 feet (91 meters) from the edge of the well field mining units and they are completed in the same sandstone unit containing the ore body (in this case, the basal Chadron sandstone). The vertical excursion monitor wells are located within the mining unit in the first aquifer above the ore body aquifer at a density of one monitor well per 4 acres (1.6 hectares) of well field. The baseline water quality in the monitor wells is determined prior to mining and excursion limits are established in accordance with regulatory guidelines. The excursion monitor wells are sampled and analyzed once every two weeks during mining. If an
analysis of a monitor well indicates that the excursion limits have been exceeded the regulatory agencies must be immediately notified and corrective actions must be initiated. The leak detection system for the evaporation ponds is checked daily.

Each employee working at the mine must receive extensive training in radiation safety provided by the full time on-site radiation safety officer. The approved radiation safety program includes personal alpha monitoring each time an employee leaves the processing plant and bioassays for uranium are performed quarterly for site personnel. Additionally, each employee must wear a personal TLD (gamma) badge which is exchanged quarterly. The radiation safety officer conducts frequent radiation checks in the plant by performing radon, radon daughters, uranium airborne particulate, and alpha/gamma surveys at numerous locations. Environmental radiation monitoring for radon and gamma levels is conducted outside of the plant building. The Nuclear Regulatory Commission (NRC) enforces the radiation safety program and conducts unannounced inspections of the operation at least once a year.

7. AQUIFER RESTORATION AND DECOMMISSIONING

Once mining has completely stopped in a particular mining unit, the regulatory agencies require the operator to commence aquifer restoration (groundwater clean-up) operations. The goal of aquifer restoration is to return the water quality in the affected aquifer to baseline (pre mining) conditions on a parameter by parameter basis. It is recognized by the agencies that not all parameters can be returned all the way to baseline values and allowances are made for these parameters; however, the overall water quality of the aquifer after restoration is complete must be such that the water can be put to the same use as before mining. For example, if the aquifer was suitable for watering livestock before ISL mining, it must be suitable for livestock watering after restoration is complete. The operator must conduct a rigorous groundwater monitoring program to document the pre mining water quality of the ore body aquifer.

At Crow Butte the operator will use the same aquifer restoration technique successfully utilized during the pilot test. The first phase is called groundwater sweep which involves pumping wells from the mine unit under restoration (no injection) and transferring this water to the next well field to be mined. This action draws the halo (or plume) of contaminated water just outside the well field back into the mining unit. The second phase uses water treatment to remove the contaminates from the water. The clean water (permeate) from the water treatment unit is injected back into the well field being restored. The contaminated water (brine) from the water treatment unit is sent to the evaporation ponds. At the end of the water treatment step a reductant such as hydrogen sulphide is added to the permeate being injected to stop the chemical reactions taking place in the aquifer undergoing restoration. The third phase is simply the circulation of the water through the ore body aquifer to make the quality of the groundwater homogenous.

At Crow Butte it is anticipated that aquifer restoration activities in Mine Unit 1 will start in early 1994. It will most likely take from one to two years to complete the restoration process in Mine Unit 1. In the second phase of aquifer restoration the operator plans on using a reverse osmosis (R.O.) unit as the water treatment system. A small R.O. unit was successfully used in the pilot test program.

Once all mining is completed and the affected aquifers have been properly restored, the regulatory agencies require that the site be completely decommissioned by returning the site to its
pre-mining condition and use. To accomplish this all equipment and buildings must be removed from the site, contaminated residue in the ponds and any other radioactive materials must be taken to a licensed disposal facility, the ponds must be backfilled and recontoured, all wells must be plugged top to bottom, and all disturbed land surface must be revegetated with native grasses. Any buried pipelines must be recovered and taken to an authorized disposal site. To ensure compliance with the decommissioning requirements and the groundwater restoration standards, the operator must maintain a reclamation performance bond of sufficient size to cover the entire cost of these decommissioning activities. For the Crow Butte mine the state of Nebraska holds the $5 million reclamation performance bond submitted by the operator.

8. CONCLUSIONS

The Crow Butte ISL mine is the newest uranium production facility in the United States with commercial production starting in April 1991. The project went through a rigorous environmental licensing process which included a pilot testing phase to demonstrate the capability of restoring the water quality in the aquifer affected by the ISL mining. The project uses an alkaline (sodium bicarbonate) leach chemistry with oxygen as the oxidant. Hydrologically the ore is ideally located in a highly permeable sandstone formation sandwiched between two thick confining shale and clay layers. Capital development costs were kept to a minimum by acquiring very inexpensive process and electrical equipment from closed ISL mines. The initial start-up year is now over and the few problems that surfaced were handled without a great deal of difficulty. The current production rate of 500,000 pounds U₃O₈ per year will eventually be increased to one million pounds U₃O₈ per year. The relatively low operating costs are within the range forecasted for the project. Environmentally, the Crow Butte mine is in full compliance with all regulations and the facility recently passed an NRC unannounced three day inspection without any violations.

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References


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STUDY OF THE GEOTECHNICAL CONDITIONS OF URANIUM DEPOSITS IN UZBEKISTAN DURING EXPLORATION WORK

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Abstract

Sheet-infiltration uranium deposits formed by oxygen pressure water of artesian basins and mined using underground leaching contain not only uranium but also selenium, molybdenum, rhenium, vanadium, scandium, yttrium, lanthanoids. The paper states the principle of study of geologic and mining conditions of uranium deposits of the Republic Uzbekistan based on 40-year experience of exploration enterprise "Krasnokholmskgeologiya" of the State Committee of the Republic Uzbekistan on Geology and Mineral Resources.

The paper also states the principle of classification of natural factors and deposit zoning by operation conditions as it applies to the underground leaching method.

Peculiar features of polyelement sheet-infiltration deposits of Uzbekistan, which effect their mining by underground leaching are also described.

1. TERMINOLOGY, METHODOLOGY

Geotechnologic conditions are understood to be the complex of natural factors which substantially effect the possibility to use underground leaching, its process and results. Geotechnologic conditions are studied in order to obtain quantitative prediction of the main operation indices for the deposit on the whole and its individual parts. The indices are:

a) average concentration of metals in the solution,
b) efficiency of metal extraction,
c) amount of leaching solution, and
d) process duration.

The indices are evaluated on the basis of the study of natural factors and empiric data which characterize experimental work on useful component leaching and its results. These empiric characteristics are called geotechnological parameters and allow to obtain functional dependence between natural factors and operation indices and to make prediction for deposit operation using underground leaching (UL).

1.1 Methodology

Methodology of the study of geotechnological conditions of polyelement deposits has a number of peculiar features:

a) all ore components should be known already at early stages of exploration works (exploration-evaluation),
b) the study of the entire section of ore-bearing horizon (lithological-facial and mineralogical-geochemical mapping of sediments with determination of lithological types, geochemical and hydrogeologic setting, peculiarities of geochemical limits, determination of mineral forms of useful components),
c) the use of thermodynamic data in order to determine geotechnological setting suitable for UL,
d) grouping of natural factors and determination of ore grade (by lithology, permeability, degree of carbonatization, sulphide content and other parameters which help to select the UL method) and geotechnologic types of ore-bearing section,
e) deposit zoning according to operation conditions.

Complex geotechnological assessment of deposits is carried out with the help of drilling, the study of core sample, testing of rocks, underground water and technological solutions, mineralogical-geochemical and geochemical research, filtration and geophysical tests, special geotechnological studies which allow to understand conditions of leaching of all useful components and provide environment protection measures.

2. GROUPING OF NATURAL FACTORS AND DEPOSIT ZONING ACCORDING TO OPERATION CONDITIONS

In order to use UL for polyelement deposits natural factors should conform with the following requirements:

a) mineral forms of exogenous epigene concentrations of useful components in ore soluble in weak water reagent solutions,
b) useful component concentration in ore should be sufficient enough to obtain a ready product,
c) corresponding permeability of ore-bearing rocks,
d) chemical composition of ore-bearing rocks mainly inert to a reagent,
e) the depth of mineralization occurrence should be suitable for drilling of geotechnological holes,
f) corresponding level of underground water.

Expediency of UL depends first of all on three groups of factors:

a) geologic structure of deposit, ore and host rock composition,
b) hydrogeologic peculiarities of the deposit,
c) ore body morphology, size and parameters.

The range of natural factor values is so wide that it is necessary to group them in order to determine ore grades, technological types of ore-bearing sections, favorable and unfavorable factors for UL as well as to compare different deposits and even different parts of one deposit and to develop measures for environment protection.

In the first group of factors the most important is the composition of host rocks having different carbonate and sulphide content (in some cases coal presence). In terms of this factor there are the following types of host rock:

a) carbonate free rock and those containing a small amount of carbonate, rocks containing a small amount of sulphide (combined carbonic acid or sulphide sulfur content is less than 2%,
b) carbonaceous (sulphidic), component content being from 3 to 5%,
c) highly carbonaceous (highly sulphidic) rock, component content - more than 5%.
Sulphuric acid is usually used for rock with a carbonate content of 2%, while the carbonate method with oxidizer is used for host rocks containing 2% of sulphide; the latter also applies to organic substance. Extraction of useful components with UL from the rock rich in reducing component requires an increased amount of oxidizer. Generally, in order to extract all useful components using the sulphuric acid method, ore-bearing carbonaceous sand containing a small amount of organic or mineral reducing components is preferable what is specific to depositions of paleovalley of large rivers of Late Cretaceous and shallow sea sediments of Upper Eocene in Uzbekistan.

In terms of stripping the easiest are rhenium and molybdenum sulphide as their extraction into solution is provided by the presence of oxidizers (oxygen and hydrogen peroxide). Stripping of selenium which is represented by a native form while the second type of selenium mineralization - metal selenides is much easier to strip by acid solutions. As to the type of uranium mineralization there is only one type of oxide ore in sheet-infiltration deposits of Uzbekistan. As for vanadium there are two mineral forms: vanadium combined with uranium and uranium-free (oxide). The latter has a better stripping and higher leaching rate. Yttrium, scandium and a number of lanthanoids also have two types of concentrations: the epigenetic one which can be easily stripped by sulphuric acid solutions and the syngenetic one which is difficult to strip.

The useful components can be divided into two main groups: polyvalent (U, Re, Mo, Se, part of V) and monovalent (Sc, Y, part of V, most of REE). For leaching of the first type elements having reducing geochemical limits oxidizers play the most important role while for those of the second type having neutralization limits medium acidity play the main role.

In addition, the following from the first group of factors has the greatest effect on the UL method:

The depth of occurrence of ore-bearing horizon which effects the design of operation hole and their cost. At present the maximum operation depth is up to 700 m. There are four types of horizons: shallow - less than 100 m, moderate deep - from 100 to 300 m, deep - from 300 - 500 m, and substantially deep - more than 500 m. The last three types of horizons are predominant.

The thickness of ore-bearing horizon specifies the required amount of solution (liquid-to-solid ratio), the solution quality and the reagent amount. The thickness can be stratigraphic, that of permeable rock, effective and the thickness of water confining strata. According to the water permeable rock thickness there are the following horizons: horizons of small (20 m), medium (from 20 to 50 m) and substantial (more than 50 m) thickness. The first and the second type of horizons are predominant in the deposits of Uzbekistan.

The factors of the second group:

Permeability and water content of ore-bearing rock determines the intensity of the leaching process, the efficiency of operation holes. According to the filtration coefficient the rocks are divided into low permeable (0.1-1.0 m/day), medium (from 1 to 3 m/day), permeable (from 3 to 100 m/day), highly permeable (more than 100 m/day). As to water content the rocks are divided into those with low water content - hole discharge is less than 0.1 l/sec per meter of lowering (sandstone), moderate water content - from 0.1 to 0.5 l/sec (fine-grain sand), high water content 0 from 0.5 to 1.0 l/sec (fine and medium coarse sand), very high water content (more than 1 l/sec - coarse sand, gravel). In the deposits under consideration two last types of rocks are predominant.
The ratio of ore-bearing and barren rock permeability (coefficient of infiltration heterogeneity) is very important for UL as it allows to determine participation of barren rock in the process. According to this factor there are four types of horizons: highly homogeneous (0.75), homogeneous (0.75-0.5), heterogeneous (0.5-0.25), highly heterogeneous (less than 0.25). In the deposits of Uzbekistan the first, second and sometimes the third type are well developed. In order to evaluate the impoverishment of productive solution taking place because of barren ore it is more convenient to use a relative permeability (the ratio of the product of rock filtration coefficient into effective thickness to the product of ore filtration coefficient into their thickness). Gradation is similar to filtration coefficients.

The depth of occurrence and underground water pressure specify conditions of solution lift and feeding. Forced feeding of solution makes homogeneous distribution of discharge among pump wells difficult. According to these indices there are four types of horizons: first one - with spontaneous discharge of underground water - shallow (up to 50 m), moderate occurrence depth (from 50 to 100 m), rather deep occurrence depth (more than 100 m) (the first two types are predominant); according to the second one - head-free water, weak head - up to 100 m, medium - from 100 to 300 m, high head - more than 300 m (the two last types are predominant).

Confining layers (distribution, lithological composition, thickness) control localization, spread out and impoverishment of productive solution in the horizon. There are regional (confining aquifers and complexes) and local (confining aquifers and layers) confining layers, by the degree of persistence they are persistent (in area and thickness) and lens. It should be underlined that any mappable confining layer can play a positive role contributing to the improvement of productive solutions quality and reducing spreading over the horizon.

Conditions of underground water supply and discharge and natural speed of the flow are important for environment protection. There are deposits with local and regional sources. All deposits of Uzbekistan are located in the area of underground water transit. Discharge takes place beyond ore fields. The natural speed of underground water varies from 1 to 10 m/year depending on lithological-facial and geologic-structural conditions.

Chemical composition and the level of underground water salinity are important factors for environment protection. There are two types of ore-bearing aquifers of fresh and slightly saline or saline water. The second type is more specific for Uzbekistan.

In the third group of factors the most important for UL are the following:

Distribution of useful component accumulations (Se, V, U, Sc, Y, Rh, Re, Mo) in productive aquifer effect selection of leaching agents, their concentration, process mode and ultimately technical-economic indices of operation. According to this factor there are the following types of ore in the sheet-infiltration zone (along the route of underground water filtration): monoselenium orvanadium-selenium, scandium-yttrium-lanthanoid-uranium, scandium-yttrium-lanthanoid-uranium-rhenium, molybdenum-rhenium, monomolybdenum. As for the plan overlap of ore deposits of individual components is 60-80%. In the section of aquifer rhenium, vanadium and sometimes molybdenum mineralization have greater wide-spread occurrence as compared to uranium ore.

Productivity [amount of useful component in kg (g) per square meter of deposit area] specifies the quality of UL solution. In terms of productivity for U, Se, Mo, V there are the following ores: very poor - up to 1 kg/m², low-productive - 1-3 kg/m², ordinary - 3-5 kg/m² and highly productive - more than 5 kg/m² while for Sc, Re and some part of TR - up to 1 g/m², 1-5 g/m², 5-
10 g/m² and more than 10 g/m² respectively. These types of ore occur in all deposits, the second and third being predominant.

Structure details and variability of ore deposits in the section are not so important for UL as the area washed by solutions is substantially greater than that of ore. However, complex, multilayer structure of ore deposit causes solution impoverishment, and greater consumption of leaching agents. The shape and the size of ore deposit in the plan are more important, above all their width. There are narrow deposits (up to 100 m), moderate wide (100-300 m) and wide (more than 300 m). Other conditions being equal, wide deposits are more favorable.

In the course of study of geotechnological conditions the ore is mapped according to mineral composition and grouped by geotechnological grade, and by permeability of leaching solutions. Geotechnological types of sections of ore bearing horizons are mapped and grouped by permeability of ore and ore hosting rocks according to the relation of ore thickness to effective thickness of aquifer.

The above grouping of natural factors effecting the UL process allow to divide them into two groups - favorable and unfavorable.

Favorable factors are: compact location of polyelement mineralization in aquifer, the absence of barren layers inside ore deposit, 20 m thickness of ore-bearing horizons, homogeneous lithological-filtration section of horizon, the presence of persistent confining layers, mainly quartz composition of ore-hosting rocks, low content of harmful components (clay particles, carbonate, phosphate, coal substance, sulphides), predominance of mineral forms of uranium and other components easily stripped by water solutions of acids and alkaline metals, hosting rock permeability equal to 1 m/day, similar or greater permeability of ore and barren rocks, ore thickness-to-horizon effective thickness ratio 1:10, useful component content of more than 1 kg/m² or 5 g/m², 50 m head on productive horizon roof, the depth of underground water level of 50 m, 30°C temperature in mineralization area, unilayer structure of ore deposits in horizon section, the width of more than 100 m, the length of 1-2 km and a number of others.

Naturally, unfavorable factors are those which have worse parameters than those listed above.

In the course of study of polyelement deposits carried out for many years it was determined that individual parts of deposits are characterized by different relations of the above listed factors affecting the UL process. That is why it is necessary to carry out zonation of deposits. The purpose of zonation is the determination of the sites which have similar operation conditions.

All studies stated above are performed during detailed exploration and the results are presented in the form of geotechnological sections for each exploration section and the maps with prognosis geotechnical estimations.

3. NATURAL CONDITIONS OF POLYELEMENT SHEET-INFILTRATION DEPOSITS OF UZBEKISTAN

The deposits suitable for UL are concentrated within the large Central-Kizilkum province in Uzbekistan. They are: Severniy Eukinai, Yuzhni Eukinai, Severni Kanimekh, Lyavlyakan, Beshkak, Alendi, Terekuduk, Varadzhan, Ketmenchi, Shark, Severni Maizak, Sabirsai, Meilisai, Kendiktyube,
FIG 1 Key-map of uranium deposits of the West Uzbekistan, 1:1 250 000 scale.

Legend

Deposits in sandy-argillaceous sediments:
1. Uchkkuduk
2. Meylisai
3. Kendyktjube
4. Bakhstiy
5. Aktau
6. Surgaly
7. Severny Bukinai
8. Yuzhny Bukinai
9. Beshkak
10. Alendy
11. Aulbek
12. Terekuduk
13. Yuzhny Kanimeh
14. Varadzhon
15. Sabyrsaj
16. Kelmenchi
17. Shark
18. Agron
19. Severny Kanimeh
20. Tutly
21. Nogorno
22. Severny Maisak
23. Altyntau ore field
24. Auminza-Beltau ore field
Aktau, partially Sugrali and Uchkuduk (Fig. 1). At present Severni Bukinai, Yuzhni Bukinai, Beshkak, Ketmenchi, Sabirsai and others are being operated.

Natural conditions of deposits of the Central-Kizilkum province affecting the UL method according to the numerous data obtained by the specialists of "Krasnokholmskgeologia" who have been explored uranium deposits in Uzbekistan for more than 40 years are as follows:

The depth of ore deposit occurring varies within a wide range - from 300 m (Beshkak, Lavlyakan, most part of Severni Kukinai, part of Ketmenchi, Aitimski site of Uchkuduk) and more
than 300 m (Severnii Kanimekh, most part of Yuzhni Bukinai, Sugrali, part of Sabirsai, etc.). Mineralization is often multilayer, which is characteristic for deposits of the Bukinai-Kanimekh ore field where there are up to six ore-bearing subaquifers (Fig. 2). The ore is carbonate-free and with low carbonate content (CO₂ - up to 2.5%) (Beshkak, Lyavlyakan, Eukinai, most part of Severni Kanimekh, part of Ketmenchi, etc.) as well as carbonaceous and highly carbonaceous (CO₂ - up to 5% and more) (Sugrali, Meilisai, part of Yuzhni Bukinai, Ketmenchi and Sabirsai). The thickness of permeable rock of ore-bearing subhorizons is mainly up to 20 m (from 8 to 30 m); the thickness of uranium deposits varies from 0.5 to 20 m, mostly 3-5 m; ore thickness to permeable host rock ratio is from 1:2 to 1:10, mostly 1:3-1:7. The uranium content is from 1 to 20 kg/m², mainly 2-6 kg/m², in Meilisai, Lyavlyakan, Beshkak, uranium content is less. Uranium content of ore varies from 0.01 to 0.1% seldom higher, the average value being 0.03-0.7%. High uranium content is in Sugrali, Ketmenchi, Sabirsai formed with participation of reducing epigenesis. The width of ore deposits is 50-300 m; the minimum width is registered in Severni Kanimekh, Yuzhni Bukinai and Tereduk, etc. All ore-bearing aquifers are confining, the minimum underground water pressure on the roof is in Beshkak, Lavlyakan, partially in Ketmenchi, and the Aitim site of Uchkuduk (20-250 m). The minimum depth of underground water is in Lyavlyakan (partially spontaneous discharge of underground water), Beshkak, Alendi, Meilisai, locally in Ketmenchi, Severni and Yuzhi Bukinai (up to 50 m). The maximum depth is in Severni Kanimekh, Yuzhni Bukinai, Sugrali, partially Ketmenchi (mainly 100 and more meters). The bed water is as a rule low saline (the solid residue is 1-7 g/l).

The main geotechnological type of ore hosting rocks is sand, more seldom gravel (part of Sabirsai, Ketmenchi) and sandstones with clay (part of Lyavlyakan) and carbonate (part of Sugruli, Ketmenchi) cement. Filtration coefficient of sand is from 2 to 10 m/day, more seldom 0.1-1 m/day.

Ore hosting rocks are composed of minerals mostly inert to acid water solutions and salts of alkaline or alkaline-earth metals. As for terrigene components quartz is predominant (65-80%), feldspar (6-22%), chloride, biotite, muscovite, debris of siliceous (1-5%) and aluminosilicate (5-16%) rocks, clay minerals like hydromica with admixture of kaolinite montmorillonite (1-15%) also occur. As for authigeneous minerals the main are calcite and dolomite (up to 10%), pyrite and marcasite (up to 3.5%), sometimes siderite, ankerite, glauconite and other occur.

For many years uranium was considered to be the main useful component of deposits of Central Kizilkum. At the same time selenium (native, selinides) 0.01-0.2%, mainly 0.03-0.07%, rhenium (presumably Re₂(+) + ReO₂⁻) - 0.5-2 - up to 15 g/t, scandium (presumably hydroxide as well as Sc containing apatite, etc.) - 0.05-0.5% more seldom more; in the deposits of Bukinai-Kanimekh ore field in ore sand yttrium (up to 120 g/t), lanthanoids (up to 50 g/t), cerium (up to 150 g/t), samarium, europium up to 3 g/t, gadolinium, terbium (up to 1.5 g/t) dysprosium, erbium, thulium, ytterbium (up to 10 g/t) also occur. Useful components in sandy aquifers (subaquifers) form complex ore deposits 100-500 m in width.

By the chemical composition the ore is mainly silicate (SiO₂ - 63-85%); in Sugrali, Yuzhni Bukinai, Severni Kanimekh, Sabirsai there are the sites with substantially carbonate ore. The phosphorus content of ore is 0.03-0.4%, reaching 4-6% in some parts of Ketmenchi; the pyrite sulphur content varies within 0.05-3% (maximum value is in a number of sites of Sugruli and Sabirsai); the organic carbon content is from 0.03-0.1 to 10% (in some sites of Sabirsai and Ketmenchi). Bitumen is well developed in Sabirsai.

The rock temperature within mineralization varies from 20 to 40°C (minimum is in Beshkak, Lyavlyakan, parts of Ketmenchi and Bukinai, maximum temperature in Severni Kanimekh, Sugrali, in the north-eastern part).
CONCLUSION

The described principles of grouping of natural factors of uranium deposits make it possible:

- to draw necessary conclusion about the possibility to use underground leaching already at an early stage of geologic study;
- at the stage of detail exploration to carry out zonation of deposit in order to evaluate geotechnical parameters in technical-economic calculations;
- to carry out field underground leaching tests in one selected region and applying the results for other regions with allowance for natural factors;
- almost all polyelement sandstone deposits of Uzbekistan are suitable for underground leaching.

References

COMPLETE MINING OF URANIUM DEPOSITS IN BULGARIA
BY IN-SITU LEACHING MINING SYSTEMS
USED AFTER CONVENTIONAL MINING

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Abstract

In-situ leaching systems for complete mining of uranium deposits after conventional mining are used at sites in Bulgaria. In-situ leaching was first applied in 1981 at the Kostenets Deposit. Combined mining methods are used, i.e. the payable ore is mined by conventional underground mining method where low-grade ore, and ore left in pillars and packs, is mined by in-situ leaching. The other uranium deposits where these mining methods are used, belong to different genetic types and metalogenic zones. The following in-situ leaching mining systems have been applied in Bulgaria: underground in-situ leaching from blasted and shrunk ore with infiltration flow of the solution; underground in-situ leaching of uranium from blasted and shrunk ore with a pulsation static regime of solution flow; underground in-situ leaching from undisturbed rock mass with filtration flow of the solution. The experience gained from using in-situ leaching after conventional mining allowed to determine and grade the actual problems and disadvantages of applying this mining technique. The results have shown that the economic efficiency of this technology is high as a result of the increase in the raw material sources and life time of the deposit; the cost of 1 kg of uranium mined by this method is lower as compared to conventional mining, so that the structure of cost is different, environmental problems arising during in-situ leaching are easier to solve because of their local character, as compared to the borehole systems of underground in-situ leaching.

Uranium production in Bulgaria started 46 years ago. By 1970 the uranium deposits were mined only by conventional methods. In 1979 in-situ leaching by surface boreholes was introduced into practice. In 1981 the first industrial tests were performed for complete mining of the deposits which had previously been mined by conventional methods. Since then 23 deposits have been mined by conventional methods, 17 have been mined by in-situ leaching borehole systems and 11 have been mined by in-situ leaching systems for complete mining and extraction of the ore after conventional mining.

A particular feature of the in-situ leaching mining systems is that the rich solutions are trapped and transported through underground excavations. These systems include also uranium in-situ leaching from low-grade ores in waste piles, trenches on the surface left from the conventional mining operations.

The search for methods of increasing the raw material sources of uranium by applying economically efficient and ecologically pure methods of mining has brought about the need to start industrial testing and experimental mining by in-situ leaching in 1981 at the Kostenets Deposit, Byala Voda Mine. The good technological and economic results obtained and the experience gained at this deposit encouraged the beginning in the years that followed of wide research, design, implementation and production activities at the RARE METALS LTD. More than 10 uranium deposits in Bulgaria, such as Kostenets, Bouhovo, Dospat, Kourilo, Sborishte, Eleshnitsa, Smolyan, Zdравets, lgralishte, Smolyanovtsi, Proboynitsa were involved in the project. Fig.1. The deposits are of different genetic types and metallogenetic zones, which determines the broad range of natural conditions; host rocks - sandstones, gneisses, shales, rhyolites, granodiorites, granites, etc.; morphology of the ore bodies - sheetlike, lenticular, stockworks, veintype, veinlets; mineral composition - a wide range of uranium ores; from primary not readily soluble nauturan, coffinite to secondary readily soluble autinite, $\beta$-uranophane,uranium blacks; distribution of the uranium minerals in the ore - from disseminated to redeposited in fractures and veinlets; presence of
Fig. 1. Sites in Bulgaria where in-situ leaching mining system are being experimented

- Smolyanovci
- Proboynitsa
- Kourilo
- Buhovo
- Kostenets
- Eleshnitsa
- Zdravets
- Dospat
- Igralishte
- Smolyan
- Sborishte

Fig. 2. Period during which conventional mining has been applied
components and minerals in the host rocks and ore which are detrimental to the chosen reagent regime of leaching - from the absence of these to the presence, within one block, of high concentrations of both sulphide and carbonate minerals; the underground waters in these deposits are as a rule interstitial and with low flow rate. The water table in these deposits is low because they are drained by the development workings.

The variety of natural conditions for the individual deposits require different approaches to geological prospecting and mining:

- stages in mining the deposits;
- some have already been mined out and closed for more than 10 years, others are being mined now, and still others are in a stage of detailed exploration and development (Fig.2);
- a great variety of applied technological and technical designs providing efficient conventional mining, starting with: opening-up the deposit-adits, shafts; the applied methods of conventional mining-opencast, underground, combined; applied systems of underground mining: room-and-pillar, filling from the surface (hardening) and filling for underground quarries, shrinkage, caving of the ore, etc.

In spite of the variety of natural and technical conditions, the complex for transporting, processing and consolidation of the solution is one and is built on the module principle. The modules are provisionally two depending on the reagent regime of leaching and the particular resource capacities in Bulgaria; a module for sulphur-acid leaching necessitating corrosion-resistant materials and equipment; a module for carbonate leaching without these constraints.

For each particular case there are specific design and technical solutions - a unit for further consolidation of the rich solutions, drainage of the sorption columns, use of the natural relief for gravity transport of ion exchange resins, solutions, reagents, etc.

The following reagents found industrial application as leaching reagents, depending on the natural conditions:

- carbonate leaching: a mixture of fused-soda ash and soda bicarbonate, \( \text{Na}_2\text{CO}_3 \text{ Na}_2\text{HCO}_3 \); fused-soda ash and feed of sodium hydroxide before feeding the sorption solutions \( \text{Na}_2\text{CO}_3 \text{ NaOH} \); fused-soda ash \( \text{Na}_2\text{CO}_3 \); soda bicarbonate, \( \text{NaHCO}_3 \);
- sulphur-acid leaching, by using sulphuric acid, \( \text{H}_2\text{SO}_4 \), as a leaching reagent.

For each deposit, as well as within one block, at every mining stage a different reagent regime is applied. Laboratory tests have been performed for chemical methods and micro-biological methods of intensification - various oxidizers of \( \text{U}_{4+} \), bacterial oxidizing of bi-valent iron, but the oxidizers have not been introduced into industrial practice under the conditions of the in-situ leaching mining systems. We have a certain experience in applying physical methods of intensification squalling shots, blasting operations in irrigation boreholes, etc.

The unit for processing enriched solutions is a sorption column which is filled, irrespective of the reagent regime, with anionite resin Varion AP or AMP. The enriched resin from all mines is processed at two hydrometallurgical plants. The capacity of the resin enriched in uranium for the individual deposits varies within wide boundaries - from 3 kg/m\(^3\) up to 25-30 kg/m\(^3\). At some deposits installations have been built for sorption treatment of mine waters discharged in the hydrosphere - Kostenets, Eleshnitsa, Dospat. Recently an industrial testing installation for electrodialysis treatment of mine waters polluted with sodium carbonate was built.
Table 1. Design implemented in applying mining systems for in-situ leaching for complete mining of uranium deposits in Bulgaria after conventional mining

<table>
<thead>
<tr>
<th>No.</th>
<th>Deposit</th>
<th>Starting date of mining</th>
<th>Dept of occurrence</th>
<th>Development method</th>
<th>Leaching reagent</th>
<th>In-situ leaching system used</th>
<th>Operation of purification works</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
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<td>CM ISL</td>
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<tr>
<td>1.</td>
<td>Smolyan</td>
<td>1958 1989</td>
<td>300-500</td>
<td>shaft, adits</td>
<td>carbonate</td>
<td>Shrinkage +</td>
<td></td>
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<td>Without disturbance -</td>
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<td>Pile leaching -</td>
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<td>Flow of solution</td>
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<td>2.</td>
<td>Kourilo</td>
<td>1955 1984</td>
<td>100-150</td>
<td>adits</td>
<td>acid</td>
<td>Infiltration +</td>
<td></td>
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<td></td>
<td>Pulsation -</td>
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<td>3.</td>
<td>Dospat</td>
<td>1957 1984</td>
<td>300-350</td>
<td>shaft, adits</td>
<td>carbonate</td>
<td>Shrinkage +</td>
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<td>Without disturbance -</td>
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<td>Flow of solution</td>
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Table 1 shows brief characteristics of implemented designs for deposits where in-situ leaching mining systems are being used, i.e. at Kostenets, Dospat, Kourilo, Smolyan. The other deposits are not in operation at present because of economic and ecological reasons. We shall describe the experience gained from applying in-situ leaching mining systems at the Kostenets deposit because, on the one hand, there we have gathered a lot of practical experience, and on the other hand, at this deposit we have tested a very wide range of technical and technological solutions.

The host rock is granodiorite. The orebodies are localized along tectonic faults and peripheral faults at a depth of 15 up to 300 m from the surface. The size and shape of the orebodies vary from stockworks to lenticular bodies and veinlets. Ore minerals such as coffinite, nasturan and beta uranophane occur in fractures. The deposit is opened up by sinking a shaft at a depth of 250 m and by driving two adits (Fig. 3). The level is 50 m high. The payable uranium ore is mined by conventional methods. The main mining method was horizontal cut-and-fill. The fill is taken from nearby quarries. As some of the orebodies have a complex morphology, they are mined by shrinkage stoping, by room-and-pillar leaving behind irregular pillars and by sublevel stoping.

A stope located above the adit level 1, 15 - 25 m from the surface and mined by the conventional horizontal cut-and-fill method from nearby quarries, was further developed and constructed as follows:

- parallel blast holes from the surface and fans of blast holes from an underground working were drilled. The same holes, after blasting operations have been performed, are used for feeding, trapping and draining working and rich solutions;
- blasting and shrinkage of: haloes of non-payable ore around mined-out blocks; protective pillars (ore and barren) left during conventional mining. As a result 70 thousand tons of mined-out and shrunk ore (including the ore from blast holes and filling) were produced;
- constructing an irrigation installation which transports the working solutions to the irrigation boreholes on the block surface;

Fig. 3. In-situ leaching complex and diagram of developing the Kostenes Deposit

1 sorption complex  2 pumping station  3. main pipelines  4 buffer tank
5. blasted and shrunk ore  6. irrigation installation  7. drainage installation
on adit level 1: the excavation in front of the in-situ leaching block was barred with a concrete wall in order to eliminate a positive disbalance of the leaching solutions and the pure mine water was drained through a pipeline out of the block contours; concrete walls were erected in the excavations underlying the stope in view of creating buffer reservoirs for the rich solutions flowing from the drain boreholes.

Main pipelines were installed on the surface and in the underground workings. A pumping station, a sorption complex and a reagent storehouse were built at the entrance of adit 1.

The in-situ leaching complex was put in action in 1981. A mixture of Na$_2$CO$_3$ and NaHCO$_3$ was used as a leaching reagent. In the same year an installation for pile leaching was mounted and connected to the sorption complex.

The choice of this reagent was necessitated because when H$_2$SO$_4$ is used as a leaching reagent, Si enters the solution thus impeding the sorption processing of the rich solutions.

Under this in-situ leaching block, on lower levels, continued the mining of payable ore by conventional techniques. When the protective pillar between the two blocks, along the vertical, was reduced to 4-10 m, it was blasted thus increasing the height of the block up to 120 m (Fig.3).

The satisfactory technological and technico-economic results obtained allowed, from the beginning of 1982, to introduce in practice uranium in-situ leaching with carbonate solutions from blasted and shrunk ore in vertical closed and open chambers with infiltration flow of the solution fed and trapped by vertical, inclined and horizontal boreholes drilled from the surface or from underground excavations. A central pumping station was built on the lowest production level of the shaft where all rich solutions flow in.

Fig. 4. Change in the relative portion of uranium mined by in-situ leaching at the Kostenets Deposit
In the years that followed, the relative portion of uranium mined by this technique increased considerably (Fig. 4) and in 1989 the in-situ leaching was applied in the whole uranium production.

In view of the short periods for introducing new in-situ leaching stopes, the mining of high-grade ore by conventional methods led to the development and implementation of a special conception of the conflict-free operation of two technologies, i.e. conventional mining and in-situ leaching within the boundaries of one stope and the mine as a whole. A reasonable balance was established between the different aspects and problems such as: the effect of bulk blasting; the siting of mine workings as a result of the numerous drill holes; the flow of working and enriched solutions in main haulage drives; weakening caused by infiltrated solutions, of the protective pillars between in-situ leaching blocks (from top) and production blocks (stopes) from bottom in conventional mining; the use of conventional mining methods in view of future complete mining of reserves (pillar blasting, filling by using low-grade ore on purpose, etc.); efficient ventilation in the presence of many sources of radon; occurrence of carbonate solutions in mine waters, etc.

For in-situ leaching blocks not commercially suitable for conventional mining with the ore-bearing coefficient lower than 55% (about 20-25% of the total reserves), an in-situ leaching system without blasting the rock mass and with a filtration regime of reagent flow was tested and implemented. In recent years, the application of in-situ mining systems for leaching metal from blasted and shrunk ore in vertical closed chambers with a pulsation regime of reagent flow was introduced. Several in-situ leaching mining systems are applied, consecutively or simultaneously, within the same stope. As a matter of fact all in-situ leaching systems used in Bulgaria have been tested at this deposit. Schematic diagrams of the main versions of in-situ leaching mining systems used in Bulgaria are shown in Fig. 5. For complete uranium mining, the most widely used in-situ leaching systems are blasting and shrinkage with infiltration flow of the reagent.

Summarizing the experience gained from using in-situ leaching systems for complete uranium mining in Bulgaria, we shall try to grade the problems according to their significance regarding their economic and ecological efficiency. Such classification of the problems is arbitrary and relates to the particular mining conditions.

1. **First level of significance:**

- a system of clearly expressed and quantitatively measurable systematic criteria for assessment allowing to make engineering decisions on the efficiency of using in-situ leaching in real time. The problem appears at this level because: a) the output data on the metal content in the shrunk ore are not always reliable (e.g. there has been loss of uranium in the packing and pillars over the past years); the arbitrariness of the in-situ leaching processes which are affected by a number of not readily measurable factors. This arbitrariness is further enhanced as compared with the in-situ leaching borehole systems because of the high filtration heterogeneity of the blasted and shrunk ore and the impossibility to separate the rich solutions from in-situ blocks which are being exploited at different periods of time. Over the last few years it has become possible to solve successfully this problem both during the design stage and during the real time production control, by using the methods of probability and stochastic modelling.
- the other set of problems are technical and technological, related to the construction of an efficient irrigation system providing sufficient quantities of solution evenly distributed in the blasted and shrunk ore. The analyses have shown that usually not more than 40-50% of the ore is irrigated. There are mainly two reasons for this: a) difficult drilling of irrigation
Fig. 5. Applied in-situ leaching system for complete mining of uranium deposits in Bulgaria

1. A system for leaching uranium from blasted and shrunk ore with infiltration flow of regent. Version: 1. Vertical closed chambers (1.1 ore irrigation by boreholes drilled from underground workings; 1.2 ore irrigation by boreholes drilled from the surface);
2. Vertical open chambers: 2.1 consecutive mining of the block - first by conventional mining and then by in-situ leaching; 2.2. simultaneous mining of the block by using both conventional mining and in-situ leaching;
3. A system for underground leaching of uranium from blasted and shrunk with a pulsation static regime of reagent flow;
4. A system for underground leaching of uranium without disturbing with filtration flow of regent.

holes in disturbed rock mass and uneven distribution of the working solutions in the irrigation holes whose number is in the order of 3000-9000 per mine and are distributed in different places; b) on the other hand, in each hole the solution should be batched every 24 hours from 1 to 2.5 m$^3$ per drilling day. We can offer technical and technological solution to the two problems.

2. *Second level of significance*

These are problems related to: conflict-free operation of the two techniques, i.e. in-situ leaching and conventional mining, within the same deposit; problems of environmental protection arising from potential hazards of polluting the mine waters with working or rich solutions.
3. **Third level of significance**

These are problems related to determining an optimal reagent technological regime and optimal designs under the particular conditions.

Finally, we shall try to summarize the main results and conclusions from the implementation of in-situ leaching systems for mining uranium deposits in Bulgaria.

1. The raw material sources at the deposits have increased by 50 per cent of the previously calculated reserves as a result of using combined methods of mining the pay ore by conventional techniques and in-situ leaching of uranium from low-grade ore and losses found in pillars and packs.

2. The implementation of in-situ leaching mining systems allowed to establish a new raw materials basis as a result of the activation of commercial reserves of uranium ore left as losses in mined-out deposits many years after these have been mined by conventional methods, e.g. Dospat - 9 years after its closure, and Kourilo - 22 years after its closure.

3. Contribution to the national balance of uranium stock at deposits where the in-situ leaching systems are the only economically effective mining technology because of the low uranium content in the ore and low ore-bearing coefficient, e.g. Zdravets and Igralishte deposits.

4. From 1 up to 5 per cent of the working and rich solutions flow into the mine waters. This has necessitated the construction of purification installations for mine waters pumped out of the central mine drainage and flown into the hydrosphere.

5. As the main problems in completing the mining of uranium by in-situ leaching mining methods also have technical and organizational aspects, the experience gained can be successfully used in completing the mining of other metal deposits, e.g. copper deposits.

6. At deposits where the two techniques are being used, the prime cost of 1 kg uranium, absolutely expressed, i.e., the cost of the uranium completely mined by in-situ leaching mining systems used after conventional mining, the prime cost goes down to 30% although the uranium content in the ore is 2-3 times lower than that mined by conventional methods.

<table>
<thead>
<tr>
<th>No.</th>
<th>Costs</th>
<th>Conventional methods %</th>
<th>In-situ leaching %</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>Materials</td>
<td>5.2</td>
<td>16.0</td>
</tr>
<tr>
<td>2.</td>
<td>Electric power</td>
<td>7.1</td>
<td>9.0</td>
</tr>
<tr>
<td>3.</td>
<td>Salaries and income tax</td>
<td>37.1</td>
<td>13.9</td>
</tr>
<tr>
<td>4.</td>
<td>Depreciation</td>
<td>12.5</td>
<td>16.9</td>
</tr>
<tr>
<td>5.</td>
<td>Development operations</td>
<td>12.1</td>
<td>42.2</td>
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<tr>
<td>6.</td>
<td>Overhead</td>
<td>26.0</td>
<td>2.0</td>
</tr>
<tr>
<td></td>
<td></td>
<td><strong>100.0</strong></td>
<td><strong>100.0</strong></td>
</tr>
</tbody>
</table>
7. The structure of costs of the uranium mined by in-situ leaching as compared to uranium mined by conventional mining, is different (Table 2).

Table 2 Structure of costs per 1 kg of uranium produced by conventional methods and in-situ leaching.

Uranium production in Bulgaria was suspended by a government decree of 20 August 1992 in connection with the sharp fall of international uranium prices over the last few years. As a result, a great number of highly qualified specialists from the whole cycle of uranium production will be dismissed and special equipment fit for secondary usage will be put out of operation.

Finally, I wish to note that in the preparation of this paper I have used both my own experience and papers and reports of the science and technology reference found of the Rare Metals Ltd.
EXPERIENCE WITH THE COEXISTENCE OF CLASSICAL DEEP MINING AND IN-SITU LEACHING OF URANIUM IN NORTHERN BOHEMIA

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Abstract

The existing DIAMO extraction business in Northern Bohemia is concentrated in the eastern part of the district of Česká Lípa. Extraction using two different methods, i.e. classical deep mining and in situ chemical leaching (ISL), has been developed since the late 1960's in the broader neighbourhood of Hamr na Jezeru and Straz pod Ralskem. The purpose of both methods is the extraction of uranium concentrate from mineralisations in sedimentary rocks of Cretaceous age. In the case of deep mining, sediments with a commercial mineralisation are lifted to the surface and transported to a chemical recovery plant, where uranium is leached out. The gangue (over 99% of the original volume) is deposited in sludge ponds. In the case of chemical extraction, uranium is leached by a 2-5% solution of sulphuric acid directly into the bed in a closed process cycle by means of a network of technological wells.

It should be admitted that an erroneous concept was adopted in the development of uranium extraction in Northern Bohemia. This was the quantitative growth of both extraction processes in a small area, resulting in negative mutual impacts. No adequate processes or measures were adopted to minimize the environmental effects of DIAMO's activities (mine water treatment plants of an adequate size and quality, desalination plants for process solutions, etc.).

Continued extraction by DIAMO in Northern Bohemia depends on solving the problems of environmental protection. This condition mainly applies to ISL, whose further development has been discontinued pending comprehensive assessment of the environmental risks of the process.

1. INTRODUCTION

The North Bohemian area with its sandstone type of the U-deposits is the newest ore producing district in Czechoslovakia (Fig. 1).

Because of its relatively large resources of U (about 200 000 t) this district was considered the most lucrative one. The U-planned production had to cover all the long-term needs of the CS-nuclear electricity supply, including export to COMECO countries (COMECO - Council of Mutual Economic Aid). For this reason the grant program for the exploitation development was started here. It hasn't been reduced, despite the decrease of U prices in the western market since the 70's.

At the end of the 80's, especially after 1989, the CS-nuclear electricity supply program, and later the whole exploitation of the CS-U-ores, was re-evaluated.

Two principal factors have changed the strategy of U-production in North Bohemia. They are the continuing transformation to market conditions and the newly applied estimation criteria of U-production's influence on the environment.

The main problem for North Bohemian U-production is the coexistence of two large production complexes - classical deep mining on the Hamr deposit and ISL on the Straz deposit. These two complexes are near to each other and they have a negative influence on each other.

The results of this coexistence are higher production costs and negative influences on the environment, especially on the ground water.
2. GEOLOGY OF THE STRAZ BLOCK

Uranium concentrations in Northern Bohemia occur within sediments of the Upper Cretaceous age belonging to platform units of the Bohemian Massif (the so-called Bohemian Cretaceous Basin or Plateau).

The most important uranium concentrations were found in the so-called Straz block (Fig. 2). This block is limited by the Straz fault in the north-west, against the deeply sunken Cretaceous sediments of the Tlustec block; by the Lugian fault in the north-east, against the crystalline rocks of the Jested complex; by a belt of neovolcanic dykes, called the "Devil's wall" in the south-east and by the Hradcany fault on the south-west (Fig. 3).

The basement of the Straz block consists predominantly of epizonal metamorphosed rocks, especially of sericitic and quartzitic phyllites and quartzites (Proterozoic, and probably also some Lower Paleozoic). In the central and western parts of this block, the cataclased granitoid rocks of the Cadomian age with signs of younger hybridization occur in the basement of the Cretaceous sequence. In the south and southeast, the Cretaceous sequence occurs under Permian redbeds and volcanics (melaphyre, quartzporphyre).

The Cretaceous sediments range from Cenomanian to Turonian in the Straz block (and from Cenomanian to Coniacian in the tectonically depressed Tlustec block). The Cenomanian sequence comprises both freshwater and marine sediments. The freshwater Cenomanian strata occur only in local depressions of the pre-Cretaceous land surface while the marine strata are found within the entire area of the block.
The part in the profile over the Middle Turonian sandstone in the Tlustec-block represents an interval of a full hole drilling (Middle and Upper Turonian, and Coniacian eventually).

FIG. 2. Cross-section of the uranium district in northern Bohemia.

FIG. 3. Schematic setting of uranium deposits in the Stráž block.
The basal part of the freshwater Cenomanian is composed of flush type sediments (silty-sandy breccia, clayed siltstones). These sediments are covered by fluvial and lacustrine sandstones, above which occur siltstones deposited in a swampy environment.

Basal layers of marine Cenomanian are formed by washout sediments (subfacies of beach deposits, mostly sandstones and sandy conglomerates) followed by friable sandstones of the subfacies of inner and outer offshore. The monotonous marlstones above the Cenomanian sequence belong to the Lower Turonian. Middle Turonian sediments are represented by sandstones (clay-belong).

The average thickness of the whole complex of Cretaceous sediments in the Straz block is about 220 m. The entire area of the Straz block has been intensely disturbed by disjunctive tectonics.

The Cretaceous beds are penetrated by volcanic rocks of the Tertiary age. The volcanics mainly occur as dikes of melilitic rocks and basalt rocks.

The uranium accumulations show a stratiform nature and are confined to freshwater Cenomanian strata (fluvial and lacustrine sandstones) and especially to the basal part of marine Cenomanian (most of the uranium mineralization is located in washout sediments). Numerous ore "horizons" usually occur above paleodepressions, where the nature of sediments is, as a rule, more varied lithologically.

All uranium concentrations have been classified as uranium deposits of the peneconcordant type, but with an unusual association of the elements: U-Zr-P-Ti.

3. HYDROGEOLOGY OF THE STRAZ BLOCK

Two aquifers are located in the Straz block, as well as in the neighbouring beds of the North Bohemian Cretaceous Basin: one in Cenomanian, the other in Middle Turonian rocks (another aquifer developed in Upper Turonian to Coniacian rocks of Tlustec Block).

The Cenomanian aquifer consists of sandstones with conglomerate intercalations, the Middle Turonian aquifer is mainly filled with medium-grained sandstones.

The two aquifers are divided by a complex of Lower Turonian, forming an artesian ceiling of the Cenomanian aquifer.

The communication of ground water between the Cenomanian and the Turonian aquifer through the aquitard of Lower Turonian rocks \((k_t = n.10^{-7} \text{ to } 10^{-10} \text{ m.s}^{-1})\) depends on the difference of hydraulic potentials. The highest values of filtration coefficient \((n.10^7 \text{ m.s}^{-1})\) of the Lower Turonian rocks (obtained from evaluated extraction tests) probably indicate the effect of another kind of vertical communication than filtration through an isotropic rock. It may be communication along tectonic features or through poorly plugged boreholes.

Transmissivity was applied to describe the aquifers. Transmissivity in the Cenomanian aquifer was determined by extraction tests as \(n.10^4 \text{ m}^2 \text{.s}^{-1}\) to \(n.10^3 \text{ m}^2 \text{.s}^{-1}\). The flow of water is interstitial to fissural, with local preferences along tectonic lines cured with volcanites.
Transmissivity of the Middle Turonian aquifer is greatly varied in relation to the lithological properties, tectonic exposure and thickness of the sediments and varies from $10^{-4}$ to $10^{-2} \text{m}^2\text{s}^{-1}$. The nature of the permeability is the same as in the Cenomanian aquifer, i.e. interstitial to fissural.

The trend of the flow in the Cenomanian aquifer is to the S and SW (in undisturbed condition), with an average hydraulic gradient of 0.005.

Originally it was thought that infiltration into the Cenomanian aquifer in the Straz block only occurred at outcrops and at the boundary of the Cretaceous sediments. This was corrected as our knowledge of the hydrogeological system of the area grew. Not only the outcrops of the Cenomanian rocks nearby the Lugian fault are of importance in the infiltration, but also the significant tectonic lines limiting the area of the Straz block. According to the model calculations, so is the overflow from the Turonian aquifer, whose magnitude is dependent on the difference of the surface potentials in the two aquifers. However, a reverse flow of water also occurs, as it did in the geological history.

The drainage of the Cenomanian aquifer theoretically begins where the piezometric level of the Cenomanian aquifer begins to exceed the free water elevation of the Turonian aquifer; the area south of there can be thought of as an overflow from the Cenomanian to the Turonian aquifer along the hydraulic gradient.

This theoretical overflow begins in the Straz deposit (the area of ISL) and goes to the line of the Labe river.

The Middle Turonian aquifer is replenished by the infiltration part of any precipitation. As was mentioned above, replenishment is also possible from the Cenomanian aquifer (depending on the pressure head of the Cenomanian aquifer). The trend of the flow in the Turonian aquifer is similar to the Cenomanian one, i.e. to the S and SW. The main drainage is into surface streams and under certain conditions part of the water overflows into the Cenomanian beds (if the Turonian level is higher than the Cenomanian). The gradient varies greatly due to the surface configuration and local erosion levels. The surface of the Turonian aquifer in the area of the Straz block is predominantly free, except where it is covered by poorly permeable Quaternary. There the surface can become moderately confined.

The Middle Turonian aquifer is utilised for the supply of good quality drinking water in the whole area of the North Bohemian Cretaceous Basin, but the original radioactive Cenomanian waters in the Straz deposit prohibited their use in water management.

4. THE METHODS OF THE EXTRACTION IN THE STRAZ BLOCK

Two methods for extracting uranium have been developed in the Straz block since the late 1960's.

Classical underground mines were opened in the northern and southern part of the Hamr deposit (the Hamr and Luzice mines) and in the Brevnioste deposit (the Krizany mine). Only the Hamr mine is being exploited at the present time. The Luzice and Krizany mines were closed and flooded in the early 1990's.

The panel and fill (or room and pillar) method is used in the Hamr mine. A relatively stable deep depression in the Cenomanian aquifer has been formed during the long-term ground-water
drainage. A central dry part has been formed at the site of the deep mining. The depression is kept up by mine pumping at the level about 32-35 m³.min⁻¹.

About 50% of the U-reserves have been exhausted in the Hamr mine.

The ISL project has been in operation for 25 years (including pilot tests). Two ISL claims were exploited in the south-western and southern parts of the Hamr deposit (Fig. 4).

The development of ISL claims in the Straz deposit really began in 1972, after some initial leaching tests. Some 500 000 m² of new claims were commissioned annually in the first few years, and 200 000 - 300 000 m² annually in later years. So far 32 ISL claims have been commissioned in the Straz deposit, comprising a total area of about 6 000 000 m².

On the basis of pre-extraction research diluted sulphuric acid was selected as the most suitable leaching agent for the Straz deposit. Concentration of ca. 5% with an addition of nitrate ions as an oxidant was chosen with regard to the uranium mineralization.

Uranium in the deposit occurs predominantly in the virtually insoluble form U⁴⁺. Only after oxidation to U⁶⁺ does it dissolve into the leaching solution forming a uranyl-sulphate complex. The oxidizing environment must be maintained throughout the process to prevent reduction and reprecipitation of uranium from the solution.
After pumping the productive solutions to the surface, uranium is extracted by sorption using strong anion resins. A solution of ammonium nitrate and nitric acid is then used to recover the ion exchange resin and obtain a concentrated uranium solution. The precipitation of the final product, i.e. ammonia diuranate, is achieved by ammonia. The process of underground leaching takes place in a closed circuit. The solutions, from which uranium was extracted by sorption, are modified by the addition of $H_2SO_4$ to the desirable acidity and reused for in situ leaching.

If the original value of the Straz deposit (ISL) reserves was set at 100%, the total recoverable reserves were 45-47%. Of these, just over half (55%) have been extracted.

5. THE MAIN PROBLEMS OF THE COEXISTENCE

The area of the Straz deposit has a relatively high piezometric level (at ca. 290 m in the ISL claims). An absolute piezometric level of the Cenomanian aquifer must be kept in the ISL claims to meet the demands of the economy of extraction by airlifts i.e. extraction by compressed air. At the same time we need to keep it under the table of the Turonian aquifer to prevent accidents with the technological wells as well as any disturbance of the Lower Turonian insulation. However, the most important aspect is the need to keep technological solutions in the extraction claim.

The area of ISL with technological solutions is separated from the area of underground mining by the so called hydraulic barrier - i.e. by a line of wells, that are forming the artificial pressure watershed (Fig. 4). This watershed makes possible the coexistence of both types of extraction, although they make contradictory demands on the hydraulics of the layer (saturation-dewatering). The belated building of the hydraulic barrier in the 70's resulted in hydrochemical influences on the water in the Cenomanian aquifer, between the Hamr mine and the area of ISL.

At present the total injected volume of solutions into HB Straz (the hydraulic barrier between the area of ISL and the Hamr mine) is about 16 m$^3$.min$^{-1}$ and into HB Sveborice (the hydraulic barrier to the S of the area of ISL) is about 8 m$^3$.min$^{-1}$.

The main problem with this situation is in the standard excess of injected solutions in the area of ISL claims, causing a westward shift of the front of "diffusion" (acid saline solutions outside the extraction site). This process is to some extent controlled by pumping units in the direction of future ISL claims. This system was a part of the extraction strategy based on the possibility of lateral overacidification of new ISL sites. However, by the commission of the last ISL claim, the variant of the drainage of the solutions from the area (without recycling) will inevitably follow. This situation is expected in 2005-2010.

6. THE MAIN MISTAKES OF PAST MINING STRATEGY IN THE STRAZ BLOCK

Though pilot tests were performed before ISL was initiated, they were only intended to supply technological data for extraction.

The original concept of uranium extraction in the Straz Block was to choose between chemical or mechanical extraction after verification by pilot operations. This was changed in the 1970's, due to strategic political concerns (higher extraction targets) and collective irresponsibility. Both methods of extraction were developed simultaneously and extensively, with increasing mutual negative effects.
To suppress the image of a confused strategy, the term ‘coexistence of mechanical and ISL’ was coined: over this was to ‘optimum extraction methods to meet the conditions’. However, the technical measures necessary to secure the coexistence of both extraction methods i.e. the hydraulic barrier, the neutralizing station were never implemented in time.

The main problem of the existing ISL is mining scheme above the balance and excessive pressurizing of the Cenomanian aquifer in consequence to the activity of the Straz hydraulic barrier and later hydraulic barrier at Sveborice. It was expected to be solved by the building of a desalination plant and subsequent shift to an under-balance extraction mode, was solved in an easier way. The drainage of Cenomanian water from the forefield of in situ leaching reduced the levels of Cenomanian piezometric levels and the new leaching area were laterally pre-acidizing.

7. THE NEGATIVE INFLUENCE OF THE COEXISTENCE

The following chemicals were injected into the exploited parts of the Straz deposit in the last 25 years (3.8 mil. t of H₂SO₄, 270 000 t of HNO₃, 103 000 t of NH₃, etc.). They affected a total of 188 mil. m³ of Cenomanian water in an area of 28 km² (ca. 90% of the total saline supply remains in the ISL claim). These accumulated underground until the last 5 years, when about 1.5% of the combined salinity was removed in neutralizing units.

The hydrogeological situation in the deposit and the vertical pattern of rock permeability do not confine the circulation of technological solutions within the mineralized part of the Cenomanian formation. One should take into account that the balance mineralization does not continuously fill the entire area of Straz Block; areas with no balance mineralization are located between individual ISL claims. Due to these factors, the solutions dispersed horizontally into the close neighbourhood of the ISL claims, and vertically into the non-mineralized part of the Cenomanian formation. They still do to a limited extent.

The liquidation of the solution between the Straz hydraulic barrier and the Hamr mine, complicates the continued development of underground mining at the present time. It must be solved quickly. This water is intentionally routed to the preshifted drainage system of the Hamr mine, where it is entrapped and pumped to the surface. Part of the water is forced into the Straz hydraulic barrier, the remainder is neutralized, decontaminated and discharged into the water streams.

Solving the hydrogeologic situation, in order to increase the ecological safety of chemical mining, consists of reducing of delivery of the Cenomanian water head bearing in the area of the leached fields. This measure could be realized with the construction of a desalination plant for the process solution. This plant should have capacity of 5 m³/min. These technologies are being tested at the present time.

The changes caused by in situ leaching are practically irreversible in real time (n.10¹-10² years); the restoration will be a long-term process, which will have to be continually corrected on the basis of survey and its verification. The start of the restoration is hindered by the fact, that the key technological installations are not in place yet i.e. a unit for the disposal of residual technological solutions and that the inertia of the existing extraction process will still be felt in the transitional period.
8. THE HYDRAULIC SOLUTION OF COEXISTENCE

The main problem of coexistence is the control of the movement of technological solutions in the Cenomanian aquifer. The Cenomanian aquifer of the Straz block is a part of the complex hydrogeological structure of the Bohemian Cretaceous Basin. The large accumulation of contaminants represents a long-term potential danger for an extensive region in the southern and the south-western direction from the Straz block. With respect to the lower Turonian formation (which separates the Cenomanian and the middle Turonian aquifer) leakage has been proved. It is not possible to exclude the contamination of the middle Turonian aquifer over the long term. This is potentially very serious as this is a source of drinking water for the whole region.

With coexistent environmental safety, both the maintenance of the hydraulic gradient from the Turonian into the Cenomanian aquifer and the control of the movement of solutions is directly proportional to the deficit in the Cenomanian aquifer.

Three basic hydraulic situations are presented—the situation at the present time (H-1) and two alternatives (H-2, H-3) (Fig. 5). The following figures plot the result of mathematical modelling in the form of isolines of the Cenomanian water table with an interval of 5 m. The flow vectors of the Cenomanian aquifer are plotted there. The length of the vectors (simple arrows) corresponds to the legends.
FIG. 7. Situation H-2.
The following tables show the input and final drained or injected volumes in the main technological units for individual variants.

**Situation H-1**

The basic situation illustrating the area at present time (Fig. 6). Input parameters are obtained from the weekly reports of average injection or pumping data. The values are processed by means of spread sheet Lotus 1-2-3 and in text form directly added to the standard marginal conditions in the mathematical model. The same maps are used weekly for the monitoring of the hydraulic situation in the area and the control of hydraulic barriers.

**Main parameters:**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pumping from the Cenomanian aquifer</td>
<td>43.6 m³.min⁻¹</td>
</tr>
<tr>
<td>Injection into the Cenomanian aquifer (including Turonian water)</td>
<td>24.5 m³.min⁻¹</td>
</tr>
</tbody>
</table>

**Total deficit in the Cenomanian aquifer**

- Total injected volume into HB Straz 16.9
- Sveborice 7.6
- Total balance of ISL area 3.1 (the pumping centres in front of ISL)

**Situation H-2**

This situation proves that with a sufficient depth in the area of ISL it is possible to keep technological solutions safely in place (Fig. 7). However, drainage and disposal of ca. 5 m³.min⁻¹ of technological leaches (desalination units) is necessary. The Straz Hydraulic Barrier Straz will be operated in the existing mode with local redistribution of injection into individual wells. The hydraulic gradient from all the sides to the centre of ISL site is sufficient.

**Main parameters:**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
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</thead>
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<td>Pumping from the Cenomanian aquifer</td>
<td>48.1 m³.min⁻¹</td>
</tr>
<tr>
<td>Injection into the Cenomanian aquifer (including Turonian water)</td>
<td>16.3 m³.min⁻¹</td>
</tr>
</tbody>
</table>

**Total deficit in the Cenomanian aquifer**

- Total injected volume into HB Straz 16.3
- Sveborice 0.0
- Total balance of ISL area 4.6

**Situation H-3**

This is the atypical situation, offering a radical solution to the entire area of chemically affected Cenomanian waters by the so-called reversal of HB Straz (Fig. 8). The map of isolines of the Cenomanian aquifer water table W of HB Straz looks similar at first sight to the situation between HB Straz and the mine in Situation H-1. This analogy is due to the fact that the line of HB Straz de facto takes over the function of the forward drainage system of Level 5 of Hamr mine. The function of the barrier is diverted to the site of the old ISL claims. The maximum hydraulic gradient forms between the centre of the ISL area and HB Straz. Pumping, i.e. its reversal, lowers the water table in the Cenomanian aquifer roughly to the level of the drainage cross-tunnels of the Hamr mine. A moderate watershed is thus formed in the area between HB Straz and the drainage system of the Hamr Mine,
and water flows back to the line of the hydraulic barrier. Practically this situation takes advantage of the large-diameter wells on the line of the barrier, fit for the injection to extract technological solutions. The situation also includes the drying out of the Hamr mine western blocks.

<table>
<thead>
<tr>
<th>Main parameters</th>
<th>m³.min⁻¹</th>
</tr>
</thead>
<tbody>
<tr>
<td>- Pumping from the Cenomanian aquifer</td>
<td>48.1</td>
</tr>
<tr>
<td>- Injection into the Cenomanian aquifer</td>
<td>10.4</td>
</tr>
<tr>
<td>(including Turonian water)</td>
<td></td>
</tr>
<tr>
<td><strong>Total deficit in the Cenomanian aquifer</strong></td>
<td>37.7</td>
</tr>
<tr>
<td><em>of this</em></td>
<td></td>
</tr>
<tr>
<td>- Total injected volume into HB Straz</td>
<td>15.6</td>
</tr>
<tr>
<td>Sveborice</td>
<td>0.0</td>
</tr>
<tr>
<td>- Total balance of ISL area</td>
<td>5.2</td>
</tr>
</tbody>
</table>
IN-SITU LEACHING OF URANIUM IN CHINA

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Abstract

In China, 61% of known uranium deposits are small ones with uranium content in each deposit of less than 3000 tons and 83% of uranium deposits are in grade below 0.2%. Exploitation of these deposits naturally brings about an unbearable high production cost by conventional mining and milling method. In order to reduce the production cost and to utilize the low grade reserves, the Bureau of Uranium Mining and Metallurgy of China National Nuclear Corporation has decided to research and develop uranium deposits by cheap in-situ leaching technique since the early 1980s and made it a strategic measure in developing uranium industry. In general, there are four main types of uranium deposits in China, namely granite, volcanic rock, sandstone and carbonaceous-siliceous-pelitic. Experiments on extracting uranium from different types of deposits have been carried out. Some deposits have been transferred into productive mines with in-situ leaching process, and favorable results have been obtained in some of them.

This paper presents a brief introduction of a few practical cases in some uranium mines including in-situ leaching sandstone, underground stope leaching in deposits hosted in granite and volcanic rock.

Uranium quantity in deposits of 1000 to 3000 ton uranium content accounts for 30% of total uranium in known uranium deposits in China and that in deposits of less than 1000 ton accounts for about 31%. This means that Chinese uranium deposits are of medium or small size. Uranium quantity in deposits of 0.2% - 0.1% grade stands at nearly 52.5% of total uranium and that in deposits of less than 0.1% grade stands at nearly 31.1%. This indicates that most deposits are of medium or low grade. No doubt, exploitation of low grade ore and small deposits must bring about a high production cost. So it is a rather important matter to research for proper productive technology to reduce the production cost in mining and milling low grade uranium ores in Chinese uranium industry. In the 1970s, the Bureau of Uranium Mining and Metallurgy (BUMM) of China National Nuclear Corporation (CNNC) commenced to make experiments on in-situ leaching of uranium ores in some deposits to seek for new low-cost process and has gotten favorable experimental results. In late 1970, many factors, including declining domestic natural uranium demand, gradually increasing difficulties in excavating ores from deeper levels in some old mines and the occasional rising of the payload of employees and the price of raw materials, began to bring the production cost of natural uranium higher and higher. For the purpose of reducing the costs of natural uranium production and keeping a necessary output, in early 1980, BUMM decided to carry out research and development activities in the field of in-situ leaching, including extracting uranium by underground in-situ leaching of ores in stopes and made it a strategic measure of profitable exploitation of uranium deposits.

There are four principal types of uranium deposits in the known uranium resources in China. They are that in granite (42%), that in volcanic rock (20%), that in sandstone (21%) and those in carbonaceous-siliceous-pelitic (13%). Some researches and developments of extracting uranium
from deposits in sandstone with in-situ leaching have been carried out successfully and fair parameters have been obtained. Tests of in-situ leaching of ores in granite and volcanic rock are on-going.

After several years of effort, in-situ leaching and relevant leaching mining methods have been adopted as the sole approach to produce natural uranium in some mines, and the share of natural uranium produced from them reached 15% of the total output in 1991 and is expected to reach 30% in 1995.

1. IN-SITU LEACHING OF URANIUM ORES AT TENCHONG MINE

A uranium deposit in LianJiang sandstone basin, Tenchong County in Yunnan Province, lies ten to two hundred meters below surface. The rock contains quartz, pyriye and little muddy and carbonaceous materials. Uranium exists either as pitchblende and uranium black, or as adsorbate in other minerals. Ore-bearing stratum is permeable and filled with water under pressure. Staged tests have been done to examine the leachability of the deposit. In the preliminary test, permeability of strata, drilling technique of well, well structure, well pattern, injection and pumping technique of leach liquor, lixiviant and oxidant in leaching solution were studies.

Twelve wells of 130 mm diameter were drilled on 5.6 to 20 m spacing. That part of well in ore-bearing stratum was enlarged to 200-400 mm in diameter. Polyvinyl chlorite casing, 91 mm in diameter, 5 mm in thickness, was put into the well with perforated screen fitted at the ore-bearing horizon position. The void of the enlarged part of the well outside the perforated casing was filled with conglomerate as a filter. An artificial plug and grout were used to seal off the clearance between the casing and the well wall above the ore-bearing horizon. Permeability of the ore-bearing horizon was measured 0.19 m/d, of the overlying horizon 0.029 m/d and that of the underlying horizon 0.0102 - 0.014 m/d.

In semi-industrial test, 31 wells were drilled. Through continuous leaching in 42 days, uranium content in leachate was rising to 150 mg/l. This concentration remained unchanged for two years thereafter. Final leaching yield of uranium reached 62%. The adsorbed solution, after adding appropriate new leaching agent and oxidant, was recycled in the circuit and about 60% acid could be saved. On the basis of preliminary test and semi-industrial test, the deposit has been developed into one of the productive mines with in-situ leaching process.

2. IN-SITU LEACHING TEST OF URANIUM DEPOSIT NO. 512

A uranium deposit in Jurassic sandstone with gentle dip has been examined to study the possibility and profitability of exploitation of the ores with in-situ leaching mining since 1989. The deposit, with an average thickness of 4 m, 0.052% in grade, lies at the lower part of the aquiferous sandstone stratum, which is composed of dark grey coarse sand cemented by muddy material and has a permeability of 0.7 - 1.5 m/d. The underlying stratum of the aquifer is a dark grey sandy clay with a permeability of 0.1 m/d, thus resulting as a proper environment for mining the deposit by in-situ leaching procedure. An area of 30 000 m² of the ore deposit is set to carry out the testwork. 43 wells have been drilled from surface at a pattern 25x25 m to a depth of a few meters into the non-aquiferous clay underlying the mineralized sandstone. The well is drilled at three different diameters, 200 mm from surface to a depth of 36 m, 150 mm from 36 m to 99.7 m, 110 mm from 99.7 to the lower end. Rotary drill is used for drilling the holes of 200 mm and
150 mm in diameter and core drill for that of 110 mm. A PVC casing of 75 mm in diameter and 8 mm in thickness, about 123 mm in length is put into the well with a 3.89 m long screen positioned in the ore body as a filter, and 3.3 m long tail pipe is set below the screen to allow suspended solids to settle down. The tail pipe is sealed at the lower end. On the bottom of the 150 mm diameter well, a support ring is placed outside the casing. Clearance between the casing and the well wall is grouted from the support ring up to the well mouth. Six to twelve hours after the solidity of the cement, the well is cleaned first by pumping water with air lift for 4-6 hours and then by sucking and washing with plunger pump. Wells are used as either injection holes and production holes or exploratory holes for upgrading confidence of the reserve. A group of 5 wells in square pattern are used to do the injection and pumping test, the production well in the center and 4 injection wells at four comers.

The leaching solution is injected into the well at 2-3 atmospheric pressure by pump and the leachate is pumped out through the production well by air lift. Sulfuric acid concentration in leaching solution is 2% at the beginning and then is increased gradually to 8%. Hydrogen peroxide concentration is 0.55 g/l. When the procedure goes forward for a period of 40 days, adsorbed liquor is added to be a part of the leaching solution.

In a period of 92 days, 9828 m³ leach liquor is injected and 11 664.4 m³ is pumped out. 41.59 tons of sulfuric acid and 2.11 tons of hydrogen peroxide are consumed. The leachate concentration remains 40-59 mg/l in 33 days. The pregnant solution is sent to a pilot plant to recover uranium. Through adsorption, elution, precipitation, qualified ammonium diuranate is obtained. In 1992, the research and development work of in-situ leaching continues to go further to examine the efficient means of leaching confine control, the better pattern and spacing of the well, and the optimum composition of the leaching liquor to get more favorable processing parameters. On the basis of the advanced test, a mine will be completed in 1995 to extract uranium from ores by in-situ leaching.

3. STORE HEAP LEACHING EXPERIMENT ON URANIUM DEPOSIT IN VOLCANIC ROCK

A Mesozoic low temperature hydrothermal fissure filling uranium deposit in rhyolite and rhyolitic porphyry of volcanic origin in Zhejian Province is controlled by strata and geological structure. Uranium-bearing hematite, pyrite and fluorite fill in the fissure. Country rock near the ore body is 0.03% to 0.02% in grade. The permeability of the country rock is much below that of the ore body of 1 m/d.

A block of ore body is prepared for stope heap leaching test, and excavated by shrinkage stoping method. Closed shallow blastholes are drilled and shotted with mini second delay detonators to keep about 70% of ores less than 50 mm in size. One third of excavated ores are drawn from the stope through ore pass and send to surface heap leaching plant. Two third of ores are retained in stope and leached with acidic solution at a speed of 61 m²/hr. Non-continuous leaching procedure is used with the leach and break time ratio 1.2 : 1. After a leaching period of 215 days, extraction rate reaches 76.0%. Sulfuric acid consumption is 1.17%. The production cost per ton of natural uranium may be 20% less than that of conventional process. Now the semi-industrial test is still underway.
4. **STOPE HEAP LEACHING TEST ON URANIUM DEPOSIT IN GRANITE**

An underground mine was initially constructed to develop the low temperature hydrothermal fissure filling uranium deposit of granite type at the north foot of Chinlin Mountain, about sixty kilometers from the provincial capital. For avoiding environmental pollution of the capital, excavated ores were originally designed to be processed in a mill far from the mine. However, expensive transportation fee made the production cost of the hydrometallurgical product too high to be borne by the mill. For reducing the production cost, surface heap leaching and underground stope heap leaching tests were made in recent years. In the surface heap leaching, heaps of several thousand tons of ores, 0.1% to 0.17% in grades, were leached with sulfuric acid solution. 84-94% uranium were leached out and 76.1-84.2% were recovered.

The production cost of natural uranium was much lower than that of the original far distance transportation option. The return of investment (ROI) for mine technical reform for new option is 19.5% and the payback period is 4 years. For further lowering the production cost and improving the environmental impact, semi-industrial test of heap leaching in underground stope is being carried out. Broken ores, 90% in size of 100 mm are retained in stope at a height of 10.5 m, and leached with acidic solution. Average grade is 0.109%. After 43 days of leaching, 92% of uranium extracted. Acid consumption is 25 kg per ton ore. Test work will be finished in early 1993.
PILOT TESTING OF URANIUM IN-SITU LEACHING AT THE SMITH RANCH PROJECT

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Abstract

During the last twenty years, In-Situ Leaching (ISL) of uranium emerged as an economically attractive and environmentally preferred means for extracting uranium ores in the United States. Successful application of the technology requires extensive knowledge of the stratigraphic, geochemical, and geological nature of the particular uranium deposit. An integral and key component of such studies is often the design and operation of a modest sized field test of the ISL process. Historically such tests were a prerequisite for issuance of commercial licenses and permits by state and federal regulatory agencies.

During the 1980s, two successful pilot scale wellfields were operated at Rio Algom’s Smith Ranch Project near Casper, Wyoming. More than 131 tonnes uranium as UO₃ were produced during these tests without violation of stringently enforced environmental rules. The first of these wellfields was successfully reclaimed and the associated ground water restored to approved conditions. The second is presently being maintained on standby awaiting commercial production from adjacent areas. Post-operational coring of the ore zones in both wellfields confirmed the effectiveness of ISL. The project is now licensed for commercial ISL operations and pre-construction activities for a 907 tonnes UO₃/year operation are nearing completion.

The design, operation, and restoration/reclamation of these pilot tests is described as a case study.

1. INTRODUCTION

The Smith Ranch uranium properties of Rio Algom Mining Corp. (RAMC) are in the final development stage leading to commercial production via In-Situ Leaching (ISL). With proven reserves exceeding 16,350 tonnes (36 million lbs.) UO₃, Smith Ranch is among the largest uranium projects in the United States and will operate at a rate of 907 tonnes (two million pounds) UO₃ per year. Potential reserves will add significantly more proven reserves, and a commercial project life exceeding twenty years is likely.

Originally conceived as an underground mine, plans for Smith Ranch were reconsidered in the late 1970s as dramatic changes in the uranium market were occurring. Laboratory studies promptly demonstrated the amenability of the ore to alkaline ISL lixiviates. Following confirming studies, an extensive field testing program was initiated in 1981 and continued until 1991. Not only did these tests clearly demonstrate the excellent compatibility of the ore with the ISL system but provided an opportunity to test numerous variations in the process flow sheet and to demonstrate the post-mining restoration of native ground waters within the ore body to acceptable quality.

This paper is a discussion of this field testing program which consisted of two separate and distinct pilot ISL operations. Given the duration and scope of this program, it is not possible to cover every aspect of the work. The focus will be narrowed to the wellfield performance during both the production and ground water restoration phases.
2. REGIONAL GEOLOGICAL SETTING

2.1. Location

The Smith Ranch Project is located in the southern portion of the Powder River Basin near Douglas, Wyoming, in the western United States (Fig. 1).

The Powder River Basin is a structural basin open to the north, bounded on the south by the Laramie Range and Hartville uplift, on the east by the Black Hills, and on the west by the Big Horn Mountains and the Casper Arch. The Basin includes an area of approximately 3 100 hectares (12 000 square miles) (Fig. 2).

The Smith Ranch Project encompasses 22 027 hectares (54 068 acres) of which 6 600 hectares (16 200 acres) are licensed for commercial ISL operations (Fig. 3). Adjoining the project at its eastern boundary is the Highland ISL Uranium Project. Developed and originally operated by Everest Minerals Corporation, this ISL uranium project has continuously operated at a rate of 454 tonnes (one million pounds) U$_3$O$_8$ per year since January, 1988.

2.2. Topography

The present day topography of the Powder River Basin is the result of uplift in Pleistocene time when rejuvenated streams began down-cutting and excavating thick sequences of Oligocene, Miocene and Pliocene age sediments. The topography of the permit area is characterized by gently rolling upland areas, broad stream valleys, steep sided draws and rounded ridge crests.

2.3. Regional Geology

All of the important uranium deposits in the Powder River Basin are in Tertiary strata, that is, Paleocene Fort Union formation and Eocene Wasatch formation (Fig. 4 and Fig. 5). At the end of Cretaceous time, structural uplifts developed and continental deposition began during Paleocene time. Most of the basal Paleocene Fort Union formation rocks were derived from erosion of Cretaceous shales and sandstones and hence are mostly fine-grained clastics. By late Paleocene time erosion had cut into the crystalline core of ancestral Laramie Mountains and intermittent loads of arkosic sediments poured into the southern end of the present Powder River Basin.

In late Paleocene to early Eocene time the Powder River Basin underwent further subsidence with corresponding uplift of the surrounding mountain blocks. Deposition during this period was primarily by large, meandering streams with associated coal swamps.

In early Eocene time large amounts of coarse clastics eroded from the highlands forming large fans and braided stream deposits. Deposition of the Wasatch formation in the Powder River Basin was cyclic with periods of quiescence allowing coal swamps formation followed by periods of uplift and rejuvenation of the coarse clastic cycle. Sedimentary studies show the Granite Mountains to be the main source of clastic material with minor clastics provided from the ancestral Laramie Mountains and Hartville uplift.

Following deposition of the Wasatch formation, minor subsidence of the Powder River Basin resulted in a northerly regional dip of about 1½ degrees in the Eocene and earlier rocks.
Degradation of the area continued from middle to late Eocene with the development of a mature topography which later was buried by Oligocene deposits.

During the Oligocene, Miocene and Pliocene times large deposits of sandstones and tuffaceous sediments collected in the Powder River Basin. Vulcanism was incessant during this period with streams choked by volcanic ash. A major regional uplift took place near the close of Pliocene time and rejuvenated streams began erosion and down-cutting of the existing sediments. This erosion continued and brought about the present topography.

2.4. Local Geology

Within the permit boundary the host sandstones for uranium mineralization are the arkosic sandstone units of the upper Paleocene Fort Union formation and lower sandstone units of the Eocene Wasatch formation. The Wasatch formation is the youngest bedrock unit throughout the permit area. Thickness ranges from 61 to 91 meters (200 to 300 feet) in the northern and southern portions of the permit area to 152 meters (500 feet) in the central area. The Fort Union formation is over 305 meters (1000 feet) thick. However, only the upper 183 to 213 meters (600 to 700 feet) contains the arkosic sandstone units with associated uranium mineralization.

RAMC has named the major sandstone and shale units within permit area. Sandstone units from youngest to oldest are G, E, W, U, S, Q, UO, O, M and K. The reference for contact between the Fort Union and Wasatch formations is the base of the School Coal seam or the correlatable lignite zone present throughout permit area (Fig. 6).

Resources for the permit area are primarily in the Paleocene Fort Union formation. The O, M, and K sandstone units account for the bulk of resources while Q, S, and U sandstone units locally contain significant leachable reserves. The E sandstone of the Eocene Wasatch formation contains lesser resources. Thickness of these sandstone units ranges from 3 to 60 meters (10 to over 200 feet) with the O sandstone the thickest and most persistent.

The ore occurs as typical Wyoming roll fronts, tightly crenulating and C shaped in cross section. Typically a favorable disequilibrium ratio is encountered with a chemical/radiogenic uranium ratio of 1.1 or more. The sandstone units, depending upon thickness, inter-bedded shales, and high lime zones, can contain 1 to 20 mineral fronts with the O sandstone unit being the most complex.

3. THE FIELD TESTS

Two ISL field tests were conducted at Smith Ranch. Both used mild alkaline lixiviate systems and uranium recovery via ion (anionic) exchange resins. Test objectives were (1) to obtain hydro-metallurgical information for economic analysis of ISL and (2) to satisfy Wyoming Department of Environmental Quality (WDEQ) requirements for commercial licensing. The WDEQ requires a field test as a condition for issuance of a commercial ISL mining license. Until recently, this test had to be performed at the proposed commercial site to demonstrate environmentally acceptable operating methods, post-mining ground water quality restoration, and surface reclamation.

Both tests employed sodium bicarbonate and carbon dioxide lixiviate combined with oxygen as the oxidant. Hydrogen peroxide was briefly tested as the oxidant. The chemical selectivity of
alkaline carbonate systems for uranium solubilization, coupled with an attractive environmental compatibility, had clearly emerged as the chemistry of choice. Most uranium bearing aquifers in New Mexico, Texas, and Wyoming are of potable water quality and suitable for use as human drinking water or, at the minimum, for use by livestock (cattle, hogs, sheep). Some early test work which considered introduction of sulfuric acid into such high quality ground waters was highly controversial. To gain regulatory and public approval for pilot scale use of this or any acid in such ground waters would have been difficult in the late 1970s. Today, such approval is unlikely in the United States.

Fortunately, the geochemical and chemical characteristics of the uranium roll front deposits were such that excellent productivity was achieved with alkaline lixiviates. An added benefit of this chemistry was the modest disturbance of the natural geochemical and ground water environment. The post-leaching restoration of ground waters proved much easier and less expensive with the alkaline lixiviate.

The Smith Ranch Project field tests were designed and operated to simulate closely commercial conditions. As shown in Figure 7, these efforts began in 1980 and continued for over a decade. The Q-Sand test operated from October, 1981 through November, 1984, with ground water restoration continuing until May, 1986. The WDEQ certified complete aquifer stability and ground water restoration for the Q-Sand in August, 1987. The O-Sand test began in August, 1984 and production continued into 1991 when the test was placed on hold, pending the start of commercial operations.

The concluding effort of this testing program was post operational core sampling within both wellfields to assess the leaching effectiveness. Each test is discussed in the following sections.

4. Q-SAND

Uranium mineralization in the area of the Q-Sand pilot patterns occurs in one to three discrete roll fronts (sub-rolls). These sub-rolls, Q-1 through Q-3 (ascending), are present depending upon the thickness of the sand in any given area and thus are stratigraphically controlled. In areas of little sand, such as the western edge of the pilot area, all the mineral is confined in one zone. Where the sand has scoured downward and the thickness is greater, separate sub-rolls develop. Many times they are indistinguishable. As in most roll front systems, the sub-rolls interweave within one another while generally following the "mega-roll front" system of which they are a part.

The pilot area is unique in that two separate "lobes" of the Q-Sand roll system coalesce in this area. A southern lobe rolls to the north while a western lobe moves north-northeast. This western lobe comes around a wedge out of the Q-Sand through a paleochannel and meets the southern lobe. Mineralization in this area is from both fronts and thus grades are quite rich.

The Q-Sand wellfield was located in Section 36, T36N R74W and consisted of five production patterns. Eight ore zone monitor wells surrounded the wellfield. Within the wellfield additional monitor wells were completed in overlying and underlying aquifers. Four patterns were five-spots. The remaining pattern (QP-2) lacked one injector from being a full five-spot (Fig. 8). Pattern spacing was nominally 30 meters (100 feet) between injectors. However, the well spacing ranged from 26 meters to 33 meters due to local topography and vertical drill hole deviations (Fig. 9). Ore depth is 137 to 152 meters (450 to 500 feet).

Text cont. on p. 144.
FIG. 1. Location map.
FIG. 2. Physiography in the Powder River Basin.
TOTAL: 22 027 HECTARE
PERMIT AREA: 6 600 HECTARE

FIG. 3. Smith Ranch land holdings.
FIG. 4. Generalized geological map of Powder River Basin post-tertiary formations.
FIG 5. Geological cross-section.
FIG. 6. RAMC Sandstone Zonimg — South Powder River Basin.

PALEOCENE - FORT UNION

K
M
O
U
S
U
W
SCHOOL COAL SEAM

Eocene - Wasatch

E
G

300m
270m
240m
210m
180m
150m
120m
90m
60m
30m
0
FIG 7 Smith Ranch field tests — chronology
FIG. 8. In-situ R&D project well pattern "Q" sand deposit (section 36-T.36N., R.74W. Converse County, Wyoming).

All wells, including monitor wells were piped into a wellfield "header house" where all injection and recovery well flow meters, pressure indicators, and flow controls as well as sample ports and oxygen mixers were located. Each monitor well contained a submersible pump for sampling at the header house. Injection and recovery well pipelines were buried 1.5 meters below ground level to avoid freezing. Pregnant and barren lixiviate were pumped via buried pipeline (1.5 meters) between the header house and ion exchange facility.

Wells were completed open hole, i.e., no screen or perforated pipe was placed across the completion interval. The depth and size of this interval were determined from exploration and
development drilling. A pilot hole was drilled to within one meter of the top of this interval, logged, then reamed, casing set and cemented in place. The hole was then deepened to one meter below the bottom of the completion interval. The hole was cleaned by air surging and back flowing.

When productivity or injectivity became unacceptable during operations, wells were cleaned with hydrochloric acid or simply brushed with a stiff bristle brush and back flowed. Poor injectivity was due to calcite scaling and deposition of fines on the well bore. Completion intervals remained open and were not subject to significant caving. Calcite scaling was reduced by using carbon dioxide instead of sodium bicarbonate for lixiviate makeup. Sodium exchanges with calcium clays and gradually builds up in the system causing Figure 8 precipitation of calcite in many locations. Carbon dioxide injection relies on cations already present in the system as counter ions.

Production of the Q-Sand pilot wellfield began in October, 1981. Within two weeks, fluid flow rates were stabilized among the five recovery and ten injection wells. Throughout the test, more water was recovered than was injected to maintain a localized depression in the natural hydrostatic pressure of the ore zone. This created a pressure gradient which caused surrounding native ground water to flow toward the wells. Water levels in the surrounding monitor wells were measured monthly to evaluate the shape of this cone of depression. The influx of native waters around the perimeter confines the lixiviate within the wellfield and, hence, within the uranium bearing sands. This simple system of overproducing the wellfield is a key to controlling the lixiviate movement within the ore body and to protecting the surrounding waters from unwarranted and unnecessary perturbation.

License conditions for the test limited total injection to a maximum of 6.3 L/S (100 gpm). As shown in Figure 10, fluid injection averaged about 5.8 L/S (92 gpm) until the latter stages of the production phase in 1984. Fluid recovery during the same period averaged about 6.2 L/S (98 gpm) with net purge of near six (6) percent.
Uranium response to the alkaline lixiviate was prompt (Fig. 11). In less than one month, the uranium concentration of the produced fluid reached a maximum of near 200 mg U\textsubscript{3}O\textsubscript{8}/L before onset of the characteristic log-normal decline. By late 1983, the uranium concentration had decreased to 20 mg U\textsubscript{3}O\textsubscript{8}/L, a concentration at which most commercial operations would cease. At this point, the cumulative average uranium concentration was 90 mg U\textsubscript{3}O\textsubscript{8}/L. This average is applicable to commercial design.

A second useful comparison is the normalized volume of fluid pumped to recover the economic uranium concentrations. We describe this fluid volume as the number of reservoir pore volumes. The reservoir pore volume is defined as the area inside the perimeter injection wells times the average height of the completion intervals times the formation porosity times a horizontal and vertical sweep efficiency. Using a vertical sweep efficiency of 100% and a horizontal sweep efficiency of 110%, the reservoir pore volume for this test is 11.7 x 10\textsuperscript{6} liters (3.1 x 10\textsuperscript{6} gallons). As shown in Figure 12, 36 pore volumes of fluid were produced when the uranium concentration diminished to 20 mg/L.

The somewhat erratic nature of the uranium concentration profile is normal with the small variations resulting from flow distribution changes within the wellfield as individual injection wells were fouled with calcite scale, cleaned, and returned to normal operating conditions. The upturn in production in mid-1983 (27 pore volumes) and in January, 1984 (40 pore volumes) was a direct result of rearranging the operating wells. As shown in Figure 13, several injection wells were converted to recovery wells (Phase 2) in mid-1983 followed by a second conversion (Phase 3) in January, 1984. These changes altered the predominant fluid flow paths within the wellfield and expedited recovery of the uranium. The advantage is that more uranium was recovered at higher concentrations and, hence, at lower cost. The uranium productivity of all recovery wells throughout the test is shown in Figure 14. Only one of the original wells, QP-2, did not respond well and this is attributed to flow limitations of its associated injection wells and the influx of native ground water along its uncovered side.

The uranium was stripped from the recovered fluids by conventional anionic exchange resins in a closed three-stage upflow system. As shown in Figure 15, the extraction efficiency throughout the production phases exceeded 99% with uranium leakage through the system averaging less than 2 mg U\textsubscript{3}O\textsubscript{8}/L. Higher leakages occurred initially, but a downward trend is clear during the first year as the operating personnel gained experience with the equipment. In August, 1984, the three-stage ion exchange system was reassigned to service the O-Sand test which was starting. A new atmospheric two-stage upflow ion exchange system was installed in the O-Sand circuit. Again, we see changes in the uranium leakage as the operators gained control of the new equipment. Estimated reserves for this pilot area were 52.9 tonnes (116 540 pounds) U\textsubscript{3}O\textsubscript{8}. Sixty-seven percent of this reserve (35.5 tonnes) was recovered during the test.

It is a condition of every pilot and commercial ISL license in the United States that the native ground water must be returned to conditions at or near the original quality. These requirements are set by the regulatory agency and are specified for each major constituent and all metal components of the water.

Restoration of the O-Sand ground water was accomplished by Ground Water Sweep (GWS). When applicable, GWS has proven to be the most economical means for restoration. Water from wellfield areas with the most elevated Total Dissolved Solids (TDS) levels is selectively removed and treated for disposal. A small portion of the treated water may be reinjected into the wellfield to maintain hydrologic control. Net water withdrawal from the wellfield is increased from a few
percent during operations to thirty or more percent during restoration. This causes a strong flow (sweeping) of native ground water into and through the wellfield. This replaces the lixiviate, returning the ore zone water to near original conditions.

If ideal plug flow conditions occurred in the reservoir, the restoration would require the net withdrawal of only one pore volume of lixiviate. This, of course, was not the situation and multiple pore volumes were required for a successful restoration. Minimizing this water volume is the key to a cost effective restoration.

Restoration of the Q-Sand wellfield began in November, 1984 and was completed in May, 1986. During this period about 20 pore volumes (pv) of water were produced and treated by ion exchange for uranium removal. Half the water was reinjected for hydrologic control and the remainder (10 pv) was disposed. Restoration of major water constituents to acceptable conditions is shown in Figures 16 and 17. The specific conductance dropped from 3 000 micro-mhos/cm to below the restoration target of 827 micro-mhos/cm in 6.8 pv, carbonate from 1 100 mg/L as HCO₃ to 294 mg/L as HCO₃ also in 6.8 pv, and uranium from about 20 mg U₃O₈/L to 3.6 mg U₃O₈/L in 4.8 pv. Other key constituents responded more quickly: calcium reached its target value in 3.5 pv, sulphate in 3 pv, pH in 3 pv, and chloride in slightly more than 1 pv.

All water quality targets were achieved with less than 7 pv of net water withdrawn. Ten (10) pv were withdrawn to insure the restored water quality was both stable and at acceptable conditions.

To certify that restoration is complete, the wellfield area must be left in a quiescent condition for 12 months, subject only to monthly water sampling which tests the water quality and its stability. In May, 1987, final water samples were collected and analyzed. These data, shown in Table 1, document the successful restoration of the Q-Sand ground waters.

In September, 1990, two cores were cut within the wellfield area. Locations are shown in Figure 18. Core recovery was excellent, averaging 98.5%. Core intervals and recovery for both holes, as well as the completion interval for the nearest production well, are shown in Table 2.

Upon retrieval, the core was immediately sealed in plastic tubing to retard oxidation. Under controlled conditions, it was split, described, and photographed. Individual 0.3 meter (one foot) sections were assayed for radiometric and chemical uranium.

Core QC-2 was located 9 meters west-northwest of injection well QI-10 in the direction toward OP-5. Chemical assays (Fig. 19) indicate that the mineable uranium was leached from the shallower sands but that substantial quantities of mineral remain in a zone below 148 meters. Mineral at the top of the interval (above 144 meters) is a lower wing feature of an overlying ore zone. The ore at 144 meters is contained in a thin lignite or coal bed and not amenable to leaching. The mineral directly below (145m - 148m) is a sub-roll in a generally clean sand and was effectively leached. A thin, continuous clay layer at the base of this sand is uranium rich and isolates the underlying sands where the mineral was not effectively contacted by the lixiviate. The lixiviate preferred to move through the upper portions of the completion interval. This preferential flow through the upper portions of a thick completion interval also occurred in the O-Sand.

The second core, QC-1, was located 7.5 meters west-northwest of injection well QI-9 toward recovery well OP-4. Nearly all mineral was leached from this core (Fig. 20). The only significant residual mineral was found at 152 meters in a thin lignite and clay rich zone. Excluding this lignitic uranium, apparent recovery is well in excess of 90%.
Based on these core assays and visual observations, the Q-Sand pilot was successful in both oxidizing the ore sands and removing the uranium. Except in lower portions of QC-2, all remnant uranium was associated with clays and/or organics. These materials are either preventing or hampering mobilization of the mineral around them. The high leaching efficiency achieved in QC-1, where the sand and completion intervals were not as thick as in QC-2, demonstrates the benefits of smaller completion zones. As the thickness of the completion interval increases, so do the chances for preferential flow in selected horizontal sub-intervals. With preferential flow some mineralized zones are bypassed and left unleached, as seen in the lower portion of QC-2.

Where mining has been efficient, residual grades are less than 0.005% U₃O₈. These "tail" assays are much less than tailings from many conventional acid mills. Horizons containing clay galls and organic debris retain most of the associated mineral. These intervals have low relative permeability and high oxidant consumption capabilities. Both characteristics make it extremely difficult to oxidize and mobilize the uranium mineral. Recognition of these zones is critical in determining mineable reserves. They are commonly less than one meter (three feet) thick, relatively high grade, and associated with organics, clays, high calcium zones, and low permeability zones. However, such impermeable zones are sometimes difficult to identify on electric logs. The first two characteristics, thickness and grade, must be relied upon heavily. That is, thin, high grade (> 1%) mineralized zones are likely to be organic rich and not amenable to leaching. Such zones should not be counted as mineable reserves.

Studies of the Q-Sand pilot test ended with the coring program. The operating wells have been plugged and abandoned. Operating information obtained in this test influenced the design and operation of the O-Sand test.

5. **O-SAND**

The O-Sand is the thickest and most prolific uranium mineral host in this area of the South Powder River Basin. It is composed of fluvial sands, silts and clays. The Smith Ranch area is near the depositional axis of the basin and clay horizons are not as well developed as they are toward the basin margins. Due to this, the O-Sand is nominally 76 to 92 meters (250 to 300 feet) thick at Smith Ranch, whereas at the adjacent Highland property (away from the axis) the O-Sand exists as at least four distinct sands. This lack of distinct, continuous interbedded clay zones presents unique challenges to mining the O-Sand at Smith Ranch.

The O-Sand wellfield is located in Section 26, T36N R74W. Six production patterns make up the wellfield with five five-spots and one half five-spot which lacks an injector (Fig. 21). Pattern spacing is 36.6 meters (120 feet) between injectors. Surrounding the wellfield are five ore zone monitor wells. Within the wellfield one monitor well is completed in the first aquifer above the ore zone and a second in the first underlying aquifer. Production is from the O-4 through O-7 sands which are in the lower half of the O-Sand proper at a depth from 198 meters (650 feet) to 229 meters (750 feet). Open hole completion intervals range from 20 to 26 meters (66 to 85 feet) with a mean of 21.6 meters (71 feet).

Uranium mineralization occurs within four sub-rolls. Each sub-roll is a unique individual roll front yet interrelated with the others. These fronts are typically narrow and very sinuous, repeatedly passing back and forth across one another. Thus, in some instances, all four sands contain ore grade mineralization in the same drill hole. This characteristic has led many ISL operators to assume that the ore grade mineral is much thicker and wider than it is.
Design features were similar to the Q-Sand test. Five-spot wellfield patterns with open hole completions were used. All pipelines were buried to avoid winter freezing. Conventional anionic exchange resins and equipment were used for uranium recovery. Oxygen, sodium bicarbonate, and carbon dioxide were the leaching chemicals, while hydrogen peroxide was the yellowcake precipitation reagent. Regulatory requirements were similar to the total lixiviate injection rate limited to 9.5 L/S (150 gpm) rather than 6.3 L/S (100 gpm). Operating and monitor wells were drilled and cased during July, 1982. Water quality sampling and hydrologic testing information for the license application was subsequently gathered.

Regulatory approval to operate was received in mid-1984 and test operations began in August, 1984. A prompt response to the alkaline lixiviate followed as the uranium content of the produced fluids quickly rose to a peak of 105 mg U₃O₈/L (Fig. 22). Production continued at a high rate until early 1986 when the uranium concentration dropped to 40 mg U₃O₈/L. This drop occurred as a higher wellfield purge flow rate was being tested. The increased flow of native ground waters diluted the effectiveness of the leaching. A reduction in the purge flow rate to ±5% and an increased bicarbonate concentration of the injection fluid restored the uranium concentration to above 70 mg U₃O₈/L in late 1986.

From August, 1984, through June, 1989, uranium production averaged 67 mg U₃O₈/L. In early 1987, the cumulative average uranium concentrations for the six recovery wells were: OP-1 - 82 mg/L, OP-2 - 53 mg/L, OP-3 - 76 mg/L, OP-4 - 81 mg/L, OP-5 - 60 mg/L, and OP-6 - 66 mg/L. These relatively uniform averages and the steady, almost flat, uranium concentration profile are likely to result from the wide completion intervals employed in the wells.

The reservoir pore volume was calculated to be 80 x 10⁶ liters (12.1 x 10⁶ gallons), about four times that of the Q-Sand test. With only a fifty percent increase in the authorized injection rate (9.4 L/S vs. 6.3 L/S), a significantly longer period of economic production was expected. If the O-Sand performance is viewed on a pore volume rather than time basis (Fig. 23) we see that at 30 pv, the uranium concentration had dropped to 45 mg U₃O₈/L with a cumulative average of 67 mg U₃O₈/L. These values compare with Q-Sand test values of 60 mg U₃O₈/L and 90 mg U₃O₈/L, respectively, at the same pore volumes (Fig. 12). The larger completion intervals of the O-Sand wellfield are likely the cause of the lower values. Large completion intervals in these sedimentary sandstones are susceptible to preferential flow through one or more of the sub-roll front sand units. As each sand unit is depleted of uranium, it continues to accept lixiviate to the detriment of other mineralized sands. Lixiviate is now traveling through barren or near-barren sands only to dilute the uranium concentration from mineralized zones when it arrives at the recovery well. Thus, the wellfield never achieves the very high peak uranium concentration and the economic life is prolonged as these barren zones continually divert lixiviate from the mineralized zones. The different and distinct shapes of the uranium concentration profiles of the two tests are a measure of dilution’s impact on the O-Sand performance.

Production of the O-Sand pilot continued into 1991 at which time the test was placed on hold pending the start of commercial operations. Estimated reserves for the six patterns are 136 tonnes (300 000 pounds) uranium of which 71% of the reserves have been recovered.

To assess the testing program and to aid planning for commercial development of this area, five cores were cut within the area in 1990. The location of each hole is shown in Figure 24. Each was offset to a nearby production well with only hole OC-1 located outside a pattern. Each core
FIG. 10. Q-sand flow rates.

FIG. 11. Q-sand production history.
FIG. 12. Q-sand production history.

FIG. 13. Q-sand pilot area production sequence.
URANIUM CONC. (mg/L - U3O8)

FIG. 14. Q-sand production history; individual well data.

FIG. 15. Q-sand pilot ion exchange extraction efficiency.
FIG. 16. Q-sand restoration data.

FIG. 17. Q-sand restoration data.
FIG. 18. Q-sand pilot area; Wellfield layout and core hole locations.

FIG. 19. Smith mine — Q pilot (core hole QC-2 — eU308 vs cU308).
FIG. 20. Smith mine — Q pilot (core hole QC-1 — eU308 vs cU308).

FIG. 22. O-sand production history.

FIG. 23. O-sand production history.
**TABLE 1. Q-SAND ISL PILOT RESTORATION SAMPLES**

**AQUIFER STABILITY DEMONSTRATION ANALYSES (2)**

<table>
<thead>
<tr>
<th>RESTORATION TARGET VALUES (1)</th>
<th>QP-2</th>
<th>QP-4</th>
<th>QP-5</th>
<th>QI-5</th>
<th>QI-7</th>
<th>QI-8</th>
<th>QI-9</th>
<th>QI-10</th>
<th>AVG.</th>
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</thead>
<tbody>
<tr>
<td>Arsenic</td>
<td>0.05</td>
<td>0.04</td>
<td>0.00</td>
<td>0.007</td>
<td>0.003</td>
<td>0.00</td>
<td>0.007</td>
<td>0.006</td>
<td>0.026</td>
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<tr>
<td>Boron</td>
<td>0.54</td>
<td>0.09</td>
<td>3</td>
<td>0.15</td>
<td>0.16</td>
<td>6</td>
<td>0.15</td>
<td>0.13</td>
<td>0.16</td>
</tr>
<tr>
<td>Iron</td>
<td>0.3</td>
<td>0.17</td>
<td>0.12</td>
<td>0.02</td>
<td>0.1</td>
<td>0.15</td>
<td>1.22</td>
<td>0.05</td>
<td>0.03</td>
</tr>
<tr>
<td>Manganese</td>
<td>0.09</td>
<td>0.03</td>
<td>0.23</td>
<td>0.08</td>
<td>0.09</td>
<td>0.06</td>
<td>0.03</td>
<td>0.05</td>
<td>0.04</td>
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<tr>
<td>Radium-226</td>
<td>923</td>
<td>230</td>
<td>0.07</td>
<td>556</td>
<td>475</td>
<td>0.05</td>
<td>492</td>
<td>608</td>
<td>778</td>
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<tr>
<td>Selenium</td>
<td>0.029</td>
<td>0.006</td>
<td>382</td>
<td>0.004</td>
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<td>292</td>
<td>0.003</td>
<td>0.005</td>
<td>0.002</td>
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<td>Thorium 230</td>
<td>5.62</td>
<td>3.6</td>
<td>0.00</td>
<td>4.7</td>
<td>2.5</td>
<td>0.00</td>
<td>3.0</td>
<td>2.7</td>
<td>2.5</td>
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<td>Uranium</td>
<td>3.7</td>
<td>0.8</td>
<td>2</td>
<td>4.3</td>
<td>1.7</td>
<td>1</td>
<td>0.6</td>
<td>0.7</td>
<td>0.9</td>
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<tr>
<td>Bicarbonate</td>
<td>294</td>
<td>234</td>
<td>244</td>
<td>268</td>
<td>273</td>
<td>239</td>
<td>268</td>
<td>278</td>
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<tr>
<td>Calcium</td>
<td>120</td>
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<td>Carbonate</td>
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<td>0</td>
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<td>0</td>
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<td>0</td>
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<tr>
<td>Chloride</td>
<td>250</td>
<td>18</td>
<td>15</td>
<td>17</td>
<td>17</td>
<td>11</td>
<td>16</td>
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<tr>
<td>Magnesium</td>
<td>26</td>
<td>20</td>
<td>17</td>
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<td>18</td>
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<td>20</td>
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<tr>
<td>Potassium</td>
<td>23</td>
<td>8</td>
<td>8</td>
<td>8</td>
<td>8</td>
<td>7</td>
<td>8</td>
<td>9</td>
<td>7</td>
</tr>
<tr>
<td>Sodium</td>
<td>41</td>
<td>39</td>
<td>54</td>
<td>42</td>
<td>38</td>
<td>26</td>
<td>43</td>
<td>39</td>
<td>22</td>
</tr>
<tr>
<td>Sulphate</td>
<td>250</td>
<td>135</td>
<td>127</td>
<td>131</td>
<td>127</td>
<td>111</td>
<td>131</td>
<td>139</td>
<td>119</td>
</tr>
<tr>
<td>Conductivity</td>
<td>827</td>
<td>656</td>
<td>661</td>
<td>670</td>
<td>666</td>
<td>608</td>
<td>691</td>
<td>622</td>
<td>563</td>
</tr>
<tr>
<td>TDS</td>
<td>571</td>
<td>450</td>
<td>431</td>
<td>466</td>
<td>449</td>
<td>384</td>
<td>472</td>
<td>490</td>
<td>399</td>
</tr>
<tr>
<td>pH</td>
<td>6.5-8.6</td>
<td>7.04</td>
<td>7.02</td>
<td>6.97</td>
<td>6.63</td>
<td>7.07</td>
<td>7.02</td>
<td>6.75</td>
<td>7.2</td>
</tr>
</tbody>
</table>

(1) Restoration target values as approved by NRC in License Amendment No. 11 and Wyoming DEQ in a letter of July 23, 1984.

(2) All values are mg/liter except RA-226 and TH-230 (pCi/L), pH, and conductivity.
TABLE 2

<table>
<thead>
<tr>
<th>Hole</th>
<th>Core Interval</th>
<th>Internal Cut</th>
<th>Core Recovered</th>
<th>Percent Recovery</th>
<th>Offset Well</th>
<th>Completion Interval</th>
</tr>
</thead>
<tbody>
<tr>
<td>QC-1</td>
<td>146-154m</td>
<td>8m</td>
<td>7.76m</td>
<td>97</td>
<td>QI-9</td>
<td>146-153m</td>
</tr>
<tr>
<td>QC-2</td>
<td>140-153m</td>
<td>13m</td>
<td>13m</td>
<td>100</td>
<td>QI-10</td>
<td>142-157m</td>
</tr>
</tbody>
</table>

TABLE 3

<table>
<thead>
<tr>
<th>Oxidation State</th>
<th>Hole OC-4</th>
<th>Hole OC-1</th>
</tr>
</thead>
<tbody>
<tr>
<td>% eU308 - radiometric</td>
<td>0.12</td>
<td>0.16</td>
</tr>
<tr>
<td>% U308 - chemical</td>
<td>0.002</td>
<td>0.35</td>
</tr>
<tr>
<td>Intergranular Clay</td>
<td>1.2%</td>
<td>4.0%</td>
</tr>
<tr>
<td>Visible Pore Space</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Primary</td>
<td>7.6%</td>
<td>8.0%</td>
</tr>
<tr>
<td>Secondary</td>
<td>13.1%</td>
<td>10.0%</td>
</tr>
</tbody>
</table>

was cut from above the completion zone of the nearby production well and through the production zone (the O-4 through O-7 sands). This allowed inspection of the overlying mineralized sands and provided information concerning possible vertical migration of lixiviate.

The first cores were cut with a 4.6 meter, 7.6 cm diameter Christensen core barrel which resulted in poor recoveries. Subsequent cores were cut with a shorter (3 meter) core barrel and recovery ranged from 92% to 100% with an average of 97%. The core was immediately sealed in plastic tubing to retard oxidation. Under controlled conditions, it was split, described, and photographed. The gamma log mineralized intervals were assayed for radiometric and chemical uranium. A reduced sample (hole OC-1) and an oxidized sample (hole OC-4) were also subjected to petrographic analysis (Table 3).

The sands in both are conducive to solution mining. There is sufficient porosity. The intergranular clays which provide lithification for these sands are relatively stable illite or a mixed layer of illite/smectite. No chemical uranium was detected in the oxidized sample but was found to be associated with the intergranular grain coating clay in the reduced sample. These results reveal interesting differences between the oxidized and reduced samples. While uranium has been quantitatively removed from the oxidized sample, the reduced sands are still enriched in uranium. The reduction in intergranular clays in the oxidized sample indicates the clays have been mobilized during leaching. This is consistent with the general observation that oxidized sands are less lithified than reduced ones where clays are the matrix "cement". This characteristic will affect both borehole wall stability and fluid flowpaths during leaching operations.

Chemical analyses of the five cores indicate that mineable uranium minerals remain in only certain vertical and horizontal zones. Two cores (OC-2 and OC-3) contain ore grade mineral in the lower portion of the cored interval whereas the other two cores (OC-4 and OC-5) are extensively oxidized and nearly void of mineralization. The fifth core (OC-1) is located outside the wellfield and
exhibited no lixiviate contact. It does, however, contain significant uranium reserves which will be exploited by future commercial operations.

By design, cored intervals were wider than completion intervals of adjacent wells to facilitate sampling of the shallower zones to test for vertical movement of the lixiviate. The only evidence of vertical movement was found in OC-3 where the uranium was leached up to one meter above the well completion interval. In general, the lixiviate remained confined to the completion intervals.

In the intervals where leaching was most efficient, residual grades were below 0.01% U₃O₈. The best example is OC-3 (205 to 207 meters). This zone, along with another in OC-2 (212 to 221 meters), are intervals of significantly high eU₃O₈ values which show total U₃O₈ depletion. Many intervals show only partial oxidation and leaching of ore grade intercepts. Based on the remnant ore and coloration of the core samples, leaching has been more efficient in the southern three patterns (OP-2, OP-3, and OP-6). In the other patterns indications are that only partial oxidation and leaching of the lower sub-rolls has occurred. Additional reserves within these patterns will be recovered during commercial operations.

Similar to the Q-Sand areas, thin intervals of high grade mineral were encountered which were not leached. They are intimately related to lignites, clays, or other sediments of low permeabilities and can occur anywhere within a sand. Characteristically they are thin (less than one meter thick) and relatively high grade (>1%) when compared to the mineral in the rest of the hole.

The O-Sand pilot test, like the Q-Sand before it, has demonstrated the amenability of the Smith Ranch uranium ores to ISL mining. In addition, the feasibility of mining thick, multiple roll-fronts has been proven. These tests provide a solid data base from which commercial plans are being developed.

6. SUMMARY OF TEST FINDINGS

Like conventional mining, the goal of ISL mining is to produce the economically optimum quantity of the mineral resource and to achieve this production in an environmentally sound and acceptable manner.

The two field pilot tests at Smith Ranch have demonstrated that:

- ISL uranium mining can be successfully operated under stringent environmental regulations with excellent productivity.
- The uranium minerals are amenable to the environmentally benign oxygen and carbon dioxide fortified lixiviate.
- Equally important, the tests have provided valuable information applicable to increasing uranium recovery at lower costs. These long duration tests clearly show that uranium in relatively clean sands can be quantitatively leached, leaving behind mineral grades below 0.01%. Conversely, these tests clearly prove that uranium resources associated with organic rich shales and clays are not mineable reserves for alkaline ISL operations. Understanding and applying these concepts in the planning of commercial operations can result in significant savings in wellfield investments.
The impact of wide completion intervals for injection and recovery wells is demonstrated by comparison of the two tests. While the narrow completion intervals of the Q-Sand lead to more efficient uranium recovery, the problem remains as to the best design of completion intervals in thick ore zones such as the O-Sand. Upon careful inspection, these thick sedimentary ore zones are often found to consist of several distinct sub-rolls within a larger mega-roll framework. A sub-roll or group of sub-rolls is usually within the efficient completion interval thickness (2 to 7 meters) and can be leached independent from the other sub-rolls.

Finally, the advantage of rearranging operating wells during production was illustrated in the Q-Sand test. By changing injection wells to recovery wells, and vice versa, the movement of lixiviate within a wellfield can be redirected to expedite uranium recovery.

7. CONCLUSIONS

The objectives of the Smith Ranch field testing program were to demonstrate environmentally sound mining methods, restore the altered ground water to regulatory standards, and develop the technical and operational basis for a commercial ISL project. In every respect, the Q and O-Sand tests were unparalleled successes. More than 131 tonnes U₃O₈ of uranium were recovered without a single violation of the stringent Wyoming and Federal rules. The ground waters of the Q-Sand area were restored using ground water sweep to demonstrate not only the technical but also the economic viability of restoration. The engineered systems withstood the harsh Wyoming winters. The low cost lixiviate chemicals (oxygen and carbon dioxide) and open hole well completions were shown to be both effective and applicable to commercial operations. Wellfield productivity in terms of both recovery and cumulative average uranium concentrations provided a firm base for developing the long term, large scale mine plan which accompanies commercial operations.

Conventional anionic exchange resins systems (upflow and downflow) were shown to function well as was the hydrogen peroxide induced precipitation of yellowcake. All such information contributes to an optimized commercial process design.

Perhaps the most important feature of any pilot program is the opportunity to assess critically design concepts and to then seek means to improve these concepts. While the wellfield performance of both tests was excellent, coring of the wellfields provided not only confirmation of extremely high recoveries in selected sands but provided equally important insight into why some uranium was still in place. This information is invaluable to our continuing search for the optimum wellfield design.

These pilot tests have provided a wealth of information which enhances our commercial planning and greatly reduce the risks inherent with new mines.
COMPARATIVE STUDY OF LEACHING BEHAVIOUR OF VARIOUS LEACHANTS IN ISL TEST ON QUBUL KHEL URANIUM ORE BODY

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In-Situ Leach Mining and Processing Project,
Pakistan Atomic Energy Commission,
Islamabad, Pakistan

Abstract

Exploitable uranium ore reserves had been delineated by drilling in Siwalik sandstones near Qubul Khel Village NWFP. Incompetent host rock, poor ground conditions and large water inflows made the conventional underground and open pit mining impracticable and hazardous. As the ore body is porous, permeable and ideally located below water table, the low cost ISL technique can be applied for exploitation of the metal values.

This paper presents detailed information on laboratory conventional leaching and column pack leach tests using various leachants. A comparative study of uranium leaching behaviour was also made by using various alkaline lixiviants employing conventional five spot pattern. The main objective of this study was to investigate the suitable leachant and its concentration through field test which does not foul the surface lines, submersible pumps and strata.

Laboratory leaching tests confirmed that $U_3O_8$ could be leached efficiently with a recovery of about 95%, 83% and 84% respectively by using $\left(\text{NH}_4\right)_2\text{CO}_3$, $\text{Na}_2\text{CO}_3 + \text{HCO}_3$ and $\text{H}_2\text{SO}_4$ as lixiviants.

In alkaline system ammonium carbonate, sodium carbonate-bicarbonate mixture and sodium bicarbonate solutions were found to be the most effective lixiviants. The column pack leaching tests revealed that 95%, 90% and 73% $U_3O_8$ recovery could be achieved with 25 g/l $\left(\text{NH}_4\right)_2\text{CO}_3$, 10.0 g/l $\text{Na}_2\text{CO}_3 + 5$ g/l $\text{NaHCO}_3$ and 10 g/l $\text{NaHCO}_3$ solutions respectively in 96, 120 and 216 hrs. leaching time.

The laboratory column pack leaching test proved the applicability of ISL uranium leaching process. A five spot pattern field test was, therefore, started in early 1990 using sodium carbonate and sodium bicarbonate solution (10 g/l + 5 g/l) as lixiviant and hydrogen peroxide (1 g/l) as oxidant. Although 160 ppm peak $U_3O_8$ concentration in the production stream was achieved but it resulted in scaling due to precipitation of calcium carbonate in the strata as well as in the submersible pumps and delivery pipes. In view of the above problem the mode of pumping was replaced with airlifting which resulted in the recovery of about 65% $U_3O_8$ in the test area.

However, frequent cleaning of the airlifting pipes necessitated to test sodium bicarbonate as lixiviant instead of $\text{Na}_2\text{CO}_3$ and $\text{NaHCO}_3$ mixture to maintain high solubility of calcium at comparatively lower pH.

Another five spot pattern was, therefore, developed at the site and leaching was started in January, 1992, initially with 10 g/l $\text{NaHCO}_3$ and 2.5 g/l oxygen as oxidant. Three months test operation indicated that scaling due to calcium carbonate could be minimized to a great extent using $\text{NaHCO}_3$ as lixiviant in place of $\text{Na}_2\text{CO}_3$ + $\text{NaHCO}_3$ mixture. The maximum $U_3O_8$ concentration in the pregnant leach solution was recorded so far, as 90 ppm.

INTRODUCTION

Various types of laboratory, column pack and field tests have been conducted to study the comparative leaching behaviour of Qubul Khel uranium ore. These types of tests are necessary for a more complete understanding of leaching parameters for economic extraction of low grade ore deposit by in-situ uranium leaching technique.

The laboratory tests are useful to evaluate the leaching rate & recovery of uranium from ore with different concentrations of leachants. However, column pack tests provide the effectiveness of selected lixiviants which is of critical importance to the success of an in-situ leaching operation.
The field test provides the effectiveness of lixiviants and informations regarding the operational problems such as mode of pumping liquid from production well, scaling problem in the strata, submersible pump and in the plant circuitry.

The experiments described in this report were systematically conducted using the core samples of Qabul Khel ore body. The laboratory leaching tests were carried out with various lixiviants such as ammonium carbonate, sodium carbonate & sodium bicarbonate mixture and sulphuric acid employing different concentrations. The column pack tests were performed in glass columns. The concentration of uranium in pregnant solution and permeability of ore zone were measured by using ammonium carbonate, sodium carbonate-bicarbonate and sodium bicarbonate as lixiviants to investigate the effect of various contaminants such as sodium, calcium, chloride and other leaching cations on permeability of ore zone and the ease of restoring the ore bed after leaching is discontinued.

Two independent five spot pattern field tests were conducted using a mixture of 10 g/l $\text{Na}_2\text{CO}_3$ + 5.0 g/l $\text{NaHCO}_3$ and only 6.0 g/l were used as oxidants in these field tests. The pregnant leach liquor was analysed for its contaminants and the results are discussed in detail to investigate the leaching behaviour of lixiviants.

GEOLOGY AND HOST FORMATION OF URANIUM ORE BODY

Geologically the host rock is the part of Dhok Pathan Formation comprising of alternate sandstone and shale beds laid down by braided river systems, deposited during Middle and Upper Miocene time. The shale beds are up to 4 meters in thickness. Uranium mineralization is hosted by a thick bed of sandstone but some uranophane mineralization is associated with the upper most part of shales as a result of reprecipitation on its surface.

The sandstones are light grey in colour, fine to medium grained, very poorly cemented, moderately compacted and composed of quartz, feldspar, mica, amphibole, garnet, magnetite, epidote and clay minerals. There are some calcified hard sandstone bands at various levels. The dip of the strata varies from 20° to 30°.

TEST SAMPLE

The test sample was obtained by blending the cores from the ore zone of a number of drill holes and used in the laboratory and column pack leaching tests. The granulometry of homogenized sample is presented in Table-1.

The blended ore sample was subjected to X-ray diffraction and chemical analysis for the characterization of the ore. Mineralogically the main uranium minerals are characterised as uraninite, coffinite and uranophane $\text{Ca (UO}_2\text{)}_2 \text{(OH)}_2 \text{(SiO}_3\text{)}_2 \text{5H}_2\text{O}$ and the mineral assemblage includes quartz, calcite, mica, amphibole and clay minerals etc. $\text{U}^{4+}$, $\text{U}^{6+}$ analysis carefully conducted on waxed core ore sample indicated the presence of 80% $\text{U}^{4+}$ and 20% $\text{U}^{6+}$. However, the ore sample when exposed to atmosphere readily gets oxidized to $\text{U}^{6+}$. The detailed chemical analysis of the ore is given in Table-2.
### Table 1: Granulometric Analysis of Core Ore Sample

<table>
<thead>
<tr>
<th>S. No.</th>
<th>Mesh No.</th>
<th>Wt. retained (g)</th>
<th>%Wt.</th>
<th>%U3O8</th>
<th>% Distribution (U3O8)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>+30</td>
<td>6.00</td>
<td>3</td>
<td>0.066</td>
<td>3.5</td>
</tr>
<tr>
<td>2.</td>
<td>-30+50</td>
<td>52.00</td>
<td>26.4</td>
<td>0.052</td>
<td>24.4</td>
</tr>
<tr>
<td>3.</td>
<td>-50+100</td>
<td>28.00</td>
<td>14.2</td>
<td>0.035</td>
<td>8.8</td>
</tr>
<tr>
<td>4.</td>
<td>-60+100</td>
<td>48.00</td>
<td>24.4</td>
<td>0.051</td>
<td>22.1</td>
</tr>
<tr>
<td>5.</td>
<td>-100+140</td>
<td>24.00</td>
<td>12.2</td>
<td>0.036</td>
<td>7.6</td>
</tr>
<tr>
<td>6.</td>
<td>-140+200</td>
<td>12.00</td>
<td>6.1</td>
<td>0.032</td>
<td>3.5</td>
</tr>
<tr>
<td>7.</td>
<td>-200+270</td>
<td>6.00</td>
<td>4.1</td>
<td>0.044</td>
<td>3.2</td>
</tr>
<tr>
<td>8.</td>
<td>-270+325</td>
<td>5.00</td>
<td>2.5</td>
<td>0.056</td>
<td>2.5</td>
</tr>
<tr>
<td>9.</td>
<td>-325</td>
<td>14.00</td>
<td>7.1</td>
<td>0.186</td>
<td>24.6</td>
</tr>
</tbody>
</table>

Feed: 127.00  100  0.0554  100

### Table 2: Chemical Analysis of Blended Core Ore Samples of Quibil Khel Ore Deposit

<table>
<thead>
<tr>
<th>S. No.</th>
<th>Component</th>
<th>Wt. %</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>SiO₂%</td>
<td>69.81</td>
</tr>
<tr>
<td>2.</td>
<td>TiO₂%</td>
<td>0.34</td>
</tr>
<tr>
<td>3.</td>
<td>Al₂O₃%</td>
<td>11.77</td>
</tr>
<tr>
<td>4.</td>
<td>Fe₂O₃%</td>
<td>1.43</td>
</tr>
<tr>
<td>5.</td>
<td>FeO%</td>
<td>1.44</td>
</tr>
<tr>
<td>6.</td>
<td>MnO%</td>
<td>0.08</td>
</tr>
<tr>
<td>7.</td>
<td>CaO%</td>
<td>4.91</td>
</tr>
<tr>
<td>8.</td>
<td>MgO%</td>
<td>2.11</td>
</tr>
<tr>
<td>9.</td>
<td>Na₂O%</td>
<td>1.15</td>
</tr>
<tr>
<td>10.</td>
<td>K₂O%</td>
<td>1.96</td>
</tr>
<tr>
<td>11.</td>
<td>FeO₅%</td>
<td>0.13</td>
</tr>
<tr>
<td>12.</td>
<td>H₂O-%</td>
<td>1.41</td>
</tr>
<tr>
<td>13.</td>
<td>H₂O₂-%</td>
<td>0.1</td>
</tr>
<tr>
<td>14.</td>
<td>CO₂%</td>
<td>3.06</td>
</tr>
<tr>
<td>15.</td>
<td>U₃O₈%</td>
<td>0.06</td>
</tr>
</tbody>
</table>

Total: 99.67

### Table 3: Results of Multistage Laboratory Leaching Tests

<table>
<thead>
<tr>
<th>S. No.</th>
<th>Lixiviant Type and Concentration</th>
<th>Recovery % 1st Stage 40 hrs</th>
<th>Recovery % 2nd Stage 50 hrs</th>
<th>Accumulated % on effluent basis</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>15 g/l (NH₄)₂CO₃</td>
<td>80</td>
<td>15</td>
<td>95</td>
</tr>
<tr>
<td>2.</td>
<td>10 g/l (NH₄)CO₃</td>
<td>75</td>
<td>18</td>
<td>93</td>
</tr>
<tr>
<td>3.</td>
<td>5 g/l (NH₄)₂CO₃</td>
<td>60</td>
<td>25</td>
<td>85</td>
</tr>
<tr>
<td>4.</td>
<td>10 g/l Na₂CO₃ + 1 g/l NaHCO₃</td>
<td>56</td>
<td>27</td>
<td>83</td>
</tr>
<tr>
<td>5.</td>
<td>5 g/l Na₂CO₃ + 1 g/l NaHCO₃</td>
<td>45</td>
<td>20</td>
<td>65</td>
</tr>
<tr>
<td>6.</td>
<td>5 g/l H₂SO₄</td>
<td>54</td>
<td>30</td>
<td>84</td>
</tr>
</tbody>
</table>
LABORATORY LEACHING TESTS

Batch tests have been carried out in glass flask mounted on a shaker apparatus. Forty grams of ore sample and 200 ml of lixiviant of various types and concentrations were used for each batch. At specific time the clear solution was obtained by filtration and analysed for uranium.

The leaching test was carried out in two stages. The results are presented in Table-3.

TEST RESULTS

It has been observed from Table-3 that more than 80% uranium leaching is obtained using all the three kinds of lixiviants i.e. (NH₄)₂CO₃, Na₂CO₃ + NaHCO₃ was however not effective. The maximum uranium extraction (more than 90%) was obtained with 10-15 g/l solution of (NH₄)₂CO₃. It is also observed from Table-3 that about 80% uranium dissolution could be obtained with 5 g/l H₂SO₄ as lixiviant. But due to high calcium carbonate content of the ore the acidity of the leach solution at the end of experiment was measured as zero, which is indicative of the consumption of all the acid in lixiviant.

COLUMN PACK TESTS

Column pack leaching tests were performed in glass columns of about 8.7 cm. I.D. and 150 cm long. About 10 Kg of ore sample was packed in each column to a height of about 110 cm. Gravel filter bed was placed in the bottom of each column prior to charging it with ore sample, the level of lixiviant in the column was kept about 10 cm above the ore level at the time of feeding of lixiviant from the top of the column. Permeability of the ore column could be calculated by measuring the pressure difference in level in a small dia glass tube attached to the leach column.

The concentration of uranium in pregnant leach and permeability of ore column were measured after specific time interval. The results are tabulated in Table-4 to 6.

RESULTS OF COLUMN PACK TESTS

It is observed from these tables that 25 g/l (NH₄)₂CO₃ is an efficient leaching agent. It gives higher recovery of uranium, does not change permeability of ore bed and yields clear pregnant leach solution. The results indicate that about 96% uranium could be leached in 95 hours without much change in permeability of ore bed except the higher leachant consumption i.e. about 72 Kg/Ton of ore.

It is also seen from these tables that 15 g/l (Na₂CO₃) is also a feasible lixiviant which yields about 91% uranium in 110 hours.

The results also indicate that permeability of ore bed changes with time and the leach solution is not clear due to the precipitation of CaCO₃ at higher pH.

The results of column pack leaching tests with 10 g/l NaHCO₃ solution as lixiviant indicate that about 73% uranium leaching could be achieved in 216 hours. Although uranium leaching rate is slow as compared to 25 g/l (NH₄)₂CO₃ and 15 g/l (Na₂CO₃ + NaHCO₃) but calcium remains in solution and does not affect the permeability of ore bed due to lower pH of the system.
### Table 4: RESULTS OF COLUMN PACK TEST

<table>
<thead>
<tr>
<th>Lixiviant (NH$_4$)$_2$CO$_3$</th>
<th>25 g/l pH</th>
<th>9.2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sample Size</td>
<td>87mm dia x 1090mm Volume</td>
<td>6435 cm$^3$</td>
</tr>
<tr>
<td>Weight</td>
<td>3.89 Kg Grade</td>
<td>0615%</td>
</tr>
<tr>
<td>Ore Height</td>
<td>9.89 Kg Grade</td>
<td>0615%</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>t/hr</th>
<th>VL/VS</th>
<th>0.2</th>
<th>0.6</th>
<th>1.5</th>
<th>2.5</th>
<th>2.0</th>
<th>3.0</th>
<th>3.8</th>
<th>4.2</th>
<th>4.5</th>
</tr>
</thead>
<tbody>
<tr>
<td>C. U$_3$O$_8$ (FPM)</td>
<td>200</td>
<td>510</td>
<td>95</td>
<td>40</td>
<td>33</td>
<td>20</td>
<td>18</td>
<td>13</td>
<td>10</td>
<td></td>
</tr>
<tr>
<td>Recovery</td>
<td>4.2</td>
<td>36.5</td>
<td>55.4</td>
<td>65.9</td>
<td>72.6</td>
<td>78.9</td>
<td>86.1</td>
<td>91.8</td>
<td>96.5</td>
<td></td>
</tr>
<tr>
<td>W. g/Kg</td>
<td>3.2</td>
<td>9.7</td>
<td>24.1</td>
<td>41.9</td>
<td>45.0</td>
<td>48.0</td>
<td>60.8</td>
<td>67.2</td>
<td>72.0</td>
<td></td>
</tr>
<tr>
<td>KO, m/day</td>
<td>2.3</td>
<td>2.2</td>
<td>2.0</td>
<td>1.9</td>
<td>2.0</td>
<td>2.3</td>
<td>2.2</td>
<td>2.1</td>
<td>2.1</td>
<td></td>
</tr>
</tbody>
</table>

### Table 5: RESULTS OF COLUMN PACK TEST

<table>
<thead>
<tr>
<th>Lixiviant</th>
<th>10 g/l Na$_2$CO$_3$ + 5 g/l NaHCO$_3$ pH</th>
<th>10.2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sample Size</td>
<td>87mm dia x 1100mm Volume</td>
<td>6660 cm$^3$</td>
</tr>
<tr>
<td>Weight</td>
<td>10.1 Kg.</td>
<td></td>
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</tbody>
</table>

<table>
<thead>
<tr>
<th>t/hr</th>
<th>VL/VS</th>
<th>2</th>
<th>0.2</th>
<th>0.6</th>
<th>1.1</th>
<th>1.6</th>
<th>1.9</th>
<th>2.6</th>
<th>3.2</th>
<th>3.6</th>
<th>3.9</th>
<th>4.4</th>
</tr>
</thead>
<tbody>
<tr>
<td>C. U$_3$O$_8$</td>
<td>121</td>
<td>220</td>
<td>120</td>
<td>76</td>
<td>30</td>
<td>25</td>
<td>22</td>
<td>18</td>
<td>16</td>
<td>12</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Recovery</td>
<td>2.6</td>
<td>17.1</td>
<td>31.6</td>
<td>45.0</td>
<td>50.9</td>
<td>58.6</td>
<td>65.7</td>
<td>72.8</td>
<td>78.7</td>
<td>85.1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>W. g/Kg</td>
<td>2.0</td>
<td>6</td>
<td>11</td>
<td>16</td>
<td>19</td>
<td>26</td>
<td>32</td>
<td>36</td>
<td>39</td>
<td>44</td>
<td></td>
<td></td>
</tr>
<tr>
<td>KO, m/day</td>
<td>1.3</td>
<td>1.21</td>
<td>1.1</td>
<td>1.05</td>
<td>0.9</td>
<td>0.85</td>
<td>0.8</td>
<td>0.81</td>
<td>0.8</td>
<td>0.9</td>
<td></td>
<td></td>
</tr>
<tr>
<td>C. Ca$^{++}$</td>
<td>1</td>
<td>1</td>
<td>2</td>
<td>1</td>
<td>1</td>
<td>1</td>
<td>1</td>
<td>1</td>
<td>1</td>
<td>1</td>
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</tr>
</tbody>
</table>

### Table 6: RESULTS OF COLUMN PACK TEST

<table>
<thead>
<tr>
<th>Ore weight</th>
<th>10 Kg.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore grade</td>
<td>0.0615%</td>
</tr>
<tr>
<td>Uranium</td>
<td>6150 mg Vol. of the Ore</td>
</tr>
<tr>
<td>Lixiviant concentration</td>
<td>(Sodium bi-carbonate)</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>t/hr</th>
<th>24</th>
<th>48</th>
<th>72</th>
<th>96</th>
<th>120</th>
<th>144</th>
<th>168</th>
<th>192</th>
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</thead>
<tbody>
<tr>
<td>VL/VS</td>
<td>1.23</td>
<td>2.50</td>
<td>3.10</td>
<td>3.67</td>
<td>4.12</td>
<td>4.53</td>
<td>4.91</td>
<td>5.29</td>
</tr>
<tr>
<td>C. U$_3$O$_8$ (FPM)</td>
<td>302</td>
<td>103</td>
<td>126</td>
<td>99</td>
<td>71</td>
<td>56</td>
<td>28</td>
<td>14</td>
</tr>
<tr>
<td>C. Calcium (FPM)</td>
<td>10</td>
<td>4</td>
<td>8</td>
<td>10</td>
<td>7</td>
<td>18'</td>
<td>21'</td>
<td>4</td>
</tr>
<tr>
<td>Recovery</td>
<td>39.06</td>
<td>52.84</td>
<td>60.8</td>
<td>66.13</td>
<td>69.49</td>
<td>71.90</td>
<td>73.02</td>
<td>73.57</td>
</tr>
<tr>
<td>W. g/Kg</td>
<td>2.66</td>
<td>3.77</td>
<td>4.42</td>
<td>4.76</td>
<td>5.21</td>
<td>5.52</td>
<td>6.57</td>
<td>7.60</td>
</tr>
<tr>
<td>KO, m/day</td>
<td>2.4</td>
<td>2.11</td>
<td>1.89</td>
<td>2.1</td>
<td>2.4</td>
<td>2.1</td>
<td>2.2</td>
<td>2.3</td>
</tr>
</tbody>
</table>
RESTORATION OF WATER QUALITY IN COLUMN PACK ORE BED

Clear water irrigation/sweeping was tested for the restoration of water quality in column pack ore bed. The results indicate that the concentration of uranium in effluent stream dropped to less than 1 ppm with 4 ore bed volumes of water sweeping and pH of the system dropped to less than 8.5 with 8 ore bed volumes of water irrigations when 25 g/l $\text{(NH}_4\text{)}_2\text{CO}_3$ was used as lixiviant. In $\text{Na}_2\text{CO}_3 + \text{NaHCO}_3$ system, sweeping of 5 ore bed volumes dropped uranium concentration in the effluent to less than 1 ppm and 10 bed volumes of clear water sweeping were found sufficient to drop pH to less than 8.5.

IN-SITU LEACH FIELD TEST

The test results of laboratory leaching and column pack tests indicated that $\text{(NH}_4\text{)}_2\text{CO}_3$ is the best lixiviant as the selectivity of leaching uranium with $\text{(NH}_4\text{)}_2\text{CO}_3$ was better than the other alkaline and acid leachants. Moreover, permeability of ore bed was not affected by $\text{(NH}_4\text{)}_2\text{CO}_3$ and it produced clear leach liquor as compared to other lixiviants.

As $\text{(NH}_4\text{)}_2\text{CO}_3$ is not locally available, therefore a field test was planned to be carried out using conventional five spot pattern in order to see the effectiveness of alternative leachants namely 10 g/l $\text{Na}_2\text{CO}_3 + 5$ g/l $\text{NaHCO}_3$ mixture and 10 g/l $\text{NaHCO}_3$.

FIELD TEST - I

A test site was selected on one corner of the ore body, and drilling of a five spot pattern was completed. The main features of the site and well field pattern are shown in Table-7. A monitoring well is drilled outside of the leaching area to observe if any excursion of the leach solution is taking place or not. The location of the pattern is shown in Fig-1.

Four injection wells, one production well and one monitoring well were developed and completed with 6 inch diameter PVC casing. A sectional view of a well is shown in Fig-2. Conventional airlifting mechanism was used for hole development. Pumping test was then conducted using 4 inch diameter stainless steel submersible pump of capacity 4.0$\text{M}^3/\text{hr}$ to find out permeability of ore zone in all the directions, hole integrity, capacity and draw down values. Average preleach water analysis sampled out during pumping test is given Table-8.

Leaching of uranium in the pattern was started by the injection of lixiviant consisting of 10 g/l $\text{Na}_2\text{CO}_3 + 5$ g/l $\text{NaHCO}_3$ added to ground water in all the four injection wells at a rate of about 0.8 - 1 $\text{M}^3/\text{hr}$ in each injection well in order to balance production stream i.e. about 4 $\text{M}^3/\text{hr}$. A slightly over production was used to establish solution control by generating hydrologic cone of depression around the leach area.

The pH in the production solution rose from 8 to 9.5 after 10 days of startup. Uranium concentration (10 ppm) at this stage began to appear in the production well. The production stream was then coupled to the surface plant which consisted of settling tanks, gravel bed filters, ion exchange columns, precipitation & process solution hold up tanks, yellow cake thickener, pilot scale rotary drum vacuum filter and a yellow cake drying oven. A flow sheet depicting uranium recovery process is shown in Fig-3.
FIG. 1. Location map of test site

LEGEND
INJECTION WELL. ⭕ PRODUCTION WELL △ MONITORING WELL ○

SCALE: 1 cm = 10 m

FIG. 2. Cross-section of well.
FIG. 3. Flowsheet for the recovery of uranium from the in-situ leach process Qubul Khel.
**Table-7: MAIN FEATURES OF TEST SITE AND WELL PATTERN-1**

Location - Between Fence Lines
No. + 16 + 17

<table>
<thead>
<tr>
<th>Field</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Well field</td>
<td>Five Spot</td>
</tr>
<tr>
<td>No. of injection wells</td>
<td>4</td>
</tr>
<tr>
<td>No. of production wells</td>
<td>1</td>
</tr>
<tr>
<td>Width of well spacing</td>
<td>6 inch</td>
</tr>
<tr>
<td>No. of Monitoring wells</td>
<td>1</td>
</tr>
<tr>
<td>Distance between injection wells</td>
<td>19 m</td>
</tr>
<tr>
<td>Dip of strata</td>
<td>26°</td>
</tr>
<tr>
<td>Depth of ore zone from surface</td>
<td>75 m</td>
</tr>
<tr>
<td>Average thickness of ore body</td>
<td>4.88 m</td>
</tr>
<tr>
<td>Porosity of ore zone</td>
<td>20%</td>
</tr>
<tr>
<td>Permeability of ore zone</td>
<td>2.5 m/day</td>
</tr>
<tr>
<td>Average grade</td>
<td>0.08% U₂O₅</td>
</tr>
</tbody>
</table>

**Table-8: AVERAGE PRELEACH WATER ANALYSIS RESULTS OF WELL PATTERN-1**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>pH</td>
<td>7.9</td>
</tr>
<tr>
<td>TDS</td>
<td>1097 ppm</td>
</tr>
<tr>
<td>Aluminium</td>
<td>1 ppm</td>
</tr>
<tr>
<td>Bicarbonate</td>
<td>210 ppm</td>
</tr>
<tr>
<td>Carbonate</td>
<td>N.D</td>
</tr>
<tr>
<td>Calcium</td>
<td>30 ppm</td>
</tr>
<tr>
<td>Chloride</td>
<td>380 ppm</td>
</tr>
<tr>
<td>Iron</td>
<td>N.D</td>
</tr>
<tr>
<td>Magnesium</td>
<td>15 ppm</td>
</tr>
<tr>
<td>Potassium</td>
<td>8 ppm</td>
</tr>
<tr>
<td>Silicon</td>
<td>20 ppm</td>
</tr>
<tr>
<td>Sodium</td>
<td>500 ppm</td>
</tr>
<tr>
<td>Sulphate</td>
<td>150 ppm</td>
</tr>
<tr>
<td>U₂O₅</td>
<td>1 ppm</td>
</tr>
</tbody>
</table>

All values in ppm unless otherwise specified.

**Table-9: TYPICAL COMPOSITION OF INJECTED LIXIVIANT IN WELL PATTERN-1**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Range</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bicarbonate</td>
<td>1400 - 3600 ppm</td>
</tr>
<tr>
<td>Carbonate</td>
<td>2000 - 5600 ppm</td>
</tr>
<tr>
<td>Chloride</td>
<td>150 - 300 ppm</td>
</tr>
<tr>
<td>Sodium</td>
<td>2000 - 5000 ppm</td>
</tr>
<tr>
<td>pH</td>
<td>5.9 - 10.2</td>
</tr>
<tr>
<td>H₂O₂</td>
<td>200 - 1000 ppm</td>
</tr>
</tbody>
</table>

All values in parts per million unless otherwise specified.
The addition of hydrogen peroxide @ of 0.5 g/l in the lixiviant was started on the 20th day of the start-up of leaching phase. The production stream pH increased from 9.5 to 10.1 due to increase in carbonate ions concentration. In the mean time flow rate of submersible pump dropped from 4M³/hr to 2M³/hr and ultimately clogged due to precipitation of CaCO₃ present in ground water and in the strata. The operation was then stopped, submersible pump and production wells were cleaned with dilute nitric acid and hydrochloric acid respectively. Scaling due to precipitation of calcium carbonate again restricted pump flow rate after two days operation. In view of the above problem the mode of pumping was replaced by airlifting at a flow rate of about 2M³/hr. The production stream was carefully balanced with injection flow rates to all the injection wells i.e. slightly less injection than production to prevent leakage of solution outside the production zone.

The well pattern remained in operation for about 20 months during which various constituents of lixiviant such as sodium carbonate, sodium bicarbonate and hydrogen peroxide were varied to obtain optimum value of leachant concentration. The approximate concentration of various chemicals in the lixiviant is shown in Table-9.

Simultaneously concentration of uranium & other contaminants in pregnant leach liquor and depleted process stream of ion exchange column was recorded to see buildup of chemical contaminants, before it was refortified with leaching agents and reinjected into the formation for new leach cycle. The pH variation, uranium, carbonate, bicarbonate concentrations and buildup of other contaminants in the pregnant leach over time are shown in Fig-4 to 10.

TEST RESULTS

i. pH
Fig 4 shows the pH record during leaching. The pH of leach solution increased from 8 to 9.5 after 10 days of injection and then leveled off between 9.9 to 10.1 throughout the operation time.

ii. Uranium
It has been observed from Fig-5 that U₃O₈ concentration in the leach liquor reached up to 65 ppm when H₂O₂ was injected in the strata initially at the rate of 0.5 g/l. Again the concentration of U₃O₈ decreased to about 30 ppm at 0.2 g/l injection rate of the oxidant. The peak enhancement of 160 ppm U₃O₈ in the leach liquor was observed at enhanced rate of H₂O₂ i.e. 1 g/l. The high U₃O₈ concentration then leveled off in the range of 150-160 ppm for about one month and then dropped to about 30 ppm until reverse cycle began, after about 570th day. At this stage about 65% of U₃O₈ has been recovered from the pattern.

iii. Carbonate and bicarbonate
Figure-6 and 7 show the high consumption of sodium carbonate and sodium bicarbonate during initial leaching stage. Afterwards the concentration of carbonate reached at about 1.5 g/l in the leach liquor when about 3.4 g/l carbonate ions were maintained in the lixiviant. On the other hand the high concentration of bicarbonate ion in the leach liquor (1.5 - 1.9 g/l) indicate it’s regeneration from the formation/ore zone. The carbonate concentration leveled off at about 0.8 g/l at 520th day at the same addition rates in the lixiviant. The high consumption of Na₂CO₃ may be attributed to the presence of about 5% calcium in the ore zone which during leaching is replaced with sodium by ion exchange on clay particles and forms CaCO₃ with the available carbonate ions of the lixiviant.
FIG. 4. pH concentration versus time in production well pattern I.

FIG. 5. U₃O₈ concentration versus time in production well pattern I.

FIG. 6. Carbonate concentration versus time in production well pattern I.
FIG. 7. Bicarbonate concentration versus time in production well pattern I.

FIG. 8. Calcium concentration versus time in production well pattern I.

FIG. 9. Chloride concentration versus time in production well pattern I.
iv. Calcium
Fig-8 shows calcium record during leaching. It has been indicated that due to high pH of the system i.e. 9.5 to 10.1 the calcium present in ground water (about 30 ppm) and in the ore zone precipitated as CaCO$_3$. This resulted in scaling in the strata as well as in the production well.

v. Chloride
Fig-9 shows that the level of chloride in leach solution began to decrease from 380 ppm to about 280 ppm after about 45 days and then leveled off to about 180 ppm for the rest of the days of leaching. The decrease in chloride content in the production stream may be due to the depletion of chloride ions in production stream and adequate washing of resin in elution cycle.

vi. Silicon
Like the chloride level, silicon level (Fig-10) decreased from 20 to 9 ppm and then leveled off. The reason being that due to high pH of the system silicon levels present in the ground water and ore zone precipitated as calcium silicate in the production well. The analysis of scale sample obtained during periodic cleaning of airlifting pipes is given in Table-10. It has been observed (Table-10) that most of the contaminants of leach solution such as magnesium, sulphate, silicon, calcium etc. have been precipitated in the production well and deposited in the form of scale due to the high pH of the system. This scale formation not only restricted the most economical mode of liquid pumping through submersible pump but also frequent cleaning of the production well, airlifting pipe lines and ion exchange circuitry necessitated to test other locally manufactured leachant, sodium bicarbonate, to maintain high solubility of scale forming cations and anions present in ground water and ore zone.
FIELD TEST - 2

In early 1992, another five spot pattern was developed at a distance of about 20 meters from the previous pattern as shown in Fig-1. The main features of the pattern comprising four injection, one production and one monitoring well are shown in Table-11. After drilling and development of well pattern, hydrological parameters of the ore zones and strata were obtained through pumping test using 4 in. diameter stainless steel submersible pump. Average preleach water analysis obtained during pumping test is given in Table-12.

Leaching commenced with the injection of 10 g/l NaHCO$_3$ solution into the well field to all the four injection wells at a rate of about 0.5 M$^3$/hr in each well. Emphasis was put on flow balancing. At no time injection rate was allowed to exceed the recovery i.e. about 2.2 M$^3$/hr.

Initially, leach liquor was produced through airlifting from the production well to observe whether the leachant composition/concentration maintains a high solubility of calcium in the system which could foul the production well and surface lines.

As a result of differential chemical reaction rates of the host rock, the concentration ranges of major constituents of the lixiviant maintained throughout the leaching process are given in Table-13. Only small amount of uranium was leached during the 1st month operation. The uranium mobilization started with the injection of oxygen in the strata. Significant concentration of uranium began to appear in the production well about 65 days after leaching started. The high average calcium concentration in production stream and no appreciable decrease in flow rate, during this period, indicated that there has been no scaling taking place due to precipitation of CaCO$_3$ in the production well. This has been physically confirmed by the inspection of airlift pipes during periodical maintenance. The mode of pumping was then replaced with a stainless steel submersible pump of 4 inch diameter. Lixiviant injection rates to all the four injection wells were increased from 0.5 M$^3$/hr to about 0.9 M$^3$/hr to balance the production capacity of the pump i.e. about 3.80 M$^3$/hr. During the last 4 months operation no appreciable decrease in the injection/production flow rate in all the five wells was observed. The pH, concentration of U$_3$O$_8$ and other constituents of production stream were recorded/analysed and the variations with leaching time are depicted in Figures 11-17.

i. pH
It has been observed from Fig-11 that pH in the production well slightly increased from the base line value 8.8 to about 8.9 in few days after the leaching started. The pH then leveled at about 8.9 for the next 120 days. The pH in the production well then dropped suddenly to about 7.8. This may be attributed to the higher consumption rate of carbonate ions by the host sandstone.

ii. Uranium
Uranium concentration in the production well increased in steps (Fig-12), that corresponded to the oxygen addition and lixiviant concentration. The uranium concentration remained stable at 3-4 ppm for about 25 days and then increased to about 8 ppm and leveled off at about 8 ppm for about 30 days. The peak enhancement in uranium concentration of production well to about 70 ppm was recorded 130 days after leaching started. Suddenly drop in uranium concentration from 70 to 40 ppm may be attributed to low bicarbonate concentration in the injected lixiviant, maintained at this stage. A significant increase in the uranium concentration was again observed with the increase in NaHCO$_3$ concentration and addition of H$_2$O$_2$ instead of oxygen in the lixiviant. The maximum concentration of 90 ppm U$_3$O$_8$ was recorded till the reporting day and was still on the increase.
### Table-10  ANALYSIS RESULTS OF SCALE SAMPLES

<table>
<thead>
<tr>
<th>Description</th>
<th>Analysis Result (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Calcium</td>
<td>35.81</td>
</tr>
<tr>
<td>Carbonate</td>
<td>53.47</td>
</tr>
<tr>
<td>Bicarbonate</td>
<td>0.56</td>
</tr>
<tr>
<td>Chloride</td>
<td>1.27</td>
</tr>
<tr>
<td>Moisture</td>
<td>7.5</td>
</tr>
<tr>
<td>Silicate</td>
<td>0.78</td>
</tr>
<tr>
<td>Sulfate</td>
<td>0.3</td>
</tr>
<tr>
<td>Magnesium</td>
<td>0.2</td>
</tr>
<tr>
<td>Aluminum</td>
<td>36.6 Fr</td>
</tr>
<tr>
<td>Sulfate</td>
<td>0.5</td>
</tr>
</tbody>
</table>

### Table-11  MAIN FEATURES OF WELL PATTERN-II

<table>
<thead>
<tr>
<th>Location</th>
<th>Between Fences Line No. Line No. 15 and 16</th>
</tr>
</thead>
<tbody>
<tr>
<td>Well Pattern</td>
<td>Five Spot</td>
</tr>
<tr>
<td>No of Injection wells</td>
<td>4</td>
</tr>
<tr>
<td>No of Production well</td>
<td>1</td>
</tr>
<tr>
<td>No of Monitoring well</td>
<td>1</td>
</tr>
<tr>
<td>Dia of well casing</td>
<td>6 inch (PVC)</td>
</tr>
<tr>
<td>Lip of strata</td>
<td>26°</td>
</tr>
<tr>
<td>Distance between injection well</td>
<td>10 0m</td>
</tr>
<tr>
<td>Average thickness of ore body</td>
<td>5 4m</td>
</tr>
<tr>
<td>Permeability of ore zone</td>
<td>28%</td>
</tr>
<tr>
<td>Permeability of ore zone</td>
<td>2 m/day</td>
</tr>
<tr>
<td>Average grade</td>
<td>0.057% 0.3%</td>
</tr>
<tr>
<td>Depth of the ore zone from surface</td>
<td>75 m</td>
</tr>
</tbody>
</table>
Table-12: PRELEACH WATER ANALYSIS OF WELL PATTERN-II

<table>
<thead>
<tr>
<th>Constituent</th>
<th>Concentration Range (PPm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>pH</td>
<td>8.3</td>
</tr>
<tr>
<td>TDS</td>
<td>860</td>
</tr>
<tr>
<td>Bicarbonate</td>
<td>190</td>
</tr>
<tr>
<td>Calcium</td>
<td>30</td>
</tr>
<tr>
<td>Carbonate</td>
<td>30</td>
</tr>
<tr>
<td>Magnesium</td>
<td>15</td>
</tr>
<tr>
<td>Potassium</td>
<td>9</td>
</tr>
<tr>
<td>Silicon</td>
<td>8</td>
</tr>
<tr>
<td>Sodium</td>
<td>200</td>
</tr>
<tr>
<td>Sulphate</td>
<td>100</td>
</tr>
<tr>
<td>$U_3O_8$</td>
<td>3</td>
</tr>
</tbody>
</table>

All values in parts per million unless otherwise specified.

Table-13: TYPICAL LIXIVIANT CONCENTRATION FOR INJECTION WELL PATTERN-II

<table>
<thead>
<tr>
<th>Constituent</th>
<th>Concentration Range (PPm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bicarbonate</td>
<td>3000 - 7000</td>
</tr>
<tr>
<td>Carbonate</td>
<td>0 - 250</td>
</tr>
<tr>
<td>Sodium</td>
<td>1000 - 3000</td>
</tr>
<tr>
<td>Calcium</td>
<td>2 - 10</td>
</tr>
<tr>
<td>Silicon</td>
<td>2 - 8</td>
</tr>
<tr>
<td>Magnesium</td>
<td>≥</td>
</tr>
<tr>
<td>Chloride</td>
<td>1000 - 2500</td>
</tr>
<tr>
<td>$U_3O_8$</td>
<td></td>
</tr>
<tr>
<td>pH</td>
<td>8.00 - 8.5</td>
</tr>
<tr>
<td>$O_2$</td>
<td>1000 - 2500</td>
</tr>
</tbody>
</table>
FIG. 11. pH concentration versus time in production well pattern II.

FIG. 12. U₃O₈ concentration versus time in production well pattern II.

FIG. 13. Carbonate concentration versus time in production well pattern II.
iii. **Carbonate and bicarbonate**

Fig. 13 and 14 show the variation in the carbonate and bicarbonate concentrations of production well with time. Carbonate/bicarbonate were generally controlled by the addition of NaHCO$_3$ in the lixiviant as a result of differential chemical reaction rates of the host rock. However, it has been observed from Fig. 13 and 14 that uranium leaching rate increases with the increase in carbonate/bicarbonate ions in the production well.

iv. **Calcium**

Fig. 15 shows high frequency of calcium depletion in production well with time. The calcium concentration remained relatively constant around 30-35 ppm during the course of leaching. The reason being that pH around 9.0 of the system maintains high solubility of calcium and magnesium which can foul the production well and surface plant circuitry.
v. **Chloride**
   It has been observed from Fig-16 that chloride level began to decrease from base line value of 280 to 150 ppm after about 30 days in production well. The chloride concentration then leveled off at about 200 ppm for about 120 days.

vi. **Silicon**
   The silicon level of the production well as indicated in Fig-17 remained constant at about-8 ppm for about 40 days. Afterwards the silicon level rose to about 20 ppm which indicates the solubilization of silicon as sodium silicate in comparatively lower pH of the system.
CONCLUSION

Laboratory and column pack tests indicated that 25 g/l (NH$_4$)$_2$ CO$_3$, 15 g/l (Na$_2$CO$_3$ + NaHCO$_3$) and 10 g/l NaHCO$_3$ could be used as feasible lixiviants for the recovery of uranium from Qubul Khel ore body using In-situ Leach Mining Technique. Column pack tests also indicated that uranium, carbonate, bicarbonate, chloride, silicon, sodium and other dissolved contents of the production solution could be brought to preleach baseline values by required ore zone volumes of water sweeping during restoration stage.

Comparative leaching behaviour of Na$_2$CO$_3$ + NaHCO$_3$ and only NaHCO$_3$ as leachants, in field, indicated that 6 g/l NaHCO$_3$ solution could effectively be used as lixiviant. The system not only maintained good permeability of ore zone but also eliminated the precipitation of CaCO$_3$ in production well and surface circuitry due to the lower pH of the system.

Moreover, leach production capacity could also be enhanced and brought to the desired level by using submersible pump in the production well. This could also lower the pumping cost of the each solution by a factor of about 4 as compared to the air lifting technique.

The study also revealed that overall cost of the sodium bicarbonate would be similar to the mixture of sodium carbonate and bicarbonate.

It has therefore been concluded that sodium bicarbonate solution could be substituted as lixiviant in place of mixture of sodium carbonate and bicarbonate for the recovery of uranium from Qubul Khel ore body by in-situ leach technique.

BIBLIOGRAPHY

ENVIRONMENTAL ASPECTS OF THE OPERATION
AND SANATION OF THE ISL URANIUM MINING AT
STRÁŽ POD RALSKEM, CZECHOSLOVAKIA

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DIAMO, o.z. Ekologie,
Stráž, Czechoslovakia

Abstract

This contribution presents a review of the ISL technological activities at the Stráž-Hamr sandstone-hosted (but not "sandstone type") sedimentary uranium deposit in Northern Bohemia, Czechoslovakia. The deposit occurs in the Bohemia Cretaceous basin, in Cenomanian sandstones. In this area mainly Turonian, poorly cemented, quartzose sandstones crop out on the surface. The area, therefore, was times overgrown by pine woods and only a minor part had been used for farming. A brief description of the geologic, climatic and pedologic features and the main types of the environmental impacts of the sulphuric acid ISL technology in the region is given. The most important impacts are (1) the contamination of the Turonian aquifer used elsewhere as a source of potable water, (2) the initial total deforestation of the whole affected area, resulting (3) in a serious destruction of the soil cover and mainly in the higher-lying parts of the area, (4) in an exposition of the whole surface to weathering and erosion, or, in the lower parts, (5) probably in a consecutive wetting of the soil in the leaching fields. In a part of the fields the net of technological wells had been designed in a manner to allow for the needs of conservation of the forest in the spaces between the technological installations. This concept did not offer greater advantages because of the weakening of the stability of the forest and the resulting increased risk of destroying the technological installations by falling trees. A short description of the measures for the prevention and the minimization of the superficial erosion is given (seeding of grass and trees, textile nets and mats, location of wells and installations in accord with the surface morphology etc.). Finally, a short review of the efforts made for the preparation of the sanation is also given.

1. INTRODUCTION

The following contribution is the third part of a series of papers presented in this meeting, which are aimed to give the present, international auditory a non-biased picture of the ISL extraction of uranium ores by sulphuric acid at the deposit of Stráž-Hamr in Northern Bohemia. The intent of this third part is to call your attention to some aspects of the environmental impacts of the ISL technology used in this deposit. The most important risk connected with the ISL technology is the contamination of underground water, not only in the Cenomanian ore-bearing horizont, but also in the aquifer in the overlying Turonian rocks. The Turonian aquifer is one of the most important sources of potable water in Bohemia. There are other environmental impacts which affected the surface of the landscape and the vegetative cover of the area of the leaching fields.

To get a criterion allowing to imagine the importance of the described features it may be first necessary to give a brief description of the affected region from the points of view of geography, climatology, geology, pedology.

1.1 Geography

Geographically, the deposit is situated near the northern borders of the so-called Czech Cretaceous Basin (Fig. 1). The height above sea-level of the surface in this region is between 292 m in the south-west in the alluvial plain of the Ploučnice river, to approximately 460 m in the slopes of the Ralsko hill. The cause of this great height difference on a distance of only about 4 km lies in the climbing of some of the southern leaching fields up the slopes of the Ralsko hill where they attain gradients as high as 30° to 40°. The summit of this hill at 696 m above the sea-level is the highest point in the immediate surroundings of the deposit.
Fig. 1 - Sketch of the geographic and geologic position of the Stráž-Hamr deposit
Table 1. Important annual climatic conditions of Stráž [1, 2, 3]

<table>
<thead>
<tr>
<th>Condition</th>
<th>Value</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean temperature</td>
<td>7.7</td>
<td>°C</td>
</tr>
<tr>
<td>Mean precipitation</td>
<td>697</td>
<td>mm</td>
</tr>
<tr>
<td>Mean relative humidity</td>
<td>78</td>
<td>%</td>
</tr>
<tr>
<td>Days with precipitation &gt; 0.1</td>
<td>147</td>
<td>days</td>
</tr>
<tr>
<td>Precipitation during vegetative season approx.</td>
<td>400</td>
<td>mm</td>
</tr>
<tr>
<td>Mean temperature during the vegetative season</td>
<td>13.3</td>
<td>°C</td>
</tr>
<tr>
<td>Days with snow cover</td>
<td>50</td>
<td>days</td>
</tr>
<tr>
<td>Duration of the vegetative season</td>
<td>159</td>
<td>days</td>
</tr>
<tr>
<td>Mean hours of sunshine</td>
<td>1700</td>
<td>hours</td>
</tr>
</tbody>
</table>

Fig. 2 - Tectonic sketch of the Stráž block

1.2 Climatology

The climatic conditions in the region around Stráž can be characterized by some of the most important parameters shown in Table 1.

1.3 Geology

The geological character of the deposit [4] is determined by its position in the northeastern part of the Bohemian Cretaceous Basin which occupies almost the whole north-eastern part of
Bohemia. The deposit is localized in the basin near its border in the so called Stráž block, a tectonic unit limited by four important faults (Fig. 2). The rocks outcropping in this block on the surface are poorly cemented, friable, highly quartzose Middle Turonian sandstones with very low contents of nutrient elements. The underlying rocks are lowly permeable siltstones and marly sandstones. This sequence contains some intercalations of limestones in the lowermost part of the Lower Turonian, which form the base of the very important Middle Turonian aquifer. Its groundwater content is mostly recharged directly from the surface precipitations. The ore concentrations are located in the underlying sedimentary rocks of Cenomanian age. The whole area is disturbed by a great number of faults of minor importance. In some cases, the faults are filled by dykes or contain diatremes of volcanic rocks or basaltic of more basic, but some times also of trachytic composition.

1.4 Pedology

The pedologic character of the most important soil types occurring in the region is defined almost totally by the bedrocks, the Cretaceous sandstones, which are extremely poor in nutrients elements (Table 2).

The prevailing soils, therefore, are podsols or similar types. Most frequently there are poorly developed podsol soils with a very thin A-horizon containing a relatively primitive, acid raw form of humus. These soils are in accordance with the prevailing type of forests growing on sandstone areas [5, 6]. There grow mainly pines, which only in some more humid areas are accompanied by other species as spruces, larchs, and in distinctly wet places in the flood plain also by ashes, alders, willows and birches. Often severe limitations on the growth of the trees are imposed by iron oxide layers, the so-called Ortsteins.

Neither the pine woods nor these soils are the primary soil type in this country. They developed since the 17 and 18 century, during which the primary naturally-grown mixed forests were totally destroyed by the excessively production of charcoal needed to maintain a primitive iron-smelting industry and glassworks. There are only very restricted areas in which other soil types, e.g. brown and grey forest soils, are developed. In these places spruce forests of greater economic value had been growing before the development of uranium mining. The lower parts of the leaching fields in the alluvial plane of the Ploučnice river contain wetland soil types. In the areas formerly occupied by fields remainders of the original soil, an anthropogene deteriorated agrisols type is preserved.

<table>
<thead>
<tr>
<th>Table 2. The content of the main plant nutrient elements in the soils of the leaching fields [1]</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sample</td>
</tr>
<tr>
<td>--------</td>
</tr>
<tr>
<td>F87/Z1</td>
</tr>
<tr>
<td>F87/N1</td>
</tr>
<tr>
<td>F87/N2</td>
</tr>
<tr>
<td>F87/N3</td>
</tr>
<tr>
<td>F88/Z21</td>
</tr>
<tr>
<td>F88/Z22</td>
</tr>
<tr>
<td>HA88/Z</td>
</tr>
<tr>
<td>H888/N</td>
</tr>
<tr>
<td>AH889/Z</td>
</tr>
</tbody>
</table>
2. THE ENVIRONMENT IMPACTS OF THE ISL ON THE SURFACE

The most important environmental impact of the ISL extraction technology on the overlying surface are as follows [7]:

- the clearing of the woods of the whole area of the leaching fields;
- the exposure of the surface to the influence of the erosion and weathering processes;
- the risk of escaping acid leaching and radioactive solutions from the wells and pipelines;
- the wetting of some of the lower parts of the leaching fields.

2.1 Deforestation and its results

The prevailing part of the surface of the leaching fields earlier were overgrown by pine woods, only some limited areas have been used for agricultural purposes (Fig. 3). The ISL technology of extraction of ores underground is based on the use of a very high number of wells. As in the woods the trees interfere with the well drilling and later constitute a risk of falling on the
technological pipelines and destroying them, in most parts of the fields the surface had been totally deforested. This deforestation, moreover, had been made hastily, what is documented by an appreciable number of trunks and almost all of the smaller trees, branches and twigs remaining still in the area. These remnants, together with an appreciable number of remaining dead, still standing trees, still are found frequently in the oldest leaching fields, which occupy mostly the lower, wetland part of the area. Here, inmidst of the birches, willows and ashes of the naturally seeded younger forest, which its mostly rather of a bushy character, a ghostly scenery of dead trees and rotten trunks inmidst the young forest had originated.

2.2 Exposing the surface to erosion

At the same time very often the prevailing part of the soil cover was removed or eventually scraped together into regular, discontinuous mounds. The deforesting and removal of the soil cover in the higher parts of the area with greater gradients finally exposed the whole underlying rocks to the disintegrative erosional processes. The extent of this area is shown in Fig. 4. Because the underlying rocks are only poorly cohesive sandstones, these measures initiated an appreciable wash-down, predominantly of the finer grained constituents of the rocks. This led to sheet erosion
and, sometimes, especially in the places of the concentration of smaller water-flows, also to the
generation of rather deep erosional furrows. These processes were accelerated by the movement
of heavy building machines during the construction of the road system, which serves to the
maintenance and monitoring of the wells.

In the lower parts of the leaching fields area, which reaches into the alluvial plain of the
Ploučnice river, the mounds and dams, constructed to be used as bases for drilling and
maintenance of wells and technological pipelines were based mainly on fascines. These
constructions disturbed and interrupted old drainage ditches which long ago had been built for
facilitating the flood waters to return into the river and to drain the wetland. Frequently it is
argued that these infringements, together with the deforesting may be the cause of a raising
groundwater level which, consecutively, may create areas of lagoons and wetlands. But, as the
exact extent of original wetlands is rather not known, it is impossible to estimate unambiguously
the difference between the original, already rather high groundwater level and the level which
resulted from these measures.

In some parts of the leaching fields an attempt was made to avoid the deforesting and to
retain the parts of the grown-up forests in the areas between the technological installations. Soon,
however, it was found that the violation of the formerly continuous forest caused its weakening
which, therefore, led to an increased exposure of the technological installations and wells to falling
trees.

In the southwestern part of the deposit the technologic wells are climbing higher up the
slopes of the Ralsko hill, where the surface gradient often is greater than 20° or even 30°. Here the
wells had been situated on platforms which, together with the connecting maintenance roads, were
built partly in cuttings, partly on embankments. This earthwork, together with the deforesting, the
scraping away of the soil cover, and the arrangement of the roads and lines connecting the wells
which sometimes had been located parallelly with the slope or along natural ravines, exposed great
areas of the hill slopes to the destroying forces of flowing water (flushing suspensions from drilling,
natural surface run-off) which often flow together to form brooklets along the roads or pipelines,
enhancing the erosion of the denuded rock surface as well as the washing-out of rills or gulls. The
immediate result of these technical measures often were the last remainders of soil together with
great amounts of sand were swept away. Thus the natural germination of rooting of seeds were
made difficult or, finally, totally impossible.

During the last some years (mostly since 1986), considerable efforts had been made [1, 8]
in the leaching fields to try to grow experimentally in limited, selected areas different species of
trees (pines, spruces, larchs, but also ashes, alders, willows, etc.) with or without grass seeding.
The application of other different anti-erosional technical and biological measures (hydromulching,
grass-seed mats, reinforcement textiles or nets etc.) had been tried with varying success.
Unfortunately, all these efforts as well as attempts to localize the routes and drilling places with
respect to the requirements of minimizing the exposure of the surface to erosion always had only
very short duration. The causes of these rather unexpected matters of fact were mostly not the
failure of these experiments, but e.g. the hostile attitude or animosity of some of the responsible
professional workers or the resignation, discharge or leave of interested enthusiasts.

2.3 Solutions escaping during failures of wells and pipelines

Inherent to the great number of wells (about 7 000) there is a corresponding risk of failure,
which may influence the environment. On the one side there is the failure of the well heads and
pipelines on the surface, which may result in a contamination of the surrounding soil by escaping of acid solutions or radioactive leachates. Such cases of failure had been detected in some cases and did affect approximately 4 per cent of the wells with the older, more primitive construction (Fig. 5). The failures occurred mainly during the re-drilling of stopped wells. The new construction with double casings and the cleaning of stopped wells by means of hydrofluoric acid instead of re-drilling eliminated this kind of risk almost totally. In some wells the radioactivity remaining in the surrounding soil had been measured and by this way it seems to be proved that the higher radioactivity in the surrounding soil did not reach farther than approximately 4.5 meters from the collar (Fig. 5).

To collect the precipitation water flowing along the surface out from the leaching fields a continuous system of drainage ditches around the whole leaching area is being operated. This system contains basins to collect the escaping waters, from which they can be transferred into the decontaminating stations. The system is monitored by automatic measurement of pH, so that any escaping acid solutions cause the closure of the system and immediately are signalled in the central dispatcher. The control of these ditches showed that till now no acidified waters escaped into the ditches. It may, therefore, be concluded that at no one of the registered failures of a well or a pipeline the escaping acid or radioactive solution did reach the periphery of the leaching area.
3. THE GROUNDWATER CONTAMINATION

The contamination of the Cenomanian aquifer by the leaching liquors had been discussed in the previous papers. But, as was shown in the introduction, in the ISL area there is another aquifer in the overlying Turonian rocks. Regardless, whether they are destined for injection or extraction, all the thousands of wells must penetrate the Turonian aquifer and its insulating basal layers of the Lower Turonian rocks. This represents another and, apparently, the most hazardous feature of the ISL technology, because the Turonian aquifer is the source of potable water in a great part of northern Bohemia.

Two types of wells are now under operation in the leaching fields. The older have single cemented casings, the newer are equipped with double cemented casings. The failure of some of these wells, mainly of the ones with the older, more simple casing system, caused a contamination of the Turonian aquifer in the region above the leaching fields (Fig. 6). The extent of this contamination (the limiting value has been chosen at 50 mg. L\(^{-1}\) of sulphates) has been stabilised by the pumping and cleaning of approximately 3 m\(^3\). min\(^{-1}\) contaminated water to the surface. A reduction of the contaminated area would be possible by pumping and cleaning of greater volumes (approx. about 5 m\(^3\). min\(^{-1}\) ), but is restricted by the capacity of the purification plants and the allowed volume of water release into the Ploučnice river. The most restricting parameter is the sulphate content of these waters.

The volume of contaminated Turonian water has been estimated to be approximately 2 x 10\(^6\)m\(^3\). Recently, MEGA proposed another estimate, based on indirect geophysical measurements, according to which the volume of contaminated Turonian waters is about ten times as great, i.e. 20 x 10\(^6\)m\(^3\). By an inspection well, drilled on one critical point of the geophysical profile, however, this conclusion was disapproved.
4. RECLAIMING OF THE SURFACE OF THE LEACHING FIELDS

At present any programme of remedial work on the ISL area depends on the future of the uranium extraction in the district, as the DIAMO, the former Czechoslovak Uranium Industry, is still a state enterprise.

In some parts of the leaching fields, where originally have been fields (around the old farms "Velké Ralsko" and "Malé Ralsko") a system of the so-called "reclamation during the operation" was tried with apparent success. It is based on a layout of the wells and other installations, which allow the access of agricultural machines between the installations. In these fields mainly green forage is been grown and used under radiation supervision.

The first exhausting of a leaching field in an area in the immediate surrounding of Stráž, where during former times regular farming had been performed, gave the opportunity to start the first reclamation programme. Last year, after a thorough clearance of the wells and pipelines, a five-year programme of biological reclaiming was put into operation. This programme includes the growing of plants which, after the field being ploughed, may serve of the renovation of the humus content of the soil. The first results of this experiment will be available during the following years.

Under preparation is another general study of the reclamation of the whole leaching area, which is being designed in accordance with the "Regional System of Securing the Ecological Stability". This study may serve as an ideal model for the preparation of separate projects for the successive reclamation of the individual leaching fields in future.

The most important item in preparing the reclamation work is the pressure, exerted by the legislative regulations, which enforce the growing in this areas of productive woods which, generally, are attractive to the future user of the woodlands only in the form of pure pine monocultures (the whole area was used by the Soviet army as military exercise ground). The preparation, however, of such woods requires a repetition of the once already performed thorough deforestation of the area. This would repeatedly expose the remaining soil to weathering and expose the fields another time to erosion and degradation. Great efforts are made recently to convince the authorities to allow here to grow mixed woods with the so-called "Wallenstein family forest mixture", consisting of pines and spruces, which are on the periphery combined with larches and deciduous trees. As a whole, the reclamation of such an area of poorly nutritive soils is a problem, the details of which are yet to be solved.

References

RESTORATION OF GROUNDWATER
AT THREE IN-SITU URANIUM MINES IN TEXAS

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Abstract

Energy Fuels Nuclear, through its affiliate, Intercontinental Energy Corporation, is currently restoring the surface and groundwater for three in-situ leaching projects in the South Texas uranium district of the U.S.A. These projects were undertaken on a contract basis in 1987 and are due for completion in 1994.

The South Texas uranium district has been the principle ISL locale in the U.S. since 1974 when the first commercial ISL project at George West began operation. This district is characterized by moderate depths, an alternating sequence of sandstones and shales, relatively high permeabilities and the use of unusually highly-concentrated ammonium bicarbonate lixiviants which have posed serious problems in restoration.

Restoration of these projects is a combined technical, political, regulatory and economic exercise, each facet of which must be carefully considered in each of those contexts.

The restoration process has several major components: surface restoration, drill hole plugging, groundwater restoration, and waste disposal. Groundwater restoration and waste restoration are interrelated variables which on these projects, involve the concurrent use of groundwater sweep, reverse osmosis, deepwell injection and irrigation disposal. Balancing the flow rates and chemical characteristics of these four components was a major task; but upon achieving the balance, the cost of restoration has been reduced to a practical minimum.

The principal cost control variable in ISL groundwater restoration is flow rate. Increasing flow rates will reduce the period of time necessary to treat the number of pore volumes required to achieve a given set of restoration criteria. Another major cost control variable is the restoration criteria itself. In many cases, it may be possible to achieve some relief from specific onerous standards by: 1) working to change those standards, or 2) convincing the regulatory authorities that "best available technology" won't solve the problem and that a variance is appropriate.

1. INTRODUCTION

Intercontinental Energy Corporation, a Concord Mining Unit subsidiary and an Energy Fuels Nuclear affiliate, is reclaiming three in-situ uranium mines in South Texas. The three mines are the Pawnee, the Zamzow and the Lamprecht. These mines operated using very high concentrations of ammonium bicarbonate and hydrogen peroxide or oxygen. As a result, the restoration of the groundwater to pre-mining conditions is especially difficult. The Zamzow and Lamprecht Mines are adjacent and the Zamzow Mine is adjacent to a reclaimed open pit uranium mine, the Felder Open Pit uranium mine. The restoration of groundwater is further complicated because the aquifer outcrops in the reclaimed open pit mine. The Zamzow and Lamprecht groundwater restoration project is, to my knowledge, the largest groundwater restoration project ever undertaken since it is reclaiming 2 miles (3.2 kilometers) of aquifer.

Restoration of the groundwater is accomplished by the use of groundwater sweep, followed by reverse osmosis for processing the contaminated water to produce very clean water for reinjection. Aquifer recharge water from outside the mine aquifer is used to supplement the
FIG. 1.

INTERCONTINENTAL ENERGY CORPORATION
LAMPRECHT-ZAMZOW PROJECTS
LOCATION MAP
natural recharge and to provide improved water balance and control of flow distribution. Waste water is disposed of in deep disposal wells or treated and used for irrigation of crops and pastures.

2. DESCRIPTION OF PROJECTS

Intercontinental Energy Corporation (IEC) was acquired by the Concord Mining Unit in 1987. I was employed in July 1990 managing the restoration of the IEC in-situ uranium mines. These mines are the Zamzow, Lamprecht, and Pawnee in-situ uranium mines located in Bee and Live Oak Counties in Texas. These projects are located about half way between San Antonio and Corpus Christi, Texas. The location of Live Oak County is shown in Figure I.

Zamzow

The Zamzow is a small in-situ uranium mine operated from 1979 until 1988 by Intercontinental Energy Corporation. The project produced about 1.3 million pounds of uranium. There are 39.2 acres of well fields containing 2291 wells. The wells are 80 to 250 feet deep (24 to 76 meters). The project used an ammonium carbonate leach system and oxygen as the oxidant. The ore zone is the Oakville aquifer. The ore extended to the east and west in a continuous "roll-front" deposit more than two miles long (3.2 kilometers). The Zamzow was bounded on the west by the Felder open pit uranium mine. The Felder mine was operated by Exxon from 1977 to 1983 and produced 5,000,000 pounds of uranium at grades around 0.24%. The two mines were operated at the same time. Both mines were mined up to the boundary between the projects. The Felder was mined to the boundary first, and by agreement the pit wall was cut to slope up over the boundary on to the Zamzow. After that portion of the ore was mined out, the pit wall was backfilled so that the Zamzow in-situ wells could be located very near the boundary. The Zamzow is adjoined on the east by the Lamprecht in-situ uranium mine.

Lamprecht

The Lamprecht Mine was operated by the Westinghouse subsidiary, Wyoming Minerals, from 1976 to 1983 when it was acquired by IEC. The mine produced about 1.5 million pounds of uranium during this time period. There are 59.2 acres of well fields on the Lamprecht project. The wellfields contain 2852 wells from 180 to 280 feet deep (55 to 85 meters). The three mines, Felder Open Pit, Zamzow In-situ and Lamprecht In-situ, produced about 8 million pounds of uranium from the same continuous ore deposit in the Oakville Aquifer. The aquifer outcrops in the pit wall of the Felder Pit. The area in underground water restoration in both projects is about 100 acres (40 Hectares) with a length of two miles (3.2 kilometers) and a width of up to 800 feet (242 meters).

Pawnee

The Pawnee is a very small in-situ mine operated by IEC from 1977 to 1979. The project only produced 80,000 lbs. of uranium. The project contains 7 acres (2.8 Hectares) of wellfields with 318 wells about 220 to 280 feet deep (67-85 metres). This project is 15 miles (39 kilometers) from the Zamzow-Lamprecht, and is also in the Oakville Aquifer.
INTERCONTINENTAL ENERGY CORPORATION
LAMPRECHT-ZAMZOW PROJECTS
GROUNDWATER SWEEP
RESTORATION PROCESS

TO LAND APPLICATION
OR DEEP DISPOSAL WELL

CONTAMINATED
GROUNDWATER

INACTIVE WELL

PUMPING WELL

INFLOW OF FRESH
GROUNDWATER

OVERBURDEN

CLAY

PRODUCTION/RESTORATION
ZONE

INFLOW OF FRESH
GROUNDWATER

FIG. 2.
3. PROCESSES USED IN GROUNDWATER RESTORATION

There are two commonly used processes in groundwater restoration. They are called "Groundwater Sweep" and "Reverse Osmosis". We quite often use both. We optimize the combination of the two processes to utilize the most effective aspects of each.

**Groundwater Sweep**

This method of groundwater restoration consists of pumping out the contaminated groundwater from the center of the contaminated area and disposing of the waste water. This induces an influx of groundwater from the surrounding aquifer. The process is illustrated in Figure 2. The water from outside the contaminated area moves through the pore space in the formation displacing the contaminated water. The process is called "Ground Water Sweep" because the natural groundwater sweeps the contaminated water ahead to the pumping wells. Flow into each well is radial so we locate a row of wells down the center to ensure more uniform flow to the center of the contaminated area from the outside clean groundwater.

The process is not as simple as displacement. The clean water has lower ionic concentrations than the contaminated water. During mining, the cations (positive ions), especially ammonia, are ion exchanged on to the clays from the high concentrations in the leach solution. During restoration the lower concentrations of ions in the natural groundwater causes the exchange of ions off the clays into the water raising the concentration in the groundwater as it moves toward the pumping wells. The cleaner groundwater strips the ammonia and other cations off the clays. Since the ammonia was loaded on the clays with at least 100 pore volumes (volume of the pore space in the well field), it must take several pore volumes to strip the ammonia off the clays until the ammonia in the water is sufficiently low to meet the restoration criteria.

In order to keep this process efficient, there is a need to minimize both vertical or horizontal channeling so that the sweep is locally uniform and efficient.

There is a very rapid reduction in groundwater contamination in the first 3 pore volumes. In our experience, it takes 6 to 10 pore volumes of groundwater sweep to reach satisfactory water quality. If ammonia is present, many more pore volumes are required depending on the level of ammonia used as a restoration goal.

**Reverse Osmosis**

With this method, the contaminated groundwater is pumped out of the ground and to a reverse osmosis unit. Before flowing to the reverse osmosis unit, the water is filtered, the pH is lowered to prevent calcium carbonate plugging of the membranes and the water is fed to a high pressure pump that increases the pressure to around 550 psig. At that pressure, about 60 to 75% of the water is forced through the membranes leaving about 70 to 90% of the salts in the brine on the high pressure side of the membrane. The brine, 25 to 40% of the feed, is sent to disposal. The clean water (permeate) is pumped back to and reinjected into the well fields. The process we use is illustrated in Figures 3 and 4. This process is useful for two reasons: 1) Because there is a hydrological limit to the quantity of water that can be pumped out of an aquifer, it is always possible to circulate several times more water than can be pumped out. For example, it is possible to pump out about 250 gpm from the Zamzow and Lamprecht areas but the wells will draw down over a couple of months to yield less water if more is pumped out.
1st STAGE RO

FLOW FROM PUMPING WELLS
6000 μmhos
CONDUCTIVITY
950 gpm

2nd STAGE RO

AQUIFER RECHARGE WATER
2500 μmhos
250 gpm

TO WELL FIELD INJECTION WELLS
2100 μmhos
870 gpm

PERMEATE
2000 μmhos
620 gpm

BRINE TO WASTE DISPOSAL WELLS
13,500 μmhos
330 gpm

INTERCONTINENTAL ENERGY CORPORATION
LAMPRECHT-ZAMZOW PROJECTS

REVERSE OSMOSIS PROCESS

FIG. 3.
FIG. 4.
However, we have been pumping out 1250 gpm and reinjecting 1030 gpm. 2) Since the permeate is so much better quality than groundwater, the contamination is removed faster. The exchange of the ions from the clays to the water is faster since there is a larger difference in concentration.

One of the problems with this process is that it requires a substantial investment in equipment and the operating costs are higher. Other processes for providing a source of clean water may be used such as electrodialysis; however, I believe that Reverse Osmosis is the lowest cost process.

Quite often it is recommended that scale inhibitors be used. We have not found that to be necessary for these projects. Also, the recommendation from the membrane manufacturers was to lower the pH to 5.5. We have found that we can operate successfully at 6.5 pH (a reduction of only pH 0.5) with considerable savings in sulfuric acid. The units we use are spiral wound, cellulose acetate membranes. We converted the units to two stage units to increase the capacity and lower operating costs. We have not needed to clean or replace the membranes in 20 months of service. We are feeding conductivities from 4500 to 6500 micro mhos. We are achieving a 60 to 75% yield of permeate with conductivities of 1600 micro mhos. The brine runs around 12000 micro mhos. However, these projects used high ammonia and carbonate concentrations when they were being mined. The water at the end of mining in the wellfield will contain 600 to 1300 ppm ammonia and 1500 to 3500 ppm carbon dioxide. Table I shows typical analyses of feed

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Units</th>
<th>Feed Water</th>
<th>Brine</th>
<th>Treated Water</th>
<th>% Change</th>
</tr>
</thead>
<tbody>
<tr>
<td>Uranium (U)</td>
<td>mg/liter</td>
<td>5.0</td>
<td>12.3</td>
<td>&lt; 0.4</td>
<td>92</td>
</tr>
<tr>
<td>Molybdenum (Mo)</td>
<td>mg/liter</td>
<td>8.2</td>
<td>22.3</td>
<td>1.0</td>
<td>88</td>
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<tr>
<td>Ammonia (NH₄)</td>
<td>mg/liter</td>
<td>425.0</td>
<td>991.0</td>
<td>128.0</td>
<td>70</td>
</tr>
<tr>
<td>Sulphate (SO₄)</td>
<td>mg/liter</td>
<td>713.0</td>
<td>3059.0</td>
<td>149.0</td>
<td>79</td>
</tr>
<tr>
<td>Calcium (Ca)</td>
<td>mg/liter</td>
<td>141.0</td>
<td>310.0</td>
<td>25.0</td>
<td>82</td>
</tr>
<tr>
<td>Sodium (Na)</td>
<td>mg/liter</td>
<td>360.0</td>
<td>945.0</td>
<td>118.0</td>
<td>67</td>
</tr>
<tr>
<td>Chloride (Cl)</td>
<td>mg/liter</td>
<td>767.0</td>
<td>1784.0</td>
<td>352.0</td>
<td>54</td>
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<tr>
<td>Carbonate (CO₃)</td>
<td>mg/liter</td>
<td>1050.0</td>
<td>1370.0</td>
<td>420.0</td>
<td>60</td>
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<tr>
<td>Total Dissolved Solids (TDS)</td>
<td>mg/liter</td>
<td>2336.0</td>
<td>6350.0</td>
<td>746.0</td>
<td>68</td>
</tr>
<tr>
<td>Silica (SiO₂)</td>
<td>mg/liter</td>
<td>31.0</td>
<td>68.0</td>
<td>9.0</td>
<td>71</td>
</tr>
<tr>
<td>pH</td>
<td>units</td>
<td>7.1</td>
<td>6.8</td>
<td>5.8</td>
<td>18</td>
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<tr>
<td>Electrical (EC) Conductivity</td>
<td>micro mhos</td>
<td>4860.0</td>
<td>11450.0</td>
<td>1570.0</td>
<td>68</td>
</tr>
<tr>
<td>Radium (Ra)</td>
<td>pico curies per liter</td>
<td>333.0</td>
<td>815.0</td>
<td>5.0</td>
<td>98</td>
</tr>
</tbody>
</table>
concentration, permeate and brine concentrations. This data shows that the reverse osmosis is particularly effective in removing radium, uranium, and molybdenum. To increase the capacity of the reverse osmosis units, we reduced the number of stages from three to two. This allowed us to increase the feed rate by 42 percent with an increase in the permeate of 25%. With two stages, we reduced the feed pressure requirements by 15% with corresponding power savings per gpm circulated.

Since we had a source of groundwater from outside the aquifer, we were able to supplement the yield of permeate with the water which we called Aquifer Recharge Water. This is baseline quality water. The Aquifer Recharge Water is baseline quality so it has higher conductivity, 3200 micro mhos, and TDS, 2200 ppm than the permeate but no ammonia.

Waste Disposal Wells

We utilize two waste disposal wells to dispose of waste water. We have one well at the Zamzow Project. This well is completed in the Yegua formation, a salt water (more than 10,000 TDS) bearing formation at depths of about 3600 feet (1091 meters). The other well is located on the Lamprecht project. It is used to inject waste water into the Wilcox Formation at depths of about 6600 feet (2000 meters). This formation also contains very salty water. It is a regulatory requirement that the water in a formation used for this purpose should be too salty for any use.

The waste water from the Zamzow and Lamprecht projects includes water pumped from the groundwater sweep operation, the brine from the reverse osmosis operation, and fluids containing radium and uranium from the decontaminated of radioactive equipment, pipe and other materials.

Protection of fresh ground is provided by the following:

1) A cemented surface casing to below the base of fresh water (1000 feet or 303 meters).
2) A cemented casing to the formation where injection occurs.
3) Injection through tubing with monitoring of the pressure in the annulus space between the tubing and the casing. A packer assembly is used to separate the annulus space from the injection fluid.

Irrigation with Waste Water

At the Pawnee site, we are treating the water pumped from groundwater sweep by adding barium chloride to precipitate and settle the radium in lined ponds. We are adding flocculent to improve the settling of the particles, and copper sulfate to control the algae growth. After chemical addition, the water flows into one pond using a flow distributor to distribute flow across the entire width of the pond. Small diameter plastic hose is floated across the ponds perpendicular to the flow of the water to stop wave action and improve settling. The water flows from the first pond to the second pond and is pumped by floating pumps from the second pond to irrigation. We use a 100 acre center pivot irrigation system to distribute the water.

The radium concentration in the water to the ponds is 200 to 400 picocuries dissolved and 200 to 400 picocuries total (All in solution). We are able to reliably lower the radium to 1 picocurie dissolved and 10 picocuries total. While there is no health problem as long as we keep the radium in the soil less than 5 picocuries per gram, we are regulated on this project to 10
picocuries dissolved and 30 picocuries total in the water to irrigation. We are limited to 44 ppm uranium in the soil.

Description of Restoration Projects

The first priority in establishing a restoration plan for this project was to evaluate the capacity of the facilities which were at the projects and develop a cost effective plan considering the ability of the aquifer to yield water and the equipment capabilities. An important consideration was to evaluate what license and permit amendments could be achieved in a reasonable time period to better use the capabilities present at these projects. Within a few months, a plan was developed to invest about $700,000 in equipment to reduce the groundwater restoration time from 10 years to 3 years. This investment would save about $1,000,000 for every year of reduced time due to the high fixed costs of monitoring, and maintaining the operations. However, the savings would be offset by higher operating costs for moving water more quickly. The net savings would be about $3,700,000 in groundwater reclamation costs.

The projects had on site two complete in-situ process plants, extensive quantities of used wellfield equipment, and extensive plastic (PVC) piping. This equipment was used during the mining process. We planned to utilize as much of this equipment as possible, however, much of it was a reclamation liability since it was not needed, and was in poor condition since it had not been maintained for years.

Lamprecht - Zamzow Restoration

Since the Lamprecht and Zamzow projects are adjacent and in the same aquifer, it was necessary to develop a plan for both projects. The capabilities on both sites when we acquired the projects were two 250 gpm (500 gpm total), three stage reverse osmosis units, and two waste disposal wells (one with each project). The Zamzow waste disposal well was permitted for 130 gpm of waste water disposal and the Lamprecht was permitted for 150 gpm. Both projects had been operated for almost two years in "groundwater sweep". During that time 0.55 pore volume had been removed from the Zamzow Project and 0.39 pore volume had been removed from the Lamprecht project.

Upon analysis of the cost of the project, it was obvious that the projects needed to move more groundwater each month to reduce the total time to complete. This would make more effective use of the personnel and other fixed costs such as license fees, landowner rentals, monitor well sampling costs, etc. We were averaging costs of about $80,000 per month. We had ten employees on site including a manager and a secretary/clerk. Some general management and accounting functions were provided by Denver based personnel. Of the $80,000 per month, only $10,000 per month was the cost of pumping water. The ratio of electric power costs to total costs can be used to determine the effectiveness of the costs. We needed to utilize the capabilities of the reverse osmosis units and the disposal wells more effectively to speed up the reclamation to reduce total costs.

A plan was developed to accomplish the following:

1) Redistribute the pumping wells to provide more efficient flow distribution with the existing "groundwater sweep".
2) Amend the waste disposal well permits from 130 gpm and 150 gpm (280 gpm total) to 250 gpm each (500 gpm total). This would allow operation of 500 gpm of reverse osmosis while continuing to run more than 300 gpm of "groundwater sweep". It would be necessary to dispose of 200 gpm of brine from the reverse osmosis unit so that an increase in disposal well capacity was essential to speed up completion of the project. Later, we tested the wells and amended the permits again to 450 gpm each for a total capacity of 900 gpm. We only could use 700 gpm of the capacity without having very large capital equipment expenditures.

3) Construct and operate a 60 gpm reverse osmosis unit pilot plant from the 3rd stage of one of the reverse osmosis units. The pilot operation would be used to test the operating conditions and costs of reverse osmosis with the groundwater quality to optimize the restoration plans. We tested operating without scale inhibitor and iron removal processes. Based on the evaluation of the contaminated groundwater quality by reverse osmosis process equipment manufacturers, we had been advised that we could not operate until we had lowered the water conductivity to 4000 micro mhos. The previous plan would require us to use groundwater sweep until the conductivity had been lowered to 4000 micro mhos. We would test the pilot on 6000 micro mhos conductivity feed water to determine the cost of cleaning and replacing the membranes at these high concentrations. The only treatment we would use would be to lower the pH to 5.5 as recommended. It would not be prudent to operate the $130,000 of reverse osmosis membranes that were on hand until we were sure we would not damage the membranes.

4) Rebuild the reverse osmosis units to make them operational and to convert the units to two-stage units.

5) Build an additional two-stage unit from the third-stage of the other unit. It was anticipated that this would raise the flow rate from 500 gpm to 620 gpm, including the 60 gpm pilot.

6) De-stage the high pressure feed pumps to reduce power costs.

7) Increase the disposal well capacity by replacing the tubing with larger diameter tubing, installing the spare high pressure pump and motor and other modifications. We also improved the filtration of the water injected in the disposal wells. This was important to increase the capacity.

This plan was based on the concept of utilizing two or more pore volumes of groundwater sweep to lower the conductivity and TDS to a level that would provide lower operating costs for the reverse osmosis unit. The restoration would then be completed with two or three pore volumes of reverse osmosis. This concept was based on previous restoration projects.

During the time we were waiting on permit amendments for the waste disposal wells, constructing the pilot unit and filter modifications, we acquired a 200 gpm three-stage used reverse osmosis unit from Westinghouse. We converted this unit to two two-stage units with a total capacity of 350 gpm. Addition of this unit would increase the total reverse osmosis capacity from 620 gpm to 970 gpm. This would reduce the time to complete the restoration by 36 percent. The reduction in time would result in saving of at least a 1/3 which would amount to savings of about $1,000,000. The savings would be offset by the acquisition costs of $100,000 (about half of new costs). Since the volume of water that must be pumped out and circulated is the same, regardless of how quickly we circulate the water, we know that there was a substantial absolute savings of about $900,000.

We spent about $700,000 on equipment to reduce the restoration time from 10 years to 3 years. The operation costs were increased from $80,000 per month to $144,000 per month. The electric power costs were increased from $10,000 per month (11.5% of costs) to $43,000
FIG. 5.

INTERCONTINENTAL ENERGY CORPORATION

ZAMZOW - LAMPRECHT PROJECTS

JUNE 30, 1991
INTERCONTINENTAL ENERGY CORPORATION

ZAMZOW – LAMPRECHT PROJECTS

AUG. 31, 1991

FIG 6
MAP KEY

LIVE OAK COUNTY, TEXAS

LEGEND
- INJECTION WELL
- PUMPING WELL
- MONITOR WELL
- AREA OF INFLUENCE
- WELLFIELD

INTERCONTINENTAL ENERGY CORPORATION
ZAMZOW -- LAMPRECHT PROJECTS
DEC. 31, 1991

FIG. 7.
FIG. 8.

INTERCONTINENTAL ENERGY CORPORATION

ZAMZOW - LAMPRECHT PROJECTS

MARCH 31, 1992
MAP KEY

LIVE OAK COUNTY, TEXAS

LEGEND

- INJECTION WELL
- PUMPING WELL
- MONITOR WELL
- AREA OF INFLUENCE
  WELLFIELD

INTERCONTINENTAL ENERGY CORPORATION
ZAMZOW - LAMPRECHT PROJECTS
AS OF
SEPT. 1, 1992

FIG 9
per month (30% of costs). The savings between 10 years at $80,000 per month and 3 years at $144,000 per month is $4,400,000.

A new concept we developed about this time was to place the pumped wells on the perimeter of the well field and inject the permeate in the center during the reverse osmosis operation. This has the effect of allowing the reverse osmosis operation to start sooner. Since the groundwater sweep operation pumped wells located in the center of the well field, the edge wells would generally be lower in conductivity and total dissolved solids (TDS) so that we could begin the reverse osmosis phase sooner by pumping wells on the edge of the contaminated area. The pilot plant demonstrated that we could operate economically at feed conductivities of 6500 micro mhos without scale inhibitor or rust removal. Later, operation showed that we could reduce sulfuric acid consumption by lowering the pH to 6.5 instead of 5.5. In 20 months of operation, we have not cleaned or replaced the membranes.

An additional aspect of the plan was to increase the flow by adding aquifer recharge water of baseline water quality from wells located nearby but outside the mine aquifer. It was necessary to initiate and expand reverse osmosis operation on the Lamprecht while we maintained and increased the groundwater sweep on the Zamzow. When the Lamprecht was partially restored, we would move a portion of the reverse osmosis operation (well field and pipeline) to the Zamzow until all the operation was moved to the Zamzow.

The attached maps, Figures 5 through 9, show the progress in increasing reverse osmosis flow by adding pumped wells and injection wells. We monitored the conductivity and ammonia concentration weekly. When the conductivity becomes less than 3800 and the ammonia less the 250 ppm, we move the pumped wells. The amended restoration table was established at 3800

### TABLE II
**LAMPRECHT WATER QUALITY COMPARISON**

IEC Lamprecht Project
September, 1992

<table>
<thead>
<tr>
<th>Concentration</th>
<th>Units</th>
<th>Pre-Mining Baseline</th>
<th>Typical End of Mining</th>
<th>Amended Restoration Table</th>
<th>Current Restoration Water Quality</th>
</tr>
</thead>
<tbody>
<tr>
<td>Uranium</td>
<td>ppm</td>
<td>&lt; 1</td>
<td>2.3-8.3</td>
<td>5</td>
<td>2</td>
</tr>
<tr>
<td>Molybdenum</td>
<td>ppm</td>
<td>&lt; 1</td>
<td>11-28</td>
<td>5</td>
<td>2</td>
</tr>
<tr>
<td>Ammonia</td>
<td>ppm</td>
<td>&lt; 1</td>
<td>751-1257</td>
<td>200</td>
<td>173</td>
</tr>
<tr>
<td>Sulfate</td>
<td>ppm</td>
<td>523</td>
<td>1619-1970</td>
<td>600</td>
<td>360</td>
</tr>
<tr>
<td>Calcium</td>
<td>ppm</td>
<td>198</td>
<td>28-231</td>
<td>240</td>
<td>142</td>
</tr>
<tr>
<td>Sodium</td>
<td>ppm</td>
<td>444</td>
<td>652-673</td>
<td>440</td>
<td>231</td>
</tr>
<tr>
<td>Chloride</td>
<td>ppm</td>
<td>574</td>
<td>1362-1580</td>
<td>600</td>
<td>511</td>
</tr>
<tr>
<td>Carbonate</td>
<td>ppm</td>
<td>303</td>
<td>1541-3504</td>
<td>1000</td>
<td>699</td>
</tr>
<tr>
<td>TDS</td>
<td>ppm</td>
<td>2059</td>
<td>3030-3320</td>
<td>2059</td>
<td>1401</td>
</tr>
<tr>
<td>Conductivity</td>
<td>Micro mhos</td>
<td>3221</td>
<td>10400-12700</td>
<td>3800</td>
<td>3056</td>
</tr>
<tr>
<td>Radium</td>
<td>Pico curies</td>
<td>218</td>
<td>354-1020</td>
<td>218</td>
<td>&gt; 100</td>
</tr>
</tbody>
</table>

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### TABLE III
**LAMPRECHT RESTORATION**
**TOTAL MINE AREA**

Pore Volume = 171,000,000 gallons

<table>
<thead>
<tr>
<th>Date</th>
<th>GWS Removed Volume (gal)</th>
<th>GWS Treated Volume (gal)</th>
<th>RO Brine Removed Volume (gal)</th>
<th>RO Permeate Injected Volume (gal)</th>
<th>RO Fresh Water Injected Volume (gal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1991</td>
<td>5,973,295</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>January</td>
<td>4,977,801</td>
<td>2,918,007</td>
<td>3,130,887</td>
<td>3,792,991</td>
<td>5,643,006</td>
</tr>
<tr>
<td>March</td>
<td>864,000</td>
<td>2,376,000</td>
<td>2,452,838</td>
<td>4,109,000</td>
<td>5,067,386</td>
</tr>
<tr>
<td>April</td>
<td>4,400,736</td>
<td>3,260,000</td>
<td>880,227</td>
<td>1,787,893</td>
<td>1,600,590</td>
</tr>
<tr>
<td>May</td>
<td>5,643,006</td>
<td>10,738,695</td>
<td>3,388,602</td>
<td>3,735,093</td>
<td>2,000,162</td>
</tr>
<tr>
<td>June</td>
<td>5,289,000</td>
<td>20,562,330</td>
<td>6,577,070</td>
<td>13,985,260</td>
<td>1,666,819</td>
</tr>
<tr>
<td>July</td>
<td>3,526,805</td>
<td>21,319,852</td>
<td>6,584,145</td>
<td>14,735,707</td>
<td>2,595,456</td>
</tr>
<tr>
<td>August</td>
<td>4,613,068</td>
<td>27,352,628</td>
<td>9,393,437</td>
<td>17,959,191</td>
<td>1,263,346</td>
</tr>
<tr>
<td>September</td>
<td>4,788,404</td>
<td>31,579,200</td>
<td>10,238,602</td>
<td>21,340,800</td>
<td>2,116,800</td>
</tr>
<tr>
<td>October</td>
<td>4,996,608</td>
<td>33,496,080</td>
<td>9,216,918</td>
<td>14,279,162</td>
<td>5,516,728</td>
</tr>
<tr>
<td>November</td>
<td>54,556,908</td>
<td>145,809,009</td>
<td>48,858,589</td>
<td>96,950,420</td>
<td>14,059,076</td>
</tr>
<tr>
<td>December</td>
<td>648,539</td>
<td>34,830,170</td>
<td>11,745,337</td>
<td>23,084,833</td>
<td>10,401,120</td>
</tr>
<tr>
<td>January</td>
<td>576,439</td>
<td>42,333,214</td>
<td>11,559,574</td>
<td>30,773,640</td>
<td>9,744,141</td>
</tr>
<tr>
<td>February</td>
<td>609,965</td>
<td>35,180,010</td>
<td>9,201,797</td>
<td>25,978,213</td>
<td>7,843,946</td>
</tr>
<tr>
<td>March</td>
<td>1,385,892</td>
<td>39,942,452</td>
<td>9,744,558</td>
<td>30,197,894</td>
<td>5,691,706</td>
</tr>
<tr>
<td>April</td>
<td>1,787,893</td>
<td>35,700,960</td>
<td>13,771,296</td>
<td>21,929,664</td>
<td>12,343,258</td>
</tr>
<tr>
<td>May</td>
<td>5,716,000</td>
<td>34,457,700</td>
<td>10,959,867</td>
<td>23,497,833</td>
<td>9,705,867</td>
</tr>
<tr>
<td>June</td>
<td>7,921,060</td>
<td>35,919,230</td>
<td>12,144,216</td>
<td>23,775,014</td>
<td>10,011,106</td>
</tr>
<tr>
<td>July</td>
<td>20,806,520</td>
<td>296,022,486</td>
<td>90,871,982</td>
<td>205,150,504</td>
<td>76,807,544</td>
</tr>
</tbody>
</table>

GWS - Ground Water Sweep  
RO - Reverse Osmosis System

Conductivity and 200 ppm ammonia. The water injected into the wellfields averages 1400 conductivity and 80 ppm ammonia so we could meet the average restoration table conditions for all parameters with 3800 conductivity and 250 ppm ammonia as shut off criteria for pumping wells. Table II compares baseline water quality and end of mining water quality with the amended restoration table which is the criteria for restoration. By only monitoring conductivity until we approached 4000 and ammonia only after that point, we keep the analytical costs lower.
TABLE IV
ZAMZOW RESTORATION
Total Mine Area
Pore Volume = 135,000,000 gallons

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>1988</td>
<td>10,319,397</td>
<td>10,319,397</td>
<td>0.08</td>
<td>5,585.125</td>
<td>0.31</td>
<td>20,561,645</td>
<td>0</td>
</tr>
<tr>
<td>1989</td>
<td>30,923,721</td>
<td>41,243,118</td>
<td>0.31</td>
<td>14,976,520</td>
<td>0.31</td>
<td>20,561,645</td>
<td>0.31</td>
</tr>
<tr>
<td>1990</td>
<td>28,155,209</td>
<td>69,398,327</td>
<td>0.31</td>
<td>0</td>
<td>0</td>
<td>20,561,645</td>
<td>0</td>
</tr>
<tr>
<td>1991</td>
<td>59,470,185</td>
<td>128,868,512</td>
<td>0.95</td>
<td>34,399,833</td>
<td>0.95</td>
<td>54,961,478</td>
<td>0</td>
</tr>
<tr>
<td>1992</td>
<td>44,691,426</td>
<td>173,559,938</td>
<td>1.29</td>
<td>12,141,753</td>
<td>1.29</td>
<td>57,103,231</td>
<td>1.29</td>
</tr>
</tbody>
</table>

1992
January 4,735,034 133,603,546 0.99 0 0
February 6,144,096 139,747,642 1.04 0 0
March 6,150,000 145,897,642 1.08 1,849,050 1,849,050
April 6,120,433 152,018,075 1.13 1,838,145 3,687,195
May 7,010,287 159,028,362 1.16 2,116,611 5,803,806
June 2,098,669 161,127,031 1.19 2,269,947 8,073,753
July 5,196,963 166,323,994 1.23 3,348,000 11,421,753
August 7,235,944 173,559,938 1.29 720,000 12,141,753

Sub-Total 44,691,426 173,559,938 1.29 12,141,753 12,141,753

Volume in Gallons.
Pore Volume calculated Oscar Paulson in 1984.

Analytical costs for one full set of analysis for one water sample cost about $250 in a contract laboratory. Costs for analysis of conductivity, ammonia and uranium in our laboratory are negligible. Since we were pumping 240 wells at the three projects, the analytical costs could be significant.

At this time, we have pumped 2.2 pore volumes of ground water sweep and 2.8 pore volumes of reverse osmosis from the Lamprechtt and 1.3 pore volumes of groundwater sweep on the Zamzow. We used 171 million gallons as a pore volume on the Lamprecht Project and 135 million gallons as a pore volume on the Zamzow Project. This assumes a 20 foot thick contaminated interval which is more than twice as thick as the injection interval. The area contaminated outside the wellfield was assumed to be 48 percent of the wellfield area for the Lamprecht Project and 76 percent for the Zamzow.

As of October 1, 1992, we have completely restored about 40 acres (16 Hectares) (67%) of the Lamprecht well fields and have substantial progress on the remaining areas. We expect to operate three more months at 400 gpm to complete the Lamprecht restoration around the end of the year. We should complete the Zamzow restoration about next October. This will be about two pore volumes of groundwater sweep and three pore volumes of reverse osmosis for both projects. Progress in achieving our flow schedule is shown in Tables III and IV for the
**TABLE V**

**LAMPRECHT WASTE DISPOSAL WELL**
**INJECTION VOLUME HISTORY**
**WDW-156**

<table>
<thead>
<tr>
<th>Year</th>
<th>Yearly Volume (gal)</th>
<th>Cumulative Volume (gal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1979</td>
<td>10,510,866</td>
<td>10,510,866</td>
</tr>
<tr>
<td>1980</td>
<td>No Records</td>
<td>10,510,866</td>
</tr>
<tr>
<td>1981</td>
<td>10,975,180</td>
<td>21,486,046</td>
</tr>
<tr>
<td>1982</td>
<td>5,554,430</td>
<td>27,040,476</td>
</tr>
<tr>
<td>1983</td>
<td>21,962,024</td>
<td>49,002,500</td>
</tr>
<tr>
<td>1984</td>
<td>31,059,960</td>
<td>80,062,460</td>
</tr>
<tr>
<td>1985</td>
<td>1,075,950</td>
<td>81,138,410</td>
</tr>
<tr>
<td>1986</td>
<td>35,769,634</td>
<td>116,908,044</td>
</tr>
<tr>
<td>1987</td>
<td>63,303,893</td>
<td>180,211,937</td>
</tr>
<tr>
<td>1988</td>
<td>56,853,958</td>
<td>237,065,895</td>
</tr>
<tr>
<td>1989</td>
<td>57,621,948</td>
<td>294,687,843</td>
</tr>
<tr>
<td>1990</td>
<td>62,792,356</td>
<td>357,480,199</td>
</tr>
<tr>
<td>1991</td>
<td>83,603,809</td>
<td>441,084,008</td>
</tr>
<tr>
<td>1992</td>
<td>92,826,165</td>
<td>533,910,174</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Month</th>
<th>Monthly Volume (gal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>January</td>
<td>10,597,464</td>
</tr>
<tr>
<td>February</td>
<td>10,156,354</td>
</tr>
<tr>
<td>March</td>
<td>10,908,852</td>
</tr>
<tr>
<td>April</td>
<td>9,951,976</td>
</tr>
<tr>
<td>May</td>
<td>11,419,322</td>
</tr>
<tr>
<td>June</td>
<td>11,644,418</td>
</tr>
<tr>
<td>July</td>
<td>13,538,490</td>
</tr>
<tr>
<td>August</td>
<td>14,609,290</td>
</tr>
</tbody>
</table>

92,826,166

Lamprecht and Zamzow projects. These tables show progress in maximizing and sustaining flow rates. Progress in maximizing and sustaining waste disposal well flow rates is shown in Tables V and VI.

The total operating cost of the groundwater restoration is expected to be about $5,000,000 for pumping out about 1.5 billion gallons of water. This includes costs of about $128,000 per pore volume of groundwater sweep at $.95 per thousand gallons, and $368,000 per pore volume of reverse osmosis at $2.14 per gallon. Included in these costs are $.51 to $.63 per thousand gallons of water pumped down the waste disposal wells. Typical operating costs for the three projects is shown in Table VII. This table compares cost per pore volume and cost per thousand gallons pumped out of the well field. Average monthly operating costs are also shown.
### TABLE VI

**ZAMZOW WASTE DISPOSAL WELL**

**INJECTION VOLUME HISTORY**

**WDW-159**

<table>
<thead>
<tr>
<th>Year</th>
<th>Yearly Volume (gal)</th>
<th>Cumulative Volume (gal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1982</td>
<td>10,063,189</td>
<td>10,063,189</td>
</tr>
<tr>
<td>1983</td>
<td>16,649,113</td>
<td>26,712,302</td>
</tr>
<tr>
<td>1984</td>
<td>28,343,387</td>
<td>55,055,689</td>
</tr>
<tr>
<td>1985</td>
<td>22,383,821</td>
<td>77,439,510</td>
</tr>
<tr>
<td>1986</td>
<td>16,205,298</td>
<td>93,644,808</td>
</tr>
<tr>
<td>1987</td>
<td>13,310,185</td>
<td>106,954,993</td>
</tr>
<tr>
<td>1988</td>
<td>24,705,517</td>
<td>131,660,510</td>
</tr>
<tr>
<td>1989</td>
<td>39,429,011</td>
<td>171,089,521</td>
</tr>
<tr>
<td>1990</td>
<td>46,350,497</td>
<td>217,440,018</td>
</tr>
<tr>
<td>1991</td>
<td>75,234,107</td>
<td>292,674,125</td>
</tr>
<tr>
<td>1992</td>
<td>71,468,000</td>
<td>364,142,125</td>
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</tbody>
</table>

<table>
<thead>
<tr>
<th>Year</th>
<th>Monthly Volume (gal)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1992</td>
<td></td>
</tr>
<tr>
<td>January</td>
<td>9,618,240</td>
</tr>
<tr>
<td>February</td>
<td>8,949,220</td>
</tr>
<tr>
<td>March</td>
<td>10,447,840</td>
</tr>
<tr>
<td>April</td>
<td>8,371,760</td>
</tr>
<tr>
<td>May</td>
<td>9,213,330</td>
</tr>
<tr>
<td>June</td>
<td>6,013,440</td>
</tr>
<tr>
<td>July</td>
<td>8,334,340</td>
</tr>
<tr>
<td>August</td>
<td>10,519,830</td>
</tr>
<tr>
<td></td>
<td>71,468,000</td>
</tr>
</tbody>
</table>

We have on site twelve employees: an experienced manager, one secretary/clerk, one experienced laboratory supervisor (no assistants), one electrician, one mechanic, a well workover and construction crew of three (consisting of a supervisor, a well driller, and a helper), and an operating crew of four (consisting of a supervisor, a reverse osmosis operator, a Lamprecht/Zamzow wellfield operator, and a Pawnee wellfield operator). Other functions for this crew include: sampling monitor wells each month (quarterly at later stages of restoration), moving wells and maintaining well flow, construction projects, and decontamination of surface equipment and materials. We have a contract crew of two to three picking up and chopping up plastic pipe for decontamination for a period of about 12 months. We have dismantled and decontaminated the Lamprecht Process plant with a contract crew for $60,000.

In addition, we had a one time investment of about $700,000 in equipment such as $100,000 for the Westinghouse used reverse osmosis unit, submersible pumps, piping, electrical cable, reverse osmosis construction, disposal well modifications and a few large electric motors.
TABLE VII

TYPICAL GROUNDWATER RECLAMATION COSTS
1992

<table>
<thead>
<tr>
<th>Project</th>
<th>Method</th>
<th>$/PV</th>
<th>$/M Gal.</th>
<th>Av. Month</th>
<th>Percent of Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lamprecht</td>
<td>RO (1) + Limited GWS (2) + WDW (3)</td>
<td>$367,899</td>
<td>2.14</td>
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<tr>
<td>Zamzow</td>
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<td>0.95</td>
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<tr>
<td>Pawnee</td>
<td>GWS + Irrigation</td>
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<td>7,000</td>
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<tr>
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<td>18,414</td>
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<td>12.8</td>
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<tr>
<td>G&amp;A (Denver)</td>
<td></td>
<td>12,995</td>
<td></td>
<td></td>
<td>9.0</td>
</tr>
</tbody>
</table>

1 RO: Reverse Osmosis
2 GWS: Ground Water Sweep
3 WDW: Disposal Well Waste

$144,028 100.0

LABOR AND ELECTRICAL COSTS
1992

Labor at Aver. $11.59/Hour with benefits $17.12 $36,475 25.3
Electric Power at 6 cents/kwh - 744,000 kwh/month $42,675 29.6 $89,150 54.9

In addition, these projects came equipped with a large inventory of pumps, motors, piping, motor control panels, laboratory equipment, office and office equipment, portable building, well workover trucks, well sampling trucks, the two 250 gpm reverse osmosis units and a great many fiberglass tanks. This represents several million dollars worth of equipment acquired when we acquired the project. We plan to find another groundwater restoration project to use the equipment after this project is completed.

Pawnee Restoration

The Pawnee Project was pumping up to 150 gpm of ground water sweep to radium removal and irrigation of 60 acres. The pumping well locations are shown in Figure 10. Since previous management had accepted a 4 ppm molybdenum limit in the water to irrigation and the wellfield averaged 20 ppm molybdenum, the previous operation had to dilute the molybdenum concentration by pumping near one edge of the well field. This was not very efficient. Soon after I took over this project, we amended the permit to allow irrigation with water containing up to 26 ppm molybdenum. At the same time, we amended the permit to irrigate 100 acres with up to an average of 250 gpm.

We expect to achieve complete restoration at about 10 pore volumes after we amended the permit. We are currently pumping about a pore volume per month. We have removed five pore volumes since we amended the permits. Progress in achieving the schedule is shown in Table VIII. The cost of the groundwater sweep and irrigation is $7,000 per pore volume or $0.75
### TABLE VIII

**PAWNEE RESTORATION**

Pore Volume - 9,240,000 Gallons

<table>
<thead>
<tr>
<th>Date</th>
<th>Yearly Volume Removed</th>
<th>PV Volume Removed</th>
<th>Cumulative Volume Removed</th>
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<tr>
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<td>19,230,235</td>
<td>2.08</td>
<td>41,855,618</td>
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<td>1991</td>
<td>26,579,265</td>
<td>2.88</td>
<td>68,434,883</td>
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<tr>
<td>1992</td>
<td>24,590,320</td>
<td>2.66</td>
<td>93,025,203</td>
<td>10.07</td>
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</table>

<table>
<thead>
<tr>
<th>Date</th>
<th>Monthly Volume Removed</th>
<th>PV Volume Removed</th>
<th>Cumulative Volume Removed</th>
<th>PV Volume Removed</th>
</tr>
</thead>
<tbody>
<tr>
<td>01/92</td>
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<td>0.00</td>
<td>0</td>
<td>0.00</td>
</tr>
<tr>
<td>02/92</td>
<td>669,694</td>
<td>0.07</td>
<td>669,694</td>
<td>0.07</td>
</tr>
<tr>
<td>03/92</td>
<td>0</td>
<td>0.00</td>
<td>669,694</td>
<td>0.07</td>
</tr>
<tr>
<td>04/92</td>
<td>685,895</td>
<td>0.07</td>
<td>1,355,589</td>
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<td>05/92</td>
<td>4,140,691</td>
<td>0.45</td>
<td>5,496,280</td>
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<td>2.66</td>
</tr>
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</table>

Volume in gallons.

per thousand gallons. This is the lowest cost restoration process, but it takes longer than reverse osmosis and we have more trouble permitting this process.

We have one man who operates this facility. As indicated by Table VII, the operation costs for this project are very low per pore volume and lower per thousand gallons of water pumped than the other processes for groundwater restoration.

**CONCLUSION**

This report provides groundwater restoration costs that are applicable in the U.S. economy. The largest variables are estimating the volume of the contaminated zone and the number of pore volumes required to restore. An equally important factor is the criteria established or to be established for completion of restoration. Several processes are taking place underground. Usually oxidation must be suppressed. Introduction of a reductant is expensive but may sometime be necessary. The hydrological limitation of the aquifer must be considered as well.
as other sources of low cost suitable water in planning a ground water restoration project. We believe this report is interesting because it compares the costs of three different groundwater restoration/waste water disposal systems. The investment costs were not very significant in these costs since the major portion of the capital investment was provided for mining purposes and the major share of the reverse osmosis unit purchase costs were in place when the project was acquired. The comparison must be evaluated considering the total time to complete the project, since 22 percent of this monthly cost is non-variable, fixed annual costs.
The United States Congress has directed the Nuclear Regulatory Commission (NRC) to issue source material licenses and regulate in situ leach (ISL) uranium recovery operations. To obtain a license, a prospective operator must demonstrate an ability to operate an ISL uranium recovery operation that is protective of human health and the environment. The prospective licensee submits a technical description of the project complete with information necessary to evaluate the proposed operation in a license application. This application includes a site radiation safety program, a site-to-regional environmental assessment, a local demographic analysis, commitments to field and process facility operation procedures, an environmental monitoring program and production and site restoration schedules. The licensees also must verify they hold an agreement for byproduct waste disposal at a licensed waste disposal facility.

The above information comprises the basis for developing an Environmental Assessment or Environmental Impact Statement and a Safety Evaluation Report. The prospective licensee obtains a source material license once they meet all regulatory requirements and the noted reports are determined to be adequate.

During facility operations, compliance with obligations and commitments specified in the license is regulated through facility inspections. Furthermore, the licensee is required to submit periodic reports which track trends in environmental conditions and safety considerations.

Surface reclamation and ground-water restoration programs are implemented once uranium recovery activities within a mine unit are complete. The surface reclamation plan considers post-mining land use and reestablishment of surface drainage and vegetation. The primary goal of the ground-water restoration program is to return ground water to premining water quality. Following restoration activities, a stability monitoring period is required to verify ground-water restoration.

A source material license is renewed periodically. At any other time, a licensee may request an amendment to their license to reflect an authorized change in site operation or management procedures. An ISL mining license may be terminated at the request of a licensee or because the license has expired.

1. INTRODUCTION

The United States Congress has directed the Nuclear Regulatory Commission (NRC) to regulate uranium recovery operations via issuance of source material licenses. The NRC administers regulations issued under the Atomic Energy Act of 1954, as amended, and Titles I and II of the Uranium Mill Tailings Radiation Control Act of 1978, as amended. Originally, open pit and underground mining were the preferred method of uranium recovery. However, as economics of the product changed, less expensive in situ leach (ISL) uranium recovery has become the only active uranium recovery process in the United States.

2. LICENSING

Any party applying for a source material license (license) must submit a license application in conjunction with an Environmental Report (ER). The ER must characterize the proposed ISL mine site and the associated processing facility, as well as the operations plan. The ER must also provide
plans for environmental protection and monitoring and a radiation safety program. Additionally, the application must include plans for post-operation ground-water restoration, land reclamation, and plant decommissioning. A license applicant is required to submit this information to demonstrate knowledge of the proposed ISL site and to ensure that operation of the proposed facility will not cause harm to human health or the environment.

The information described above provides the basis for the NRC’s preparation of an Environmental Assessment (EA) and Safety Evaluation Report (SER). The EA comprises an evaluation of the potential environmental impacts of a proposed ISL facility. In compliance with the National Environmental Policy Act, if the conclusion of the EA is that there will be no impact to the environment, the NRC publishes a Finding of No Significant Impact. Publication in the Federal Register provides notice to the public, and solicits comments with regards to the proposed project. All project documentation is available for review in the NRC Public Document Room. If the NRC finds a significant impact may occur as a result of proposed facility operations, a more detailed document, an Environmental Impact Statement, is required. This report is also available to the public.

The SER is an evaluation of the prospective workforce and site occupant radiation safety at an ISL facility. This evaluation is based on proposed site operations and the predicted sources of accompanying radiation hazards. The SER includes a description of the applicant’s radiation safety program to limit radiation exposure from gamma emitters as well as ingestion and inhalation of airborne radionuclides. Methods for reducing radiation exposure may include reducing occupancy time in high-radiation areas, implementing general housekeeping procedures to prevent surface contamination, and minimizing airborne uranium and radon. The SER also includes a description of an in-plant monitoring program to ensure that the workers’ environment is adequately characterized.

An ISL mine operator is required to provide proof of financial surety arrangements prior to commencement of operations. This requirement assures that there are sufficient funds available for decontamination, decommissioning, and reclamation of the facility in the event that the operator is incapable of performing these tasks.

Upon meeting all regulatory requirements necessary to ensure an environmentally successful and radiologically safe ISL operation, an applicant receives a source material license authorizing operation of their proposed facility.

3. SITE CHARACTERIZATION

Characteristics reviewed to assess the suitability of a site for ISL mining, as well as to determine the potential impact of ISL mining upon human health and the environment, include geography and demographics, land use, geology, hydrology, local meteorology, and other environmental factors.

Characterization of the subsurface geology and hydrologic regime commonly begins during uranium ore body exploration. Geophysical well logs provide data necessary for developing geologic cross sections. Aquifer testing is usually conducted to provide the type of data required for a more detailed evaluation of hydrologic properties. This information ensures that the ore body is suitable
for ISL mining, defines the mineralized aquifer, and locates both the aquitards and aquifers immediately overlying and underlying the ore body aquifer.

Prior to any mining activities, the licensee collects ground-water quality data to determine baseline water quality in the mining area. Baseline water quality values are statistically derived concentrations of selected ground-water constituents referred to as indicators that are used as part of the monitoring program to determine mining solution movement. Ground-water monitoring indicating that a particular indicator concentration is within an expected range indicates operational control of ISL lixiviant. Loss of lixiviant control is verified when a predetermined indicator concentration is met or exceeded; if this occurs, a licensee is required to implement corrective actions to recover control of the well-field fluids.

Baseline ground-water quality values are also used to define ground-water restoration standards. The primary goal of a ground-water restoration program is to return ground-water quality to pre-mining, baseline conditions, or same class of use.

The NRC requires a license applicant to evaluate the potential for airborne effluent releases and to evaluate the potential impact to the public or environment as a consequence of these releases. All facility components that may act as a source for airborne effluents must be identified, and annual cumulative doses to receptors calculated using MILDOS, a computer program developed by the NRC for estimating airborne releases from the various types of contaminant sources. At an ISL site, these sources commonly include such elements as radon gas vents in well fields and well-head houses, processing tanks which vent radon gas, and yellowcake dryer vent stacks that emit particulates. The estimated annual doses are used to determine if the ISL facility can operate within Federal standards for radiological effluent releases.

4. FACILITY DESIGN AND OPERATION

The NRC requires that source material license applicants submit facility design and construction quality control criteria to ensure water-borne and airborne radioactive material will be adequately controlled at all times. The applicant includes information for the injection and recovery well-field patterns, well-field and process pipelines, other elements of the processing plant, the yellowcake drying and handling area, and the waste-water treatment and storage facilities.

In addition to describing the design and quality control criteria for the proposed ISL facility, a prospective licensee prepares written Standard Operating Procedures for all work involving radioactive material. These procedures are required to be available at all times and for all employees. Similar procedures are also prepared for all equipment and programs developed for monitoring or protecting the environment.

5. RECLAMATION

The reclamation plan for an ISL facility must describe methods for reclaiming a site so that there are no long-term, detrimental impacts to the environment as a result of mining and associated activities. Additionally, it must demonstrate that there will be minimal risks of exposure to radiation after the project is completed. As previously discussed, the licensee must provide a surety sufficient to cover all reclamation procedures.
All buildings, building foundations, and equipment must be decontaminated prior to release to an unrestricted area. Alternatively, contaminated equipment may be transferred to another holder of a source material license or disposed of as byproduct waste material. Injection, recovery, and monitoring wells must be cut off below the surface, and appropriately abandoned to prevent cross contamination between aquifers.

Radioactive wastes from in situ operations comprise well-field piping, contaminated soil, decommissioned equipment, and aquifer bleed and restoration water. Contaminated solids are disposed of as waste byproduct material. Currently, tailings piles at conventional mills serve as receptors. Waste water is usually transferred to storage ponds for evaporation. Alternatively, a licensee may hold a permit granted by the U.S. Environmental Protection Agency for deep well injection.

The final step in site reclamation includes restoring a mine site to pre-mining conditions. Facility building sites and areas such as evaporation ponds are commonly returned to original topographic contours. Following this, top soil is replaced and revegetation is implemented.

6. LICENSE RENEWALS, AMENDMENTS, AND SURETY UPDATES

Periodically, all NRC source material licenses require renewal. In situ mining licenses are in effect for 5 to 7 years. Prior to expiration of the license, each licensee is required to submit a new Environmental Report and a request for renewal. The request must summarize the operating history and performance of the in situ mine and address any significant changes to the site or the facility which have occurred during the previous licensing period. License renewal is typically routine, but requires justification that the facility continues to protect public health and safety and that radiation exposures are as low as reasonably achievable.

In addition to renewals, licensees routinely request license amendments. Typical amendment requests address operating problems, establish new procedures, or request release from certain regulatory requirements which the applicant finds do not apply to their operations or facility. Simple license amendments may include a change of mailing address or a change in the licensee’s corporate organization. Complex amendments may include design changes for the processing plant or evaporation ponds, new environmental monitoring plans, or instituting new standards for radiation and environmental protection.

Finally, the NRC requires a licensee to review and update the financial surety covering projected costs of reclamation, on an annual basis.

7. INSPECTIONS

NRC periodically performs onsite inspections during construction, operation, and reclamation of in situ uranium mines. Inspectors review the licensee’s operations in light of applicable government regulations, license conditions, and commitments made in a license application. Areas of review include management oversight, training, waste disposal, environmental protection and radiation safety, and emergency preparedness. NRC requires each licensee to periodically submit reports providing data on worker exposures and environmental monitoring. NRC also conducts special nonroutine inspections. These are usually in response to specific allegations of problems or
wrongdoing. In addition, NRC responds to incidents or accidents which may result in unanticipated or unauthorized releases of radioactive materials to the environment.

Violations of license requirements require a response from the licensee. The licensee must address the cause of the violation and steps taken to correct the situation. In serious cases, licensees are subject to escalated enforcement and penalties for noncompliance.
POTENTIALITIES OF IN-SITU LEACHING OF URANIUM AND COPPER ORE MATERIAL IN WEST CENTRAL SINAI, EGYPT

M.A. MAHDY, N.T. EL HAZEK, T.E. AMER
Nuclear Materials Authority,
Cairo, Egypt

Abstract

A low grade uranium-copper ore material is recently recorded in the Paleozoic sedimentary sequence in west central Sinai, Egypt. The host rocks composed mainly of siltstone, clay, sand and gravel. A number of secondary uranium and copper minerals have been reported including mainly uranophane, carnotite, metatyuyamunite and meta-autunite beside atacamite, para-aticamite and langite. Series of lab percolation leaching experiments in both vertical and horizontal columns have been performed using acid and alkaline leaching reagents. Due to the fact that the ore material is situated above the water table, the in-situ leaching technique is doubtful. However, other natural leaching methods especially heap and in place leaching could be proposed.

1. INTRODUCTION

The southern part of Sinai Peninsula is mostly covered by the basement complex of the Arabo-Nubian massif. A Paleozoic sedimentary sequence of varying lithology and thickness outcrops on both the eastern and western sides of the massif. Uranium-copper mineralization has been discovered in the western sedimentary succession at Wadi Nasib area. The latter belongs to Um Bogma environs famous for its manganese-iron ores. Wadi Nasib area is situated at 40 km east of Aba Zeneima town along the eastern coast of the Gulf of Suez. It extends between long. 33°15’ and 33°25’ E and between lat. 28°95’ and 29°05’ N. Two localities in this area are most promising for investigating potential application of in-situ leaching for both uranium and copper values.

2. GEOLOGICAL SETTING

From the topographic point of view, west central Sinai area can be considered as a plateau of an average height of about 500 m above sea level. The general slope of the plateau is to the north as well as to the east and is dissected by several drainage wadies. The interesting west central Sinai area whose geological setting has been described in detail by [1], is covered by a Paleozoic sedimentary succession that overlies a peneplained surface of the basement complex (Fig. 1). The latter is composed of crystalline igneous and metamorphic rocks. On the other hand, the Paleozoic rocks are separated from the Pre-Cenomanian rocks by basaltic sheet of probable Triassic age [2].

According to Barron [3] and Ball [4] the Paleozoic sediments have been divided into three series namely:

- the Lower Un-fossiliferous Sandstone Scale Series of about 150 m thickness,
- the Middle Dolomitic Limestone Series of about 40 m thickness,
- the Upper Sandstone Series which varies greatly in thickness from 60 to 200 m.
FIG. 1. Geological map of the west central Sinai area (after El Aassy et al., 1986).
FIG. 2. Structural map of the west central Sinai area (after Hussein et al., 1992).
Recent works have assigned different ages for the three mentioned series including [2], [5], [6], [7] and [8]. According to most of these authors the Paleozoic sediments of west central Sinai could be divided into two major divisions:

- Early Paleozoic sediments (Cambro-Ordovician): these are equivalent to the Lower Sandstone Series and could be further subdivided to 4 rock formations, namely Sarabit El Khadim, Abu Hamata, Nasib and Adebia Formations.
- Late Paleozoic sediments (Carboniferous): these are further subdivided into two formations: namely Um Bogma (Tournaisian - Middle Visean) Formation equivalent to the Middle Dolomitic Series and Abu Thor (Upper Visean) Formation equivalent the Upper Sandstone Series.

The interesting Um Bogma Formation does not exceed 40 m in thickness and shows important lateral variation in thickness as well as in facies. Thus it varies in thickness from 40 m in the north to 21-10 m in the central part to only 6 m in the south. Regarding the facies, it changes from carbonate in the north passing into silty and carbonate in the central part to sandy in the south.

Structurally, west central Sinai area is strongly tectonized and often belongs to the Gulf of Suez tectonic province. Faults play the main role in tectonic deformation while joints and folds are common structural forms. Indeed, the main structural trends in Um Bogma are the N-S, ENE-WSW, NE-SW, and NNW-SSE as noted by [9] and [10]. The authors have concluded that most of the radioactive anomalies are located in sediments present in graben structure (Fig. 2). The N-S faults represent the main trend followed by that of the ENE-WSW which is, however, older than the N-S faults. Folds in the area are mainly affected by the faulting structures and are mainly represented by tilted fault blocks and drag folds. Syn-depositional open and wide folds are also noticed. Both faults and folds have played an important role in governing the mode of occurrence of the mineral deposits in the area.

3. LITHOSTRATIGRAPHY OF UM BOGMA FORMATION

Interest in Um Bogma Formation is due to the presence of manganese-iron ores including pyrite, galena and barite [11], secondary copper mineralizations [12], gibbsite [13] and recently uranium mineralization [14] and [15].

The composite lithostratigraphic section of Um Bogma Formation is shown in Fig. 3. It has indeed been subdivided into 3 or 4 members by several authors. However, the classification of Mart and Sass [16] of Um Bogma Formation into 4 members would be used herein:

- Upper Dolomitic Member which is not promising from the radioactivity point of view:
- Marly Dolomitic and Silt Member;
- Silty Shale Member;
- Dolomite and Ore Member.
On the bases of lithological, mineralogical and geochemical aspects, Afifi [1] has subdivided Um Bogma Formation into 3 member (Fig. 4) as follows from base to top:

- Upper Clay Member (equivalent to the base of the intercalations);
- Sandy Dolomitic Member;
- Lower Clay Member.


4. HYDROGEOLOGICAL FEATURES

The water resources in Sinai Peninsula are depending essentially upon rainfall and springs. Indeed, Sinai Peninsula belongs to the area lying within the arid belt which included the northern part of Africa and extends to the southwest of Asia [17]. So, the average annual precipitation in west central Sinai is low (mostly around 25 mm/y). Heavy rainfall occurs in winter for very short periods and results in torrential floods in the wadis, whereas in summer, the climate is hot and temperature reaches 40°C.

The important water-bearing rock formations in Sinai Peninsula include the basement complex, the Nubian sandstone, the limestones of Mesozoic and Tertiary ages and the Quaternary sandstones, alluvial plains and sand dunes [17]. However, the west central Sinai area, water-bearing rock formations are only restricted to the Nubian sandstone of Paleozoic age (Upper and Lower Sandstone Series) as well as to the Quaternary alluvial plains.

The Nubian sandstone represents a prolific water bearing formation. It is expressed on the surface overlying the basement complex rocks and is concealed below the thick sedimentary section of sandstone and shales which form an impervious surface. The lower surface of the aquiferous sequence is not known, while its upper boundary is well marked by the calcareous sequence of marls and limestones, base of which has aquiclude properties. Since the lower boundary is not known, the exact thickness of the aquifer is unknown too, but it is at least 200 m. In most of the area this sequence forms a confined aquifer system recharged from local precipitation along with subsurface flow. The latter probably comes from the far northeast e.g. Lebanon mountains [17]. The Nubian sandstone water is fresh and is under artesian pressure 9170 m above sea level.

The phreatic surface of the Nubian sandstone aquifer in west central Sinai is mostly found at about 600-800 m below the ground level. However, the uncontaminated water in the Nubian sandstone at Ras Sudr, Asl and Lagia was found at depths of 1642 - 2140 m [18]. All the Nubian sandstone waters are characterized by the general ionic sequence of Na > Ca > Mg, and Cl > SO₄ > HCO₃. The waters are extremely saline with a TDS up to 249, 600 m/l. No relation between increasing depth and changes of salinity has been discerned.

The alluvion deposits (Quaternary) occupy the alluvial plains which are parallel to the Gulf of Suez and the Gulf of Aqaba. This aquifer is composed essentially of gravels which are intercalated with clay lenses. The water of this aquifer appears in the form of normal water table condition (fresh zone) floating on the sea water at a depth of about 10 m [17]. The salinity of water of this aquifer ranges from 2 to 4 gm/l. It is, in general, fresher towards the east but salinity increases with pumping.

Hamman Faraun is one of the water bearing alluvial plains. It is represented by several pockets of an alluvial delta which is met along the Gulf of Suez. These are represented at Ras Malaab (outlet of Wadi Gharandal), Aby Zeneima (outlet of Wadi Baba and Wadi Sudri) and Feran (outlet of Wadi feran). The springs at Hamman Faraun are probably recharged from the subsurface Nubian sandstone through fault line. The water which is under artesian pressure is normally hot (72°C) and relatively saline. Hamman Faraun waters are characterized by high content of iron. Iron minerals impregnate the dolomitized fault face below which the hot waters of Hamman Faraun exist and are a characteristic sign of these waters.
In general, the west central Sinai water formations have equivalent ionic ratios similar to those of the hot brines found in the three Deeps of the Red Sea. Indeed, the thermal regime and mineralization phenomena in west central Sinai area which are characterized by high gradient foci, show a resemblance to those of the Deeps. According to Issar et. al. [18], the high gradient foci of the thermal regime are located in the immediate proximity of major faults, Fe-mineralization, dolomitization and concentration of heavy metals on the eastern shore of the Gulf of Suez.

4. URANIUM AND COPPER MINERALIZATION

Most of the significant uranium and copper mineralization beside the manganese-iron ores are linked to the Middle Dolomitic Limestone Series of the Paleozoic sediments (Um Bogma Formation). This formation includes silt, gravel, shale and sandy facies. According to Mahdy [19], the average mineralogical composition of the uraniferous siltstone include kaolinite 46%, montmorillonite 10%, quartz 10%, iron oxides 25%, and carbonates and/or gypsum 9%. Dabbour and Mahdy [15] identified the radioactive mineral assemblage in west central Sinai and classified them into two types. The first is the secondary uranium minerals which include 11 uranium minerals (Table 1) while the second type is the detrital radioactive minerals (monazite, uraniferous, xenotime and zircon). Other uraniferous minerals (jarosite and atacamite) have also been reported.

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<th>Mineral</th>
<th>Chemical Formula</th>
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<td>liebigite</td>
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<td>K-zippeite</td>
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<td>Carnotite</td>
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</tr>
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<td>Rb-carnotite</td>
<td>Rb$_2$(UO$_2$)$_2$(VO$_4$)$_2$. 1-3H$_2$O</td>
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<td>meta-autunite</td>
<td>Ca(UO$_2$)(PO$_4$)$_2$. 8H$_2$O</td>
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<td>hydrogen-autunite</td>
<td>H$_2$(UO$_2$)$_2$(PO$_4$)$_2$. 8H$_2$O</td>
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<tr>
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<td>meta-torbernite</td>
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<td>Cu(UO$_2$) (AsO$_4$)$_2$. 8H$_2$O</td>
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<td>meta-tyuyamunite</td>
<td>Ca(UO$_2$)$_2$(VO$_4$)$_2$. 3-5H$_2$O</td>
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<tr>
<td></td>
<td>jarosite**</td>
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<td>atacamite**</td>
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<td>uranophane</td>
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<td></td>
<td>K-zippeite</td>
<td>K(UO$_2$)(SO$_4$)(OH)$_2$.4H$_2$O</td>
</tr>
<tr>
<td></td>
<td>monazite*</td>
<td>(Ce, La, Y, Th) PO$_4$</td>
</tr>
<tr>
<td></td>
<td>zircon*</td>
<td>Zr SiO$_4$</td>
</tr>
<tr>
<td></td>
<td>atacamite</td>
<td>Cu$_2$(OH)$_3$.Cl</td>
</tr>
<tr>
<td>Ramlet</td>
<td>uvanite</td>
<td>U$_2$V$_6$ O$_2$1. 15 H$_2$O</td>
</tr>
<tr>
<td>Hemyir</td>
<td>xenotime*</td>
<td>YPO$_4$</td>
</tr>
</tbody>
</table>

* Detrital minerals.
** Uranium-bearing minerals.
Table (2): Secondary copper minerals identified in west central Sinai area (after Hilmy and Mohsen, 1965).

<table>
<thead>
<tr>
<th>Locality</th>
<th>Mineral</th>
<th>Chemical Formula</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wadi Nasib</td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$ Cl</td>
</tr>
<tr>
<td>Wadi Allouga</td>
<td>Chalcantite</td>
<td>CuSO$_4$. 5 H$_2$O</td>
</tr>
<tr>
<td></td>
<td>Antlerite</td>
<td>Cu$_3$(SO$_4$)(OH)$_4$</td>
</tr>
<tr>
<td></td>
<td>Buttgenbachite</td>
<td>Cu$_{19}$(NO$_3$)$<em>2$(OH)$</em>{32}$Cl$_4$.3H$_2$O.</td>
</tr>
<tr>
<td></td>
<td>Cuprite</td>
<td>Cu$_2$O.</td>
</tr>
<tr>
<td></td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$ Cl</td>
</tr>
<tr>
<td>Wadi Abu Thor</td>
<td>Chrysocolla</td>
<td>Cu$_2$H$_2$Si$_2$O$_5$(OH)$_4$</td>
</tr>
<tr>
<td></td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$. Cl</td>
</tr>
<tr>
<td>Gebel Um Bogma</td>
<td>Malachite</td>
<td>Cu$_2$(CO$_3$)(OH)$_2$</td>
</tr>
<tr>
<td></td>
<td>Connellite</td>
<td>Cu$_{19}$(SO$<em>4$)(OH)$</em>{32}$Cl$_4$.3H$_2$O.</td>
</tr>
<tr>
<td></td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$. Cl</td>
</tr>
<tr>
<td>Wadi El Sih</td>
<td>Crednerite</td>
<td>Cu Mn$_2$O$_4$</td>
</tr>
<tr>
<td></td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$. Cl</td>
</tr>
<tr>
<td>Gebel Sid El Banat</td>
<td>Spangolite</td>
<td>Cu$_6$Al(SO$<em>4$)(OH)$</em>{12}$Cl.3H$_2$O</td>
</tr>
<tr>
<td></td>
<td>Langite</td>
<td>Cu$_4$(SO$_4$)(OH)$_6$.2H$_2$O.</td>
</tr>
<tr>
<td></td>
<td>Shattuckite</td>
<td>Cu$_5$(SiO$_3$)$_4$(OH)$_2$</td>
</tr>
<tr>
<td>Wadi Abu Hamata</td>
<td>Crednerite</td>
<td>Cu Mn$_2$O$_4$</td>
</tr>
<tr>
<td></td>
<td>Turquoise</td>
<td>Cu Al$_6$(PO$_4$)$_4$(OH)$_8$.4H$_2$O</td>
</tr>
<tr>
<td></td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$. Cl</td>
</tr>
<tr>
<td>Wadi Um Rinna</td>
<td>Atacamite</td>
<td>Cu$_2$(OH)$_3$. Cl</td>
</tr>
</tbody>
</table>

Table (3): Iron-manganese ore minerals identified in west central Sinai area (after El Shazly and Saleeb, 1959).

<table>
<thead>
<tr>
<th>Locality</th>
<th>Mineral</th>
<th>Chemical Formula</th>
</tr>
</thead>
<tbody>
<tr>
<td>Olikat, Marahil, Um Sakran, Um Bogma, Um Sayalat, Wadi Nasib, Gebel Nasib, Higazia.</td>
<td>Fe</td>
<td>FeO (OH)</td>
</tr>
<tr>
<td></td>
<td>Hematite</td>
<td>$\alpha$ Fe$_2$O$_3$</td>
</tr>
<tr>
<td></td>
<td>Geothite</td>
<td></td>
</tr>
<tr>
<td>Mn</td>
<td>Pyrolusite</td>
<td>MnO$_2$</td>
</tr>
<tr>
<td></td>
<td>Manganite</td>
<td>MnO(OH)</td>
</tr>
<tr>
<td></td>
<td>Psilomelane</td>
<td>Ba(Mn$^{2+}$ Mn$^{4+}$)$_{16}$(OH)$_4$</td>
</tr>
<tr>
<td></td>
<td>Cryptomelane</td>
<td>K (Mn$^{2+}$, Mn$^{4+}$)$<em>8$O$</em>{16}$</td>
</tr>
</tbody>
</table>
Hilmy and Mohsen [12] have reported 13 secondary copper minerals in the different localities of east central Sinai area (Table 2). On the other hand, El Shazly and Saleeb [20] have listed the mangenese-iron minerals which are widely spread in the study area (Table 3).

5. **LEACHING STUDIES**

To investigate the potentialities of in-situ leaching of uranium and copper in west central Sinai ore material, both agitation and percolation techniques were performed. A sample from Allouga locality assaying 1250 ppm uranium and 2105 ppm copper was chosen whose average major chemical composition is given in Table 4. The trace elements analysis of the selected sample is as follows (in ppm):

- As 212,
- Ba 2600,
- Cu 2105,
- Mn 1200,
- Nb 60,
- Ni 42,
- Pb 320,
- Rb 30,
- Sr 211,
- Ti 3000,
- V 503,
- Y 195,
- Zn 97,
- Zr 102,
- U 1250,
- Se 23.

Agitation leaching tests were performed in 250 ml rounded-bottom flasks. Leaching experiments were done using 10 ml reagent solution of known concentration was added to a sample of 50 g ground ore. On the other hand, percolation leaching tests have been also performed in both vertical and horizontal position. For this purpose, the ore material was coarsely crushed (-10 mesh) to be easily packed in the glass column. To overcome the impractical flow rate of reagents, the uraniferous silty material was sometimes mixed with sand.

Table (4): Average chemical composition of the uraniferous siltstone from west central Sinai area.

<table>
<thead>
<tr>
<th>Oxides</th>
<th>Wt. %</th>
</tr>
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<tbody>
<tr>
<td>SiO₂</td>
<td>53.30</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>1.80</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>25.80</td>
</tr>
<tr>
<td>FeO</td>
<td>0.70</td>
</tr>
<tr>
<td>MnO</td>
<td>0.15</td>
</tr>
<tr>
<td>MgO</td>
<td>0.93</td>
</tr>
<tr>
<td>CaO</td>
<td>1.00</td>
</tr>
<tr>
<td>Na₂O</td>
<td>1.20</td>
</tr>
<tr>
<td>K₂O</td>
<td>1.06</td>
</tr>
<tr>
<td>P₂O₅</td>
<td>0.14</td>
</tr>
<tr>
<td>TiO₂</td>
<td>0.55</td>
</tr>
<tr>
<td>Organic Matter</td>
<td>2.10</td>
</tr>
<tr>
<td>H₂O⁻</td>
<td>2.00</td>
</tr>
<tr>
<td>H₂O⁺</td>
<td>5.10</td>
</tr>
<tr>
<td>CO₂</td>
<td>1.60</td>
</tr>
<tr>
<td>SO₄⁻²⁻</td>
<td>2.51</td>
</tr>
<tr>
<td>Cl⁻</td>
<td>0.64</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>100.58</td>
</tr>
</tbody>
</table>
5.1 Results of agitation leaching

All the mineral acids can dissolve both uranium and copper under reasonable conditions, however with different efficiencies [21]. It is included that the leaching efficiency of copper is much greater than that of uranium when using nitric and sulphuric acids. However, the contrary is true when using hydrochloric acid, where almost complete uranium leaching can be achieved by using 300 g/l HCl compared to only 65% and 55% uranium leaching when using the same acid concentration of HNO₃ and H₂SO₄ respectively. These results were obtained at 1:2 solid-liquid ration, 90°C leaching temperature and for an agitation time of 4 hours using -100 mesh size ore material. Under these conditions, hydrochloric acid solution of 300 g/l can leach only 45% of the copper content, while complete copper leaching can be performed by using 300 g/l HNO₃ or 200 g/l H₂SO₄.

In this work, it was decided to study in detail some relevant leaching factors. Thus, the effect of H₂SO₄ concentration was studied between 100 and 200 g/l while all other variables were kept constant (-100 mesh, 1:2 S/L ration, 72 hours agitation time under room temperature). The obtained data of leaching efficiency of both uranium and copper are shown in Figs. 5 and 6 from which it is noticed that uranium dissolution increases with the increase of H₂SO₄ concentration. Thus at 100 g/l H₂SO₄, uranium leaching efficiency attained 69.2% while at 200 g/l H₂SO₄, the leaching efficiency was almost complete (99.6%). Under these conditions, Cu leaching efficiency was found to be 88.6% using 100 g/l acid. It is interesting to mention that copper is more easily leached than uranium under similar experimental conditions. These results are generally in agreement with those of Mahdy et al. [21] obtained at 90°C for 4 hours.

Mahdy et al. [21] found that time has an important effect upon leaching efficiencies of both uranium and copper where they have concluded that complete uranium and copper leaching could be achieved under the following two sets of conditions:

a) using 100 g/l H₂SO₄ at 90°C, copper could be completely leached after 6-8 hours, while only 80% of uranium could be leached after 18 hours.

b) using 200 g/l H₂SO₄ at 90°C, complete leaching of copper could be obtained after 4 hours, while uranium complete leaching requires 24 hours.

![FIG. 5. Effect of acid concentration on leaching efficiencies of uranium and copper.](image1)

![FIG. 6. Effect of acid concentration on leaching efficiencies of uranium and copper.](image2)
In the present work, leaching at room temperature has indicated two alternatives for uranium and copper leaching, namely:

a) using 100 g/l \( \text{H}_2\text{SO}_4 \) gives complete copper extraction after 72 hours, however, only 70% of the uranium content was achieved.

b) using 200 g/l \( \text{H}_2\text{SO}_4 \) it was possible to extract both uranium and copper completely after 48 hours agitation time.

Finally, it is noteworthy to mention that Mahdy [19] has found that temperature has a great effect on the extraction efficiencies of uranium and copper in either acidic or alkaline leaching processes. Also, neither grinding of the ore material nor addition of an oxidant has any influence on uranium or copper leaching efficiencies of west central Sinai [19].

In this paper, both sodium carbonate and bicarbonate were used as leaching reagents with concentrations ranging from 50-150 g/l. The leaching conditions were 1:2 solid/liquid ratio at 60°C leaching temperature and for 24 hours agitation time while the ore material was crushed to -100 mesh size. Both of these reagents can dissolve uranium selectively under reasonable conditions with different efficiency where it is much greater when using \( \text{Na}_2\text{CO}_3 \) than when using \( \text{NaHCO}_3 \) solution (Figs. 7 and 8). This may be explained in the light of the fact that at relatively high temperature, \( \text{NaHCO}_3 \) would be decomposed to \( \text{Na}_2\text{CO}_3 \) with consequent dilution of the alkali reagent during the leaching process, viz,

\[
2\text{ Na HCO}_3 \xrightarrow{\Delta} \text{ Na}_2\text{CO}_3 + \text{H}_2\text{O} + \text{CO}_2
\]

On the other hand, agitation time is effective up on uranium extraction efficiency when applying alkaline leaching technique (Fig. 8).

5.2 Results of vertical percolation leaching

Mahdy [19] has performed some percolation leaching tests upon west central Sinai uraniferous siltstone in the vertical position to prospect the potentialities of the natural leaching technique. He stated that uranium leaching efficiency increased from about 5% after 12 days to
FIG. 9. Extraction efficiency leaching curves of uranium and copper during acid percolation leaching.
about 30% after 42 days by using 10 g/l HCl. However, sharp increase in uranium leaching efficiency has been achieved in the next 30 days from 30% to almost complete leaching. On the other hand, copper leaching was complete in about 30 days at only 15% uranium leaching efficiency. By percolation leaching using 1 g/l Na$_2$CO$_3$ for two months through a packed west central Sinai uraniferous siltstone sample, uranium leaching efficiency did not exceed 10-12% [19].

In this work, some acid percolation leaching studies were performed using H$_2$SO$_4$, however, for the purpose of wetting, washing water was firstly allowed to pass through the packed ore material at a flow rate ranging from 2 to 5 ml/min. In case of the study ore, the obtained unrestricted water flow rate was only about 2.5 ml/h due to excess fineness of the ore material. When acid solution was passed, the flow rate increased to about 3.2 ml/h, however, the convenient average of the flow rate should be 10 ml/min [22].

Almost complete leachability of uranium and copper could be obtained by applying acidic percolation leaching upon west central Sinai uraniferous siltstone. Thus by using the vertical position (column diameter 3 cm, ore height 35 cm, ore weight 300 g), 10 g/l H$_2$SO$_4$ solution was able to achieve complete uranium dissolution after about 71 days (27 bed volumes) while the copper content was leached after only about 53 days (20 bed volumes, Fig. 9). Using lesser acid concentration e.g. 1 and 5 g/l H$_2$SO$_4$, it was possible to dissolve only 9.1% and 46.9% of the uranium contents while 1% and 62.8% of the copper content was dissolved respectively. The poor leaching efficiencies of uranium and copper at the latter acid concentration might be due to the acid consumptions in dissolving iron oxides (hematite and goethite) which averages up to 25% of the chemical composition of the ore material beside dolomite which averages 4%.

On the other hand, alkaline percolation leaching using 1, 10 and 15 gl Na$_2$CO$_3$ or NaHCO$_3$ for 3 months resulted in only 25 to 29 leaching percent for uranium while copper did not respond to leaching. It has to be mentioned in this regard that the flow rate during these experiments did not exceed a range of only 0.3 - 1 ml/hr.

5.3 Results of horizontal percolation leaching

Due to the impractical extremely low flow rate of the leaching reagents during previous percolation experiments, it was decided to switch from vertical to horizontal position. Two series of acidic and alkaline percolation leaching experiments were thus performed using 5 and 10 g/l H$_2$SO$_4$ and 10 g/l of only (NH$_4$) HCO$_3$ or in mixture with (NH$_4$)$_2$ CO$_3$. It was also tried to further improve the flow rate by mixing the uraniferous siltstone with sand or else by adding some modifying reagents to the leach liquor.

Acid percolation leaching experiments using the horizontal columns proved to be relatively efficient. Thus, it was possible to dissolve both uranium and copper after passing 7.2 litres (13 bed volumes) of the leach solution (5 g/l H$_2$SO$_4$) for 53 days (Fig. 10-a). When using 10 g/l H$_2$SO$_4$ as the leach solution, both elements were almost leached (Fig. 10-b) after 6 litres (10 bed volumes) after 44 days. However, in both experiments the flow rate was only 5.6 ml/hr. The experimental conditions include 800 g of -10 mesh size ore material packed in 3 cm diameter column to a length of 60 and 78 cm in the 5 and 10 g/l H$_2$SO$_4$ experiments respectively. It is worthy to mention that Grant et al. [22] reported that while working with arkose ore material packed in a horizontal column they have been able to realize effective uranium leaching using a flow rate of about 600 ml/hr. Therefore, it might be mentioned that, the recommended tests of about 2 kg ore/ml length (IAEA-TEC-DOC-492) and which would give acceptable flow rates and reliable results are applicable in
sand information. In other words the obtained extremely impractical low flow rate is due to the fineness of west central Sinai siltstone. However, it might not be a matter of only fineness but rather the ration of ore weight to its length in the column that would control the flow rate. This might be referred to as the packing degree of the ore. Also, bench scale tests must be taken with care as the packing of ore could be greatly different than that in the site.

Trials to improve flow rate of percolation of the leach liquor through the study siltstone involved performance of the following two series of experiments:

a) Adding of some modifying chemical reagents that are used for site restoration to the acid leach liquor involved AlCl₃ or NaHSO₄ (3 g/l). The former might lead to alum formation with its known flocculating properties. Results obtained were still not encouraging where AlCl₃ addition increased the flow rate from 5.6 to only 7.5 ml/hr and that of NaHSO₄ increased the flow rate to 11.2 ml/hr.

b) Mixing sand with the uraniferous siltstone in amounts of 25% and 50% resulted in improving the flow rate from 5.6 to only 11.8 and 16.7 ml/hr respectively. However, embedding sand rings (3 cm) along the ore bed length (20 cm) has resulted in a remarkable improvement in the flow rate where it reached the desired flow rate of sandy ore materials (600 ml/hr).

On the other hand, alkaline percolation leaching experiments using horizontal columns have been performed using (NH₄)₂CO₃ / NH₄HCO₃. Thus using the same experimental condition but with 10 g/l NH₄HCO₃ and using sand/ore ratio 1:1, almost complete leaching of both uranium and copper, was achieved (Fig. 11-a) after passing 18.6 litres (31 bed volumes) in 46 days. The attained flow rate was similar to that of the acid leaching comparable experiment. Using 3 g/l (NH₄)₂CO₃ with 7 g/l NH₄HCO₃ as reagent improved slightly the leaching results (Fig. 11-b).

Addition of NaHSO₄ as a modifying reagent to the alkaline leach liquor using NH₄HCO₃ improved the flow rate from 16.7 to only about 27.1 ml/hr beside improving uranium leaching efficiency. Therefore, embedding sand rings along the ore column length would be the only promising technique to improve the flow rate.
FIG. 11a. Extraction efficiency curves of uranium and copper during alkaline percolation leaching (7.0 g/l $\text{NH}_4\text{HCO}_3$, 3.0 g/l $(\text{NH}_4)^2\text{CO}_3$).

FIG. 11b. Extraction efficiency curves of uranium and copper during alkaline percolation leaching (10.0 g/l $\text{NH}_4\text{HCO}_3$).

FIG. 12. Tentative flowsheet for extraction of uranium and copper from west central Sinai ore material (after Ritcy, 1991).
6. CONCLUSION

The foregoing results have indicated that uranium and copper could be co-dissolved in either acid or alkali leaching liquor. Furthermore, manganese leaching as a valuable element, was tested in the spent ore material of same experiments and was found to have been leached to the extent of about 88% and 100% when using 5 and 10 g/l H$_2$CO$_4$ respectively.

A tentative flow sheet for the recovery of uranium, copper and manganese from west central Sinai ore material has been proposed by Ritcey [23] using 3 circuits of solvent extraction. Thus, copper is first extracted using LIX solvent followed by uranium using an amine and finally manganese could be extracted by DEPHA solvent. To ensure recovery of pure manganese, iron should be separated before extraction by either a prior reduction to the ferrous state of a prior precipitation using CaCO$_3$ (Fig. 12).

Due to the importance of As and Se as groundwater contaminants the spent ore was checked for their content after acid leaching. It was found that As dissolves to the extent of about 40% and Se to about 90%.

The achieved results indicate that almost complete leaching of the interesting elements uranium, copper and manganese is possible by either agitation or percolation techniques. Accordingly, potentiality of in-situ leaching of west central Sinai siltstone could be taken in consideration if the problem of the flow rate which is critical for in-situ leaching technique could be overcome. This question can be answered only after detailed study of several lithostratigraphic sections because of the rapid lateral and vertical changes in facies. Disposition of the ore body relative to local aquifers require detailed study of the entire Nubina sandstone aquifer.

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