URANIUM RECOVERY IN SWEDEN
HISTORY AND PERSPECTIVE

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ACKNOWLEDGEMENTS
SUMMARY

In 1945 the potential of nuclear energy for military and peaceful purposes had initiated a strong interest in Sweden to establish a national programme in the field. The Atomic Energy Committee was formed at the end of 1945 and charged the Research Institute of National Defense in January 1946 with the task to investigate possibilities of uranium recovery in Sweden. AB Atomenergi, a semi-state owned body, was formed in November 1947 to take charge of nuclear energy related research and development. The search for and the recovery of uranium from indigenous resources became a dominant objective over the first years of the company.

The report gives a broad overview of the efforts to establish a national uranium fuel supply programme in Sweden from the early days of the post-war years of the 1940s. Process and plant development, plant operations at Kvarntorp and Ranstad, and the development of the uranium supply policy are presented against a background of the nuclear power development in the country. The nuclear weapons debate and the decision to sign the Non Proliferation Treaty in 1968 are briefly reported.

Special attention has been given an account of the technical development and the various projects related to the Ranstad plant and its final restoration in the 1980s.
1 INTRODUCTION

At the end of the second world war there was a strong interest in Sweden to investigate the potential of nuclear fission for peaceful and for military uses. While its destructive capability had been clearly demonstrated in August 1945, a hope had also developed to harness its power to a source of energy supply that could reduce the dependence on imported fuels, severely felt in Sweden during the war years of isolation.

On December 17th, 1945, the Research Institute of National Defence in Sweden (FOA) sent a P.M. on the uranium issue to the then recently appointed Atomic Energy Committee. The committee charged FOA on January 4th, 1946, with the task to investigate possibilities of uranium recovery in Sweden. Contacts were established with potentially interested industries, such as the Swedish Shale Oil Co., the Boliden Mining Co., and Wargöns AB, and with various research institutions at universities in Stockholm, Uppsala, Lund and Göteborg.

The first goal was to produce pure uranium metal from indigenous raw materials. The different stages required for this purpose were developed over the next few years and comprehensively reported in 1949 (1-1).

In the national interest to develop atomic energy from uranium the Atomic Energy Committee proposed a semi-state owned body to take responsibility for applied nuclear energy related research and development in Sweden. AB Atomenergi (AE) had its statutory meeting in November, 1947. The primary objectives of the company, according to its statutes, were "to investigate and recover basic materials necessary for the exploitation of atomic energy, to construct experimental piles for the exploitation of atomic energy, later to construct piles on a larger scale for the utilization of atomic energy in the service of research and industry, and to pursue research related to aforementioned activity, as well as industrial and commercial activity" (1-2, 1-3). The search for and the recovery of uranium from indigenous resources became the dominant objective over the first years of the company.

Already at the beginning of the 1890's, uranium had been found in the ashes of "kolm", a carboniferous substance distributed in some horizons of
shale formations in Sweden. The company AB Kolm operated during the years 1909-1915 to produce radium from Swedish kolm, with little success, however (1-4). At the beginning of the 1950's a small industrial plant, located at Kvarntorp, was constructed to recover uranium from kolm that had been separated from shale. The process was modified, however, to work directly with shale.

A larger plant, at Ranstad, was in operation 1965-1969 with the use of a more sophisticated process based on the experience from Kvarntorp (1-5), and studies were performed for a still larger plant. Changes in the international uranium market and in the Swedish nuclear power programme led finally to the decision to decommission the Ranstad plant with a high degree of environmental restoration, and to rely solely on imported uranium to supply the Swedish nuclear power programme.

This report is an effort to summarize the development and the experience gained in Sweden in uranium recovery against a background of changing market and political conditions for nuclear energy. The report encompasses the period 1945 up to the present time, 1993.

REFERENCES

1-1 Ljunggren, G.

1-2 Peterson A.


1-4 Svenke, E.

1-5 Carlsson, O.
Ranstad-one of the world's largest uranium deposits. JKA No.1/80. (In Swedish).
2 EARLY DEVELOPMENT

A systematic literature search on uranium was started by FOA as one of its first activities in the field. A register was established and copies of relevant papers provided to people engaged in the work on uranium.

The experimental work was focused on analytical development work needed for uranium determinations, preparations of high purity uranium compounds, and on process development for uranium recovery and purification (2-1, 2-2).

2.1 Analytical and preparatory work

The reduction of U(VI) to U(IV) was studied early, as precipitation reactions of U(IV) were assumed to be a guide to similar reactions of Pu(IV). U(VI) was successfully reduced by passage through metallic cadmium in sulfuric and perchloric acid solutions, with checks by back titration with potassium permanganate. Hydrogen chloride solutions were found to be unsuitable for this purpose.

Fundamental research on the complex chemistry of the uranyl ion was started in 1948 at the University of Lund and continued for a long time (2-3).

Precipitation of U(IV) was carried out with i.a. phenylarsinic acid, benzene sulfinic acid, and with disodium hydrogen phosphate. The latter was found to give the most quantitative precipitation, which was later utilized in the first uranium recovery plant at Kvarntorp.

A laboratory for routine determinations of uranium was set up at FOA in the autumn of 1946 and operated to the autumn of 1948, when the major part of routine uranium analysis was transferred to AE. This enabled the FOA group to concentrate its work on impurity analysis of high purity uranium by means of spectrographic methods.

Radiometric determinations of uranium in shale and kolm were performed using GM-counters (2-4). Later, specimen preparations were worked out by evaporation and electrolysis to obtain a thin, even layer for alfa counting by proportional and scintillation counters, and for observations of the alfa energy spectrum with the aid of ionization chambers (2-5).
However, the first routine method for the determination of uranium, used at both AE and FOA, was continuous diethylether extraction from a nitric acid solution followed by spectrophotometric measurement at 3900 Å after the addition of NaOH and H₂O₂. Also gravimetric determination after precipitation with oxine and calcination to U₃O₈ was used frequently (2-6).

The analytical research at AE led in 1952 to the very selective adsorption of the uranyl sulfate complex on strongly basic anion exchangers. This was applied as a standard analytical method for low concentrations of uranium in high concentrations of iron, aluminium, and magnesium, by the separation of these elements through sorption-elution on e.g. Dowex-2 anion exchanger in sulfate form and spectrophotometric uranium determination at 3900 Å after proper adjustment of the eluate. What started as an analytical development in the laboratory, which reduced the time for the determination of uranium in various samples from several days to less than 1 hour, also exhibited the potential for large scale process application. The anion exchange procedure was thus developed into a large scale industrial process, tried out in pilot scale at the Kvarntorp plant and later introduced in full scale at Ranstad (2-7, 2-8, 2-9).

The requirement to analyze uranium in shales of various origins and compositions and in process solutions of concentrations from very low to fairly high lead to further developments. Examples are extraction with hexone (methyl isobutyl ketone), fluorimetric determinations of uranium at the nanogram level, chromatographic separations on cellulose-aluminium oxide columns, and x-ray fluorescence analysis, which drastically increased the efficiency of the analytical work (2-6).

In the early preparatory work at FOA methods were worked out for the preparation of simple uranium compounds:
- oxides, U₃O₈, UO₂
- carbide, UC₂
- fluorides, UF₆, UF₄
- tetrachloride, UC₁₄

Also a number of ways were tried for the preparation of uranium metal. This included:
- electrolysis of KUF₅ in molten salt
- electrolysis in aqueous solution with mercury
cathode
- reaction of uranium carbide with mercury
- reduction of UF₄ with silicon
- reduction of UO₂ with UC₂
- reduction of four-valent uranium halides with calcium or magnesium

The best results were obtained by reduction of NH₄UF₅ or UF₄ with Ca. A simple way to prepare the NH₄UF₅ was found to be to add UO₂ to molten NH₄F. Re-melting in vacuum of the uranium metal obtained by reduction with Ca was performed to separate the slag. The continued programme for the up-scaling of a process for the production of uranium metal was based on these findings.

2.2 Uranium recovery and purification

While the large shale deposits in Västergötland and Närke contain 0.02-0.03% U, the small bituminous, lens-formed bodies called "kolm" and found in some layers of shale, often contain 0.3-0.4% U; the ashes of kolm contain around 1%. Some of the kolm-containing layers are relatively easily accessible. It was thus early considered most practical and productive to start with kolm to obtain fair amounts of uranium for the original work (2-1).

In December 1946, 3905 kg of kolm-containing shale were delivered to the Royal Institute of Technology (KTH) in Stockholm for manual kolm separation as a first step (2-10). A sink-float process was developed at KTH to separate the kolm from shale. A concentrated solution of zinc chloride of density 1.85 served as separation medium, where the shale goes to the bottom and the kolm to the surface. All in all 18 tonnes of high-grade kolm were produced with this process at the KTH.

The Swedish Shale Oil Co. had been involved since 1946 in the problem of uranium recovery, and in 1950 it was decided to install equipment for kolm separation at its plant for shale oil recovery at Kvarntorp. Even though this area did not offer optimum kolm contents, it was deemed the only possibility to achieve reasonable costs for the kolm recovered. The new equipment would work with an aqueous medium of fine-grain magnetite sludge.

This process was operated from its start in January 1951 to January 1954 and delivered a kolm
product containing 1.5 tonne uranium. The process was then changed to the separation of shale from lime in a medium of higher density for subsequent recovery of uranium directly from shale.

The early work at FOA for the recovery of uranium from kolm started with nitric acid leaching of the ashes of kolm in steam-heated containers. The acid solution of uranium was separated from the ashes, and the uranium extracted with diethylether. Further work at FOA resulted in >97% uranium recovery by direct leaching of kolm ground to <1 mm grains, fractional separation of iron and aluminium by precipitation with calcium carbonate to yield a raw product of uranate after precipitation from ammonium carbonate solution, and final purification by ether extraction. This process was after laboratory scale development, tried on batches of 20-50 kg of kolm. A continuous ether extraction process and its equipment were also developed at FOA with a capacity of 1 kg/h. Material problems (glass, PVC, SS) and safety problems in the handling of the highly volatile and flammable ether were given special attention.

At AE the first pilot plant work on uranium recovery from kolm started in an old factory, rented at Vinterviken in the outskirts of Stockholm, where Sweden's second sulfuric acid plant had been in operation in the years 1865-1930, and where Alfred Nobel once had started his successful development of explosives. Kolm, separated at Kvarntorp and hand-picked at Billingen (Ranstad), was treated by leaching in FeCl₃ solution under stirring. The dissolved uranium was reduced by Rongalit and was precipitated as U(IV)-phosphate by phosphate ions originating from the source material. The precipitate contained about 30% U after drying. The mother liquor, where FeCl₃ was reduced to FeCl₂ in the leaching process and uranium was oxidized to U(VI), was re-oxidized by chlorine gas and re-used as leach solution. About 1.5 tonne uranium was recovered in this way. It was of great value for the supply of uranium at a time, when import of uranium was not possible, for use in metallurgical R&D work and, partly, in exchange for French metallic uranium used for the supply of the first Swedish reactor, R1, in Stockholm (2-10).
REFERENCES

2-1 Ljunggren, G.
Research works on the utilization of the uranium resource in Sweden. (In Swedish).

2-2 Peterson, A.
Uranium recovery and the preparation of high purity uranium products. (In Swedish).
Svensk Kemisk Tidskrift 79:3 (1967).

2-3 Ahrlund, S.
On the complex chemistry of the uranyl ion. I The hydrolysis of the six-valent uranium in aqueous solutions.
Acta Chemica Scandinavica 3 (1949).

2-4 Arnfelt, A.
Radiometric and chemical determinations of uranium contents in shale specimens, performed in 1955.

2-5 Haeffner, E., and Hultgren, Å.
A method for the separation of plutonium from uranium.

2-6 Arnfelt, A.
The determination of uranium. (In Swedish).
AB Atomenergi, undated PM, around 1960.

2-7 Arnfelt, A.
Laboratory investigations on the recovery of uranium from kolm and shale.
Investigation regarding a new method for the treatment of leach solutions. (In Swedish).
AB Atomenergi, Dep. of Chem. PM, 4 Nov., 1952.

2-8 Arnfelt, A.

2-9 Arnfelt, A.

2-10 Svenke, E.
Trials to utilize kolm industrially.
(In Swedish).
3 GEOLOGICAL SURVEY AND EXPLORATION

3.1 Trends in Swedish uranium exploration

As mentioned in chapter 1 the existence of uranium in the Swedish alum shales was known already at the end of the 19th century (3-1). However, first after World War II, when the possible importance of uranium as an energy raw material became apparent, the interest for uranium exploration gained momentum in Sweden.

During the end of the 1940’s and at the beginning of the 1950’s there was an intensive hunt for uranium mineralizations going on over the whole country. Due to the fact that ionizing radiation can be monitored by comparatively simple and cheap exploration instruments, many private people were engaged in the search for uranium, as well as a great number of private industrial enterprises.

The state-owned AB Atomenergi (AE), having the domestic uranium supply as one of its main objectives, established early a geological section for uranium exploration and geological investigations. The fairly small section co-operated with specialists at the universities and, above all, with the Swedish Geological Survey (SGU), which to its normal activities also added the search for radioactive mineralizations.

A very important result of the AE-SGU collaboration during the 50’s was the comprehensive investigation of the uranium resources in the alum shales in different parts of Sweden. The Billingen-Falbygden area was identified as the most promising province for uranium recovery. After detailed investigations, a suitable area for open pit mining was found at Ranstad. The alum shale resources are further discussed in section 3.3.

Among other activities during the 1950’s may be mentioned the efforts to find workable amounts of uranium and thorium in bed rock pegmatites and granites in Central Sweden, sometimes showing locally high concentrations (3-2). Also some iron ores, especially a number of skarn- and apatite ores, were found to be associated with uranium mineralization (3-3). One of the iron ores investigated by AE was at Masugnsbyn in the County of Norrbotten, where some tonnes of uranium were found concentrated in a nearby peat moor. The exploration methods used by AE included e.g. regional survey with car-borne instruments and
detailed investigation with diamond core drilling and hole logging instruments. SGU used to a great extent survey from airplanes for regional investigations.

The government funding for uranium exploration was directed to AE until 1966. Compared to the costs for investment in uranium supply and nuclear power generation facilities, financed by the government, the exploration funding was small, totally about 22 MSEK for the period 1965-74 (3-4). First after the mid 1960’s when the Swedish programme for nuclear power supply began to materialize, the government funded uranium exploration reached markedly higher levels. The uranium resources in alum shales were certainly gigantic, but their poor grade resulted in a high cost of production. The exploration of richer uranium ores in workable quantities was therefore of great importance with a forecast domestic uranium demand of 1000-1500 t/y some years ahead. From fiscal year 1967/68 the SGU was given responsibility for the government funded uranium exploration and was granted yearly funding steadily increasing from 0.6 MSEK to 17 MSEK during the period ending 1976/77 (3-5, 3-6).

After the change of government in 1976, the new nuclear policy, leaving more of the funding responsibility to the nuclear industry, was demonstrated by a cut in the SGU uranium exploration grant for 1977/78, down to 10 MSEK (3-7). In the following years the state funding was somewhat increasing, up to 14.5 MSEK for fiscal year 1980/81. With the additional support from SKBF the SGU budget for uranium exploration was totally about 25 MSEK per year until 1980/81.

After the referendum in 1980 on the use of nuclear power in Sweden, the long term domestic uranium demand became more uncertain. The government decided to finance only SGU’s regional radiometric survey (3-8) while uranium ore exploration was left to the nuclear industry to support. In the years around 1980 also the uranium market prices culminated and started to decline. SKBF continued exploration activities for some more years until 1985, when their funding was terminated (3-9).

The practical work from the mid 1970’s was concentrated to the northern part of Sweden. A great number of uranium mineralizations was found and investigated, notably at Lilljuthatten in the community of Krokom, Sågtjärn in the community of Ange, and, most important, at Pleutajokk, in the community of Arjeplog, further discussed below.
3.2 Pleutajokk

SGU found a radiometric anomaly when surveying the Pleutajokk area by air in 1967–68. After tracing finds of mineralized boulders, the source rock was hit by diamond drilling in the beginning of the 1970’s. The detailed investigation, including the driving of a short tunnel, was taken over by LKAB in 1976, with financial support from SKBF (3-10). The uranium minerals here occur in the form of fissure fillings in an originally volcanic rock formation, later strongly changed by metamorphosis into a granitic appearance. LKAB estimated a uranium reserve of about 4000 t down to a depth of 500 m.

LKAB planned for a mining of 0.6–0.7 Mt/year and a production of 450 t/y U. The anticipated production steps were in short: Underground mining, mineral dressing with radiometric sorting and rod mill grinding, fine particle leaching with sulphuric acid, uranium recovery from the leaching solution with solvent extraction, and precipitation with ammonia to produce an ammonium uranate concentrate.

LKAB applied for a siting permit according to the section 136a of the Building Law in 1980 (3-11). After a lively debate the community of Arjeplog chose not to veto the proposed project. Nevertheless LKAB decided to withdraw their application and offered SKBF to take over. Considering the complicated type of mineralization, which made a reliable ore reserve calculation difficult, this offer was not accepted by the SKBF and the Pleutajokk project was terminated.

3.3 Uranium in alum shale

Alum shale deposits in Sweden have been of interest since a long time for their content of valuable constituents, e.g. alum during the 18th century, oil during the second World War and uranium in recent time. Figure 3-1 shows the alum shale districts in Sweden.

The geological investigations performed by SGU and AE during the 1950’s established Billingen-Falbygden the most important district as a uranium resource. Over a large area (about 500 km²) the uranium is there found in a nearly horizontal, 2.5–4 m thick layer, with a mean uranium concentration of about 300 g/t (0.03%). The geological conditions in the Billingen-Falbygden area are illustrated in figures 3-2 and 3-3.
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Figure 3-1
Alum shale occurrences in Sweden (B-F = Billingen-Falbygden)

Figure 3-2
Geological map of the Billingen-Falbygden area

Figure 3-3. Billingen - general geological section
Figur 3-4 illustrates the variation in uranium and kerogen contents over the whole alum shale formation (about 22 m thick) at Ranstad. Also the paleontological zones are shown. Figure 3-5 details the uranium-rich zone and adjacent members above and below. Of practical importance are the stink stone horizons, defining the roof and bottom when mining, and the kolm lenses, contributing with some 10% of the mean uranium content.

Figure 3-4. Variation in the contents of uranium and organic carbon in the alum shale formation of the Ranstad area.

Fig 3-5. The Ranstad member of the alum shale formation.
The mineralogy of the Billingen-Falbygden alum shales is dominated by illite (30%), quartz (25%), feldspar (10-15%), kerogen (18%) and pyrite (12%).

The small kolm lenses, usually a few centimetres in thickness, have 2000-7000 ppm uranium content.

The stinkstones generally contain 80-90% of calcium carbonate, while the remaining 10-25% have the same mineralogy as the shale.

Quantities of uranium and kerogen in the Billingen-Falköping alum shale formation are presented in the following table (from ref 3-14).

Table 1. Quantities of shale and stinkstone and resources of uranium and kerogen in the Upper Member of the Alum Shale Formation in the Billingen-Falbygden area, Västergötland. (The tonnages are metric.)

<table>
<thead>
<tr>
<th>Units in Upper Member of Alum Shale Formation</th>
<th>Shale (excluding kolm and stinkstone)</th>
<th>Sinksone (wt %) (billion tonnes)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Shale (billion tonnes)</td>
<td>Kerogen (billion tonnes)</td>
</tr>
<tr>
<td>Top shale</td>
<td>2.2</td>
<td>14.5</td>
</tr>
<tr>
<td>Uranium-rich unit</td>
<td>3.4</td>
<td>15.5</td>
</tr>
<tr>
<td>Bottom shale</td>
<td>2.6</td>
<td>12.0</td>
</tr>
<tr>
<td>Total</td>
<td>8.2</td>
<td>14.0</td>
</tr>
</tbody>
</table>

Comprehensive geological descriptions of alum shales in Sweden are given in ref 3-13 and 3-14, from which the illustrations in this chapter have been taken. The geology of the Närke area is described in ref 3-15, including general views on the processes involved in the sedimentation of the shale constituents.

The potential of the alum shale deposits as mineral and energy raw material is investigated in ref 3-16.

3.4 Concluding comments

The uranium exploration efforts in Sweden have so far not resulted in a persistent uranium production. In the latest IAEA "Red Book" (3-16) Sweden reports 4000 tonnes "Reasonably Assured Resources" and 6300 tonnes "Additional Assured Resources" in the cost category up to 130 US$ per kg U (Lilljuthatten, Åsele, Pleutajokk and Kvarnån). From low-grade black shales (Ranstad) 300 000 tonnes are reported extractable in higher cost categories.
REFERENCES

3-1 Svenke, E.
Trials to utilize kolm industrially.
Lecture at the Royal Swedish Academy of Engineering Sciences, 12 October, 1967

3-2 Svenke, E.
The occurrence of uranium and thorium in Sweden
UN Conference, Vol. 6, Geology of uranium and thorium.
Geneva, 1955

3-3 Mårtensson C., Welin, E.
Uranium mineralization in iron ore of central Sweden.
UN Conference, Vol. 2, Survey of raw materials,
Geneva, 1958

3-4 Ore resources and exploration.
Report from the mineral policy investigation committee.
(In Swedish)
DsI 1978:16
Department of industry, 1978


3-7 Government Bill 1976/77:100, Appendix 17.


3-9 SKB Annual Report 1985
Technical Report 85-20
SKB, May 1986.

3-10 Account of the actual status regarding nuclear fuel and the activity of the Swedish Nuclear Fuel Supply Co. during 1978 and 1979 up to and including the month of September. (In Swedish).
SKBF, Report to the Department of industry, November 1979.

3-11 Projekt Pleutajokk
Application according to Section 136a of the Building law.
(In Swedish).
LKAB, December 1980.

3-12 Account of the actual status regarding nuclear fuel and the activity of the Swedish Nuclear Fuel Supply Co. during the period October 1980 - October 1981. (In Swedish).
SKBF, Report to the Department of industry, November 1981.

3-13 Andersson, A., Dahlman, B., Gee, D.G., Snäll, S.
The Scaninavian alum shales
Ser. Ca, No.56.
3-14 Andersson, A., Dahlman, B., Gee, D.G.
Kerogen and uranium resources in the Cambrian alum shales of
the Billingen-Falbygden and Närke areas, Sweden
Transactions of the Geological Society in Stockholm,

3-15 Eklund, J.
The rock formation. The primitive ages and the future of
Kumla. (In Swedish).
The Kumla region: Past, present and future.

3-16 Alum shale.
Sind PM 1978:3
Investigation by the State Industry Authority.
Stockholm, 1978
4 PROCESS AND PLANT DEVELOPMENT

When work on uranium recovery was first started by FOA in 1946, it was hoped that the ashes from shale oil recovery during the Second World War could be used as source material. Melting of shale ashes in an electric furnace resulted in the formation of two layers, a top layer of silicate slag and a bottom layer of mainly FeSi-metal. Experiments at FOA suggested an enrichment of uranium in the metal layer, when direct current was used. However, the results were inconsistent, as were also the results from trials to heat the ashes with carbonates of calcium and potassium followed by alkaline leaching (4-1).

4.1 The leaching process

Another road to utilize the ashes obtained from shale oil recovery was tried in 1947 at KTH, showing that a large part of the uranium could be transferred to volatile or leachable form by sulfatizing or chloridizing roasting. Difficulties in the treatment of the shale ashes turned the interest to direct leaching of shale, at the Shale Oil Co. with sulfuric acid, and at KTH and the Boliden Mining Co. with hydrochloric acid after treatment of the shale with chlorine.

These first leaching experiments on shale were not successful, and the interest was turned to kolm. The work at FOA was concentrated on leaching of kolm with nitric acid. In parallel a process was developed at KTH which first included chlorine treatment followed by hydrochloric acid leaching, and finally resulted in a multistage leach system with a circulating Fe(III)-chloride solution and reoxidation of the leach solution with chlorine (4-2). This process was applied at the small pilot plant at Vinterviken (see chapter 2). The Shale Oil Co. at the same time investigated kolm leaching with sulfuric acid, continuing this work until the middle of the 1950s.

Also carbonate leaching was studied during the 1950s, in laboratory scale on untreated shale and on shale roasted at low temperature. The results were not encouraging. The very selective alkaline leaching required finer grains and higher temperature than acid leaching. An identified risk was also that sulfur contained in the shale is oxidized to sulfuric acid, which neutralizes the alkaline leachant. The alkaline leach process was con-
Roasting of shale as pretreatment before leaching was studied at AE. It was found that pure oxidizing roasting at low temperature, around 600°C, under careful control, could increase yields by 10-15% when combined with percolation leach. After roasting, particles break down easily in water to fine grain material. A good economy of this process was found to be achievable, if the heat produced in the roasting stage is utilized, the sulfur converted to and sold as sulfuric acid, and the residue after roasting and leaching marketed as light-weight building materials (4-3). Later investigations showed, however, that such a market of sufficient size would not be available, and the process was never demonstrated on large scale. In hindsight it may also be concluded, that the radioactive contents of the residues would have prohibited their use as building materials.

Leaching was tried both as percolation leach on coarse-grain material and as agitation leach on fine-grain material. Higher yields were obtained in the latter case in a continuous process with fairly small vessels. The problems were found to be the separation of solids and the disposal of leach residues. Economic estimates indicated that costs both for investment and operation were lower for percolation leach. The main part of the development work was thus focused on percolation leach.

Compared to other uranium ores, the alum shales at Kvarntorp and Billingen are special, not to say unique. They have been formed by sedimentation in a shallow bay followed by metamorphosis, and are composed of silicates, mainly quartz and mica, microcrystalline pyrite, and an organic kerogen substance. The shales are extremely dense, giving low radon releases in mining operations. This also means, however, that long leaching times are required. After leaching with 15% sulfuric acid during 5 days at 60°C the penetration depth is only 0.5 mm. Uranium at greater depths remains to great extent unavailable for leaching (4-1, 4-4).

The choice of acid concentration and temperature of the leach solution needs a careful technical and economical optimization. Increased acid concentration and temperature result in the dissolution of more aluminium due to the decomposition
of minerals, while lower acid concentration and temperature give a slower dissolution of uranium.

If shale after milling is stored in the open and allowed to weather, a higher yield of uranium is obtained in the leaching step. The yield increase observed was as high as 10% for the Ranstad shale but quite small for the Kvarntorp shale. The phenomenon was found from seemingly inconsistent leaching yields and is due to an autooxidation process (4-5).

4.2 Kvarntorp

Shale exploitation

The first Swedish trials to recover oil from alum shale were seemingly started in 1864. During the 1890s a few tonnes of oil were recovered from shale from the Kinnekulle mountain. During the first world war the Kinnekulle Works were established and a state committee investigated future potential. The military marine authorities were early interested and started a plant there in 1939 to recover oil for the Swedish navy (4-6).

In the same year the Swedish Geological Survey (SGU) investigated the shale formation at Kvarntorp and confirmed its extent, bituminous content, and ease of the surface removal required. The cut off of foreign import and the growing demand of oil supply for the national defence and industry initiated activities to explore the Kvarntorp shale.

In 1940 a committee was charged with the task to prepare industrial shale oil recovery at Kvarntorp. The Swedish Shale Oil Co. was formed in January 1941 and produced the first oil in April 1942. Oil was produced in the following years both by pyrolysis of shale in furnaces and in situ by electrical resistance elements emplaced in drill holes. The environmental effects were extensive, however, due to the releases of dust (>10 t/d) and sulfur dioxide (265 t/d).

Yet a great optimism surrounded the utilization of the Kvarntorp shale in the middle of the 1950s with uranium as only one of the products from the shale exploitation. One of its largest potentials was considered to be a supply of reduction gas for the production of high-quality iron from iron ores in Middle Sweden (4-7). The shale formation was estimated to be sufficient for a 400 year
supply of oil, ammonia, and lime for industry and farming. The ashes from total gasification of the shale would be used for the production of materials for the construction of roads and buildings. Besides uranium, also aluminium, potassium, molybdenum, and vanadium would be produced from the shale. However, steadily increasing losses and the overall economic development led to the decision in 1961 to liquidate the Swedish Shale Oil Company.

The overall sales from the total operation had included oil and oil products amounting to around 2.4 mill. m³. Also considerable amounts of ammonia, sulfur, gas, building materials, and electricity had been produced and sold. What is left as a monument from the period of shale exploitation is a 100 m high mountain of shale ashes, which for a long time has continued to spread a sticky smoke to its surroundings—a sharp contrast to the careful restoration of nature after the Ranstad operation (see chapter 8).

Uranium recovery

Based on R&D work by AE in laboratories at Liljeholmen and Vinterviken in Stockholm, a small industrial plant for a uranium production of 5-10 tonnes per year was started at Kvarntorp in the summer of 1953. A flowsheet for the original plant is shown in figure 4-1 (from 4-4).

The shale ore is quarried in opencast mining and crushed to a size of -100 mm to +6 mm. Shale is separated from limestone as a float product in a heavy media separation process (HMS) with the aid of a magnetite suspension with a density of 2.2-2.5 t/m³. The pulp of the HMS-medium contains around 32% solid iron oxide by volume. Magnetite in wash waters from washing screens for float and sink materials is recovered by means of magnetic separators and demagnetized in a solenoid coil before reuse.

In the second part of the flowsheet the separated shale is fed to vats for percolation leach by sulfuric acid. Feeding is by a stainless steel belt and a screw conveyor distributing the material evenly over the surface of the vats. The percolation leach temperature is 60°C with a maximum acid concentration of 180 g/l. Acid consumption was kept at 65 kg per tonne of shale.

The pregnant leach solution was separated from
Fig. 4.1 Flowsheet for the Kvarntorp uranium recovery plant.
the solid residue with an optimized yield of the leaching process of 65-70%. The leach liquor is a highly impure solution with a uranium content of around 0.6 g/l. Uranium was first precipitated as hydroxide followed by digestion with sodium or ammonium carbonate (4-3). A uranous phosphate precipitation process was developed in 1950 and used at Kvarntorp between 1953 to 1957. The end product was then a dried, highly impure uranium(IV) phosphate.

The selective sorption of uranium from sulfate leach solutions by strong anion exchangers was tried on pilot scale at Kvarntorp in 1954 with a commercial scale equipment installed there in 1957. An acidified ammonium nitrate solution was used for the elution of the uranium. The pH was adjusted to 3.5 in the eluate with limestone, which separated most of the impurities. Uranium was precipitated in the filtrate from this step with ammonia as ammonium uranate, and after filtration and washing dried and sent to the refinery in Stockholm.

The solid residue after leaching was sucked out from the vats with a rubber hose and sent in a pipe line to the dump 400 m away. The spent process solutions were neutralized with limestone and disposed of in the same area. The waste water from this area was collected and treated further before release to aqueous recipients.

The Kvarntorp plant was shut down in 1963. It required an investment of 10 MSEK (2 MUS$). During its operation from 1953 to 1963 a total of 62 tonnes of uranium were produced, part of which originated from import material.

The construction and operation of the Kvarntorp plant made possible full scale testing and evaluation of process flowsheet design, equipment, and materials. The materials used included acid proof stone, tubing and equipment lining of rubber and plastic, and impervious graphite. The Kvarntorp experience, including waste treatment, thus formed an important basis for the construction and operation of the plant at Ranstad.

4.3 Refining of uranium concentrate

In 1949 AE started small scale purification of uranium from impure sodium and calcium uranates at Vinterviken. The uranate was dissolved in nit-
ric acid, purified by ether extraction, and uranium precipitated with hydrogen peroxide from aqueous solution (4-8).

The processes used in the Stockholm factory at Liljeholmen for the purification of uranium concentrate in the form of uranous phosphate, and for the production of uranium oxide and metal were presented at the second PUNE conference in Geneva in 1958 (4-9).

A simplified flowsheet for the refining process is shown in figure 4.2. The two steps in the flow-sheet called "Caustic Digestion" and "Carbonate Process" were developed at KTH, while the rest of the flowsheet was developed at AE.

The purpose of the caustic digestion is to remove phosphate by converting the uranous phosphate to uranous hydroxide. This is accomplished with 2M sodium hydroxide solution at 95°C in two stages with a 2 hr hold up in each stage. The uranous hydroxide is washed with water, oxidized with hydrogen peroxide, and treated with ammonium bicarbonate at 60-70°C for 1 hr. Most of the uranium is then dissolved. The solution is filtered and ammonium uranyl carbonate is crystallized by the addition of more ammonium bicarbonate and cooling the solution to 10°C. The crystalline product is filtered and washed with saturated ammonium bicarbonate solution. Half of the filtrate is recirculated. The rest is brought to pH 5 with sulfuric acid, carbon dioxide expelled by boiling, and the precipitated uranate filtered off. The insoluble residue from the carbonate process is fed back to the shale leaching step to decrease overall losses.

Further uranium purification is achieved by solvent extraction. The ammonium uranyl carbonate is dissolved in nitric acid and calcium nitrate added to the solution. In 6 in. diameter multistage mixer glass columns with plates, baffles and impellers of stainless steel, uranium is extracted with hexone (methyl isobutyl ketone) and stripped with distilled water. This produces a reactor grade quality of uranium.
Fig. 4.2 Refinery flowsheet.

<table>
<thead>
<tr>
<th>UO$_3$ from Ammonium Uranyl Carbonate</th>
<th>UO$_3$ from Homogeneous Precipitation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Contents in p.p.m.</td>
<td>Contents in p.p.m.</td>
</tr>
<tr>
<td>B</td>
<td>&lt;0.5</td>
</tr>
<tr>
<td>Cd</td>
<td>&lt;1</td>
</tr>
<tr>
<td>Co</td>
<td>&lt;3</td>
</tr>
<tr>
<td>Mn</td>
<td>&lt;3</td>
</tr>
<tr>
<td>V</td>
<td>40</td>
</tr>
<tr>
<td>Ni</td>
<td>&lt;3</td>
</tr>
<tr>
<td>Cr</td>
<td>3</td>
</tr>
<tr>
<td>Cu</td>
<td>9</td>
</tr>
<tr>
<td>Fe</td>
<td>~100</td>
</tr>
<tr>
<td>Na</td>
<td>&lt;1000</td>
</tr>
<tr>
<td>Ca</td>
<td>~100</td>
</tr>
<tr>
<td>Ba</td>
<td>low</td>
</tr>
<tr>
<td>P</td>
<td>&lt;30</td>
</tr>
<tr>
<td>Si</td>
<td>low</td>
</tr>
<tr>
<td>Sn</td>
<td>&lt;3</td>
</tr>
<tr>
<td>Mg</td>
<td>20</td>
</tr>
<tr>
<td>Pb</td>
<td>&lt;3</td>
</tr>
<tr>
<td>R.E.</td>
<td>not detectable</td>
</tr>
<tr>
<td>B</td>
<td>&lt;0.5</td>
</tr>
<tr>
<td>Cd</td>
<td>&lt;1</td>
</tr>
<tr>
<td>Co</td>
<td>&lt;3</td>
</tr>
<tr>
<td>Mn</td>
<td>&lt;3</td>
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<tr>
<td>V</td>
<td>&lt;3</td>
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<tr>
<td>Ni</td>
<td>&lt;3</td>
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<tr>
<td>Cr</td>
<td>&lt;3</td>
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<td>Fe</td>
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<td>Ca</td>
<td>&lt;100</td>
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<td>&lt;3</td>
</tr>
<tr>
<td>Pb</td>
<td>&lt;3</td>
</tr>
<tr>
<td>R.E.</td>
<td>not detectable</td>
</tr>
</tbody>
</table>

Table 4.1 Impurities in UO$_3$ prepared from ammonium uranyl carbonate and from homogeneous precipitation.
In the final step of the refinery process, uranium is precipitated from the nitrate solution obtained by ammonia, slowly formed by hydrolysis of carbamide at elevated temperature. The homogenous precipitation is carried out in 150-200 kg batches of uranium in steam heated vessels under stirring. The solution is heated to boiling and a readily filtered precipitate obtained. It is dried and calcined to UO₃ in a tray furnace at 250°C.

Table 4.1 compares the impurity levels of uranium from the carbonate process and from the final homogenous precipitation step.

Later, at Ranstad, a continuous precipitation process was developed to give a uranium concentrate of sodium uranate. This was refined at the Stockholm refinery by dissolution in nitric acid and extraction with tributyl phosphate. The homogenous precipitation process was kept, however, with carbamide added already in the reextraction step to increase reextraction efficiency.

4.4 Production of uranium oxide and metal

Until 1956 the main direction in Sweden aimed at uranium metal as a reactor fuel, which then changed to sintered uranium dioxide.

For metal production the calcined UO₃ was reduced by hydrogen at 625°C in a continuously operating furnace. The oxide powder was fed in 10 kg batches through an externally heated, rotating tube with hydrogen in 1.5 times stoichiometric excess passing in counter-current direction over the oxide bed. The furnace capacity was 25 kg/hr of UO₂. The UO₂ was by intention produced in a reactive state to facilitate its subsequent fluorination, and was stored under an atmosphere of carbon dioxide.

Hydrofluorination of UO₂ was carried out at 450°C in a flow of preheated HF-gas in a continuous furnace. Production capacity was >10 kg/hr. The UF₄-product contained <0.5% each of UO₂ and UO₂F₂, respectively.
The final reduction to uranium metal was carried out in a batch process. UF₄-powder and a 10% excess of calcium in 4-5 mm granules were intimately mixed in a cylindrical graphite container, connected to a sintered CaF₂-crucible at its bottom. The reaction pot was enclosed in a stainless steel container, which was evacuated and filled with argon before initiation of the reaction. A normal batch operation permitted the reduction of 30 kg ingots with a yield of 98%.

The metal billets obtained were remelted under high vacuum in an Al₂O₃-lined graphite crucible and cast into moulds of graphite or steel. In a melting operation of 5 hr duration 400 mm long rods of 30 mm diameter were produced. Also 12 mm rods were produced.

A large effort was made to establish optimum process conditions for the production of sintered uranium dioxide, both in the oxide preparation and in the sintering step (4-9, 4-10).

Investigations of the precipitation process were performed with carbamide (urea), ammonium bicarbonate, and gaseous ammonia. A detailed characterization was made of the products obtained under different precipitation and sintering conditions. Optimum conditions to achieve a high sintered oxide density (>97%) were i.a. a reduction temperature of 700-800°C and a hydrogen supply of 2-3 times stoichiometric requirements. With these conditions the dioxide obtained had a specific surface of 5 m²/g.

After an agreement between the Swedish state and the private company AB ASEA the fuel element factory at Liljeholmen was transferred to the new and jointly owned company AB Asea-Atom 1 January, 1969. The restructured AB Atomenergi was also during 1969 made a 100% state company (4-11). ASEA later acquired all shares of Asea-Atom, which today, as ABB Atom, owns and operates a 400 tonne per year fuel element factory at Västerås.
REFERENCES

4-1 Strandell, E.
Recovery of uranium from shale and kolsm.
Personal communication, 1992. To be published.

4-2 Stelling, O., and Svenke, E.
Method to recover uranium compounds from uranium containing kolsm and/or shale.

4-3 Peterson, A.
Status report from Sweden.
IAEA, Vienna 1967.

4-4 Svenke, E.
Recovery of uranium from uranium bearing alum shale.

4-5 Svenke, E.

4-6 Ruhnbro, R.
Kvarntorp, one summer.

4-7 Eklund, J.
The alum shale of Närke and the iron ore of Bergslagen.
The Kumla region. Past, present and future.
(In Swedish), Kumla 1961.

4-8 Peterson, A.
Uranium recovery and preparation of high purity uranium products. (In Swedish).
Svensk Kemisk Tidskrift 79:3 (1967).

4-9 Gelin, R., Mogard, H., and Nelson, B.
Refining of uranium concentrate and production of uranium oxide and metal.

4-10 Runfors, U., Schönberg, N., and Kiessling, R.
The sintering of uranium dioxide.

5 URANIUM PRODUCTION AND PROCESS DEVELOPMENT AT RANSTAD

5.1 Siting

As briefly described above, the decision to erect an industrial uranium production plant based on the shale resources was taken in 1958. The previous study (5-1) had been concentrated on two siting options, namely Närkes Kvarntorp, with possibilities to co-operate with the SSAB (the Swedish Shale Oil Company) alum shale mining for oil production, and the Billingen-Falbygden area in Västergötland, where the geological investigation had indicated a suitable mining area at Ranstad. The higher uranium content at Ranstad, 300 g/t, compared with 235 g/t at Kvarntorp, resulted in lower calculated capital and operation costs for production and thus the decision was in favour of Ranstad. The long term shale mining conditions were also considered more favorable at Ranstad.

5.2 Design and construction

The pre-project (5-1), presented to the government in the summer of 1957 and proposing a 120 t/year uranium plant at Ranstad, had been worked out by a project group within AE. A number of consultants had contributed, among them SGU (geology), KTH (mining and mineral dressing) and VBB, the Road and Building Construction Co (water treatment and industry planning). After the government decision in 1958, the project group, placed in Stockholm, went ahead with the design, detailed planning, and construction of the Ranstad plant, based on the 1957 pre-project. Eventually a local organisation was established at Ranstad for the mine preparation, building construction, and machine installations.

Ranstad was one of the first Swedish projects using a computer supported net work system (PERT) for the project administration (5-2).

Several big and small (local) Swedish contractors were engaged for civil engineering and construction. Foreign companies were also engaged with their own personnel for installation work, notably German deliveries of heavy machinery and acid resistant brick work.

A number of key operation personnel was recruited early, to get used to systems and machinery during installation. The mining preparation was thus performed exclusively by personnel within AE, using early purchased mining equipment.
When the plant was ready for production in the summer of 1965, a complete and trained operation organisation of about 135 employees was available.

5.3 The Ranstad plant of 1965

The uranium production plant of 1965 (5-3, 5-4, 5-5, 5-6, 5-7, 5-8) was dimensioned for a production of 120 tonnes of uranium per year, requiring an annual shale mining of 800 000 tonnes. However, the plant lay-out was designed to allow a future expansion to the double capacity by increasing the working shifts per day (e.g. in mineral dressing) or adding twin units (e.g. in the leaching plant). The mining was based on an open-pit developed for 15 years of production at 800 000 tonnes per year.

The mineral dressing and uranium recovery processes were specially developed for this particular kind of low grade shale material (see chapter 4). Most important was the choice of acid percolation leaching of relatively coarse material, influencing the processes up- and down- stream, as will be commented on in some detail below.

The flow sheet for the plant is illustrated in figure 5-1.

Fig. 5-1 Flow Sheet for the Ranstad Uranium Plant
The uranium recovery was legally based on a license for shale mining (according to the Mineral Coal Law, later replaced by the Mineral Law), and on a decision by the Water Court giving permission to take out fresh water from deep wells and to release mine drainage water and process water into aqueous recipients. The company's general license under the Atomic Energy Law covered also the nuclear aspects of the Ranstad operation.

For the mine, industry, and waste disposal areas about 7.5 km² of land was acquired from private owners.

**Mining**

The geological situation (chapter 3) at the Ranstad site offers a considerable area with favorable conditions for shale mining in open pit. However, for a long term operation, especially at an increased production rate, underground mining is necessary. For this reason the primary crushing plant was placed underground, with the intake hopper at the uranium shale bottom level, connected to the open pit area by a double transport tunnel (1300 m long) driven in the uranium and top shale layer. The crushing station and transport tunnels could be used for the development of a future underground mine (figure 5-2).

![Diagram of plant and mine areas](image-url)
The sedimentary formation in the Ranstad region is nearly horizontally layered and tectonically very little disturbed. The uranium-rich zone of the alum shale has a fairly constant thickness (3.5 m), uranium content (300 g/t) and uranium distribution over a large area. At the bottom and top of the uranium shale there are more or less continuous layers of limestone in the form of "stinkstone" lenses, easy to follow during mining, both underground and in open pit. The overburden conditions in the area developed for the first 15 years of opencast mining are shown in figure 5-3.

Mining methods, especially the use of walking draglines, were studied at Kvarntorp and at English coal and iron stone quarries, with similar geological conditions.

The opencast mining process (5-9), see figures 5-4 and 5-5, was planned for the handling of all overburden with a large dragline machine. An American machine with a dumping radius of 57 m and with a 12 m³ bucket was chosen - the biggest walking dragline ever used in Scandinavia. Walking on top of the blasted limestone-top shale layer the dragline machine could uncover a 20-25 m wide strip of uranium shale along a 2000 m long front of advancement. All soil and rock overburden was dumped on the excavated side, refilling the pit to approximately the original surface level.
The uncovered uranium shale was blasted, loaded with an electric excavator into trucks (payload 35 tonnes) and transported via the tunnels to the underground primary crusher. Large boulders of stinkstone included in the uranium shale were separated with the excavator and stayed on the pit bottom.

Rotary drilling machines with electro-hydraulic drive were used for the blast hole drilling in the limestone/top shale and uranium shale benches. These were the first rotary rock drilling machines to be developed in Sweden.

The dragline machine was delivered and commissioned in 1961. It was used for the mine development work from 1962 to 1965, together with two 25 t crawler tractors and drilling machines. About 1.5 million m$^3$ of overburden were moved during this period to open the first mining cut ("box cut").
The soil and rock from the box cut was dumped on the original surface along the mining area border, creating a 2000 m long, 10-15 m high new hill top silhouette in the landscape.

The water inflow to the mine, about 1.5 million m³ per year (double the nominal shale production by weight), was mainly groundwater from the limestone layer.

Underground mining was also thoroughly investigated. A new type of room-and-pillar-mining, was designed and practically tested (5-10), figure 5-6, offering the possibility to separate the drilling and blasting operations from the shale loading and transport. However, the opencast mining as described was found to be the best option from economical and safety aspects for the first period of operation.

![Fig. 5-6 Underground mining concept](image)

**Primary crushing, storing & mineral dressing**

The mineral dressing processes have the purpose to transform the coarse mine output into a suitable material for the leaching process. At Ranstad the desired leaching material was a product with a grain size of 3 mm, as free as possible both from oversized grains and fine particles, and also with a low CaO-content. The mineral dressing problem was to accomplish the necessary comminution and separation processes without too high losses.
The raw material from the mine, with shale size up to 0.5-1 m, was first crushed in the underground primary crushing station, figure 5-7.

The trucks dumped the material in a granite lined hopper. The shale was fed to the crusher via a roller screen. The gyratory crusher (German delivery) with intake opening 1900 mm had a capacity of 700 t/h, dimensioned for a doubling of the plant production. The material from the crusher, reduced to minus 350 mm, and from the roller screen was transferred by an oscillating table feeder and an accelerator to an inclined, about 400 m long rubber conveyor belt, direct from the underground to the top of the silo. To accommodate the material for the down-stream limestone separation plant, there was a screening station on top of the silo. The material was divided into three fractions, 0-6 mm, 6-40 mm and 40-350 mm. The fraction minus 6 mm was removed as waste. The storage capacity in the silo, dimensioned for the double plant production, was about 7000 m$^3$ for the coarse fraction, 2800 m$^3$ for the fine fraction and 550 m$^3$ for the waste fraction.

From the bottom of the silo the fine and coarse fractions were withdrawn by oscillation feeders onto two parallel conveyor belts, leading to the limestone separation system in the mineral dressing plant, figure 5-8.
The difference in density made it possible to separate the limestone from the alum shale with a heavy media separation system, having one line for the 6-40 mm and one for the 40-350 mm fraction. After a washing screen the material entered the separation vessel, containing the heavy media slurry. The limestone inclusions sank to the bottom, together with some heavy shale owing to high pyrite content. This sink product was lifted out of the separation vessel by a large diameter, inclined wheel with radial lifters, and was then passed over a de-watering screen into a discharge hopper. The separated alum shale, floating on the heavy media, was pushed over the forward rim of the separating vessel with a paddle wheel onto a washing screen, and from there into the crushing department. The heavy media was a slurry of very fine ground, high quality magnetite concentrate, held in water suspension by continuous circulation with heavy pumps. The effective density of the slurry was maintained at 2.35 t/m³. The water containing magnetite from the washing screen after the separating vessels was pumped to a magnetic separator, from which the magnetite was returned to the heavy media slurry circuit, thus keeping the magnetite losses down. The heavy media separation plant was of a French design, normally used in continental plants for coal washing, with considerably lower slurry density.

The waste sink product represented a material loss of about 10% of the feed tonnage to the dressing plant. The CaO-content in the separated shale was about 1.2%.

Fig. 5-8. The mineral dressing plant
The crushing and screening of the shale from the heavy media separation was the final step in producing the -3 mm material for leaching. Comprehensive testing at KTH, confirmed in larger scale in the pilot plant at Kvarntrup, showed that rod mill grinding produced a particle form and size distribution unfavorable for leaching. The final choice was to install impact crushers and free-swinging resonance screens. As shown in fig 5–8 the coarse fraction from the heavy media separation was crushed in three steps, the fine fraction in two steps. The screening was performed after the second crushing step with a recirculation over the third crushing step. Crushers and screens were both of German design. The mineral dressing was dimensioned for 350 t/h leaching material, sufficient for doubling the nominal uranium production by increasing the operating hours per day.

An important part of the crushing-screening plant was an effective system for the removal of the dust fraction (here defined as -0.1 mm particles). An air flow rate of 375 000 m³/h created under-pressure in the hoods over all dust producing machines, ensuring a good working environment and giving a product with good percolation characteristics in the leaching. The dust was precipitated in an electro-filter. The dust fraction represented a material loss of about 10%. However, this was later eliminated by the development of a pelletizing process (see section 5.6).

The product on the outgoing conveyor belt ready for leaching was continuously sampled and monitored for oversize particles by test-screening.

The crushers and dust control system represented a major electric power consumer, with an installed power of about 3 MW.

Open storage and leaching

A buffer storage for the leaching product between the mineral dressing and the leaching was necessary to ensure a good operation efficiency in both plants, making them in the short term independent of each other. During the process development it was also found that the leaching characteristics were improved when the shale had been stored in free air for a couple of weeks before leaching. For these reasons a buffer storage with a capacity of 40 000 tonnes, or 3 weeks production, was installed, see figure 5-9.
To minimize segregation, that could cause inhomogeneous flow of the leach solutions, the shale from the dressing plant was built up of successive layers with a height of about 1 m by the remotely controlled stacker crane.

The storage area had an asphalt bottom surface, equipped with a pipe system, making it possible to blow air into the stored shale to promote the chemical processes. However, in practice this system was found to give only a local and too intense effect (rise of temperature with risk for ignition). A reversal to suction of air from the outside into the shale gave some improvement, but on the whole the weathering effect observed in the development work could never be fully reproduced in large-scale operation.

From the storage the shale was loaded with a grab crane onto a conveyor belt leading to the leaching plant. The stacker and the grab crane were delivered by a German company.

The leaching plant was designed for percolation leaching of 2000 tonnes of shale per day with sulphuric acid. The main leaching parameters at nominal production were as follows (5-8):

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shale particle size</td>
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<tr>
<td>Dust fraction (-0.1mm), max</td>
<td>4 %</td>
</tr>
<tr>
<td>CaO-concentration, max</td>
<td>1.4 %</td>
</tr>
<tr>
<td>Total cycle time</td>
<td>4 days</td>
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<tr>
<td>Effective leaching time</td>
<td>50 h</td>
</tr>
<tr>
<td>Leaching temperature</td>
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</tr>
<tr>
<td>Mean acid concentration</td>
<td>80 g/l</td>
</tr>
<tr>
<td>Max &quot;</td>
<td>140 g/l</td>
</tr>
<tr>
<td>Min &quot;</td>
<td>15 g/l</td>
</tr>
<tr>
<td>Depth of shale bed in the leaching vats (2000 t)</td>
<td>3.6 m</td>
</tr>
</tbody>
</table>
The leaching plant (figure 5-10) had 4 leaching vats 25x25 m, effective height 3.6 m, each taking 2000 tonnes of shale. The vats were constructed of strongly reinforced concrete. On the inside they were protected from the acid by a rubber lining and a heavy acid resistant brick work. The bottom of the vat was designed as a sand filter, supported by an acid resistant brick construction and protected by granite "fenders" against the grab shovel used for emptying the vats from the leaching residue. The leaching solution entered the vats through pipes along the upper rim and was withdrawn through four bottom openings, underneath the filter.

The concrete construction (Swedish) for the leaching vats as well as their acid resistant protection system (German) were technically very advanced features with strict quality requirements.
The shale entering the leaching plant on the conveyor belt from the open store (capacity 250 t/h) was pre-heated and moistened with steam, before being transferred to the leaching vat. The filling machine was programmed to unload the shale from a steel band conveyor in thin layers, to prevent particle segregation.

The leaching solution was transferred in such a way between the leaching vats and buffer tanks, that a batchwise, counter-current leaching was established. Very schematically this is shown in the flow sheet, figure 5-1.

Three vats were involved in the leaching cycle, the fourth being occupied for unloading of leaching residue and loading with fresh shale. At full operation ("7/7-operation") one leaching vat was unloaded and loaded each day in the week. The material flow rate was then 7 x 2000 tonnes of shale per week.

At full operation the leaching cycle was totally 4 days, including one day for unloading and loading. Each batch of shale was passed by 4 leaching solutions with rising acid concentration, from 15 g/l H₂SO₄ in the first up to 140 g/l in the last leaching solution, followed by 2 washing solutions. Each batch of solution replaced the previous solution downstream from top to bottom of the vat. However, 2/3 of the first leaching solution to the fresh shale was pumped in from the bottom to drive out air from the shale bed.

For the reduced capacity operation during 1965-69 three vats per week were loaded and unloaded ("3/7-operation") and a somewhat modified leaching cycle was used.

One leaching cycle needed about 900 m³ of fresh water and 100 m³ of concentrated acid. About 250 m³ of water was lost in the leaching residue, and 750 m³ was transferred as pregnant solution to the uranium recovery plant. As an average 69% of the uranium content in the shale was found in the pregnant solution.

To enable the transport of solutions between the vats there was a comprehensive system of rubber lined pipes and pumps in the cellar compartment under the leaching vats. This system also included heat exchangers to keep the leaching temperature, and storage tanks (outside the building) for leaching solutions and for concentrated acid.
The leaching residue was unloaded from the vats by an overhead grab crane into 35 t trucks (of the same type as those used in the mining).

The percolation leaching process had the advantages of a low leaching solution volume, a particle free pregnant solution to the uranium recovery plant, and a drained leaching residue that was possible to handle with ordinary transport equipment without extra filtering.

**Uranium recovery from the leaching solution**

The pregnant solution leaving the leaching plant contained 15-20 g/l free sulphuric acid (pH 0.7-1), about 0.6 g/l uranium and in addition iron, aluminium, magnesium, sulphate and phosphate as main constituents.

The uranium was recovered from the solution with the ion exchange-ELUEX-process. The flow-sheet for the uranium recovery is included in figure 5-1 and the equipment is illustrated in figure 5-11.

As a preparation for the ion exchange the pH of the solution was raised to about 1.8 by the addition of fine-ground limestone. The resulting gypsum was removed by filtration, first on a 12 m² drum filter and then on a pre-coat filter. The net filtration capacity was about 0.6 m³/min.

The clear solution was pumped to the ion exchange system (a British delivery), having four rubber lined steel columns, each with 9 m² of resin. Normally three columns in series were used for the adsorption cycle.

The adsorption capacity of the resin (Amberlite IRA 400) was about 40 kg/m². The uranium content in the percolate at saturation was about 2 mg/l. A strong sulphuric acid (15%) was used for the elution of the uranium from the saturated resin. The elution was performed in a 3-step cycle. The eluate from the second step had a uranium content of 7-18 g/l and was transferred to the solvent extraction plant. The capacity of the ion exchange resin was gradually declining by "poisoning" from such elements as Mo and S, and from SiO₂.
Figure 5-11: Equipment for the uranium recovery from the leaching solution
The resin therefore had to be regenerated after 30-15 adsorption-eluation cycles, depending on the condition of the resin. For regeneration the resin was transferred from the ion exchange column to a special tank, where it was treated with sodium hydroxide and sodium sulphate.

The eluate from the ion exchange plant to the ELUEX solvent extraction process contained about 15 g/l uranium and 100 g/l sulphuric acid, forming complex uranium sulphate ions. From this solution the uranium was recovered in a 5-step mixer-settler equipment with an organic solution of kerosene containing the strong complexing agent EHPA (di-ethyl-hexyl-phosphoric acid), 10%, and TBP (tri-butyl-phosphate), 5%, to prevent the formation of an intermediate, third phase in the settlers.

The mixer-settler was made of acid resistant steel but was rapidly broken down by corrosion and had to be replaced by a new equipment made of glass fibre reinforced furan resin after a few months of operation. The reason was found to be the presence of tetrathion acid in the eluate, originally caused by polythionate producing processes in the shale during the storage before leaching.

From the organic solution, the uranium was re-extracted in a second mixer-settler with a sodium carbonate solution in three steps, where the uranium was extracted from the organic solution to the sodium carbonate solution as uranyl carbonate.

The uranium was precipitated as sodium uranate from the uranyl carbonate solution by the addition of sodium hydroxide. The precipitation was performed as a continuous process in a series of three heated tanks with a mixer and a bottom scraper. To get a heavy precipitate with good sedimentation and filtering characteristics the pH had to be kept in the range 12.5-12.9 and the temperature at 80°C. The precipitate was transferred to a thickening tank with a slow bottom scraper.

The sodium uranate concentrate was finally batch-wise produced in a double mantled, steam heated dryer tank, equipped with electric elements for heating the top layer, continuously scraped down into a central pipe, connected to a 200 l steel drum. The sodium uranate concentrate, "yellow cake", had a uranium content of about 71%.
Waste treatment and disposal

The barren solution from the ion exchange process was treated in a neutralisation plant equipped with 6 reaction tanks ("pachukas") in series, each taking about 100 m³ of solution. The tanks were made of steel and internally protected with a rubber lining and acid resistant brickwork. Lime-stone slime was added for the neutralisation, and pressurized air was used for agitation and for oxidation of the iron from Fe²⁺ to Fe³⁺.

For the treatment of miscellaneous acid waste solutions from the leaching, ion exchange and eluex processes, the neutralisation plant had two similar 40 m³ pachukas in series.

The resulting precipitates of gypsum, iron and aluminium hydroxide was pumped to the disposal area.

In connection with the neutralisation plant there was an installation for the treatment of waste water collected in the disposal area. The water was affected by pyrite weathering in the leaching residue and was contaminated mainly by iron sulphate, why it was treated by air oxidation of Fe²⁺ to Fe³⁺ and the addition of hydrated lime for neutralisation. A series of concrete chambers lined with acid resistant bricks and equipped with mixers were used for this treatment. After the addition of flocculation aid, the resulting gypsum and iron hydroxide was precipitated in sedimentation basins. The precipitate was pumped to the disposal area.

The solid waste products from the crushing, mineral dressing and leaching processes, together of the same quantity as the mine product, was transported by truck to the disposal area.

Figure 5-12 illustrates the arrangement of the disposal area and the system for the handling of contaminated and cleaned water.

All solid waste was transported by 35 t trucks, working together with a 25 t crawler tractor to successively fill the disposal area up to a roughly horizontal top surface in level with the top of the surrounding, permeable dams. The slimes from the neutralisation and the water treatment plant were pumped to lagoons surrounded by permeable dams of leaching residue, functioning as a
filter for the drainage water from the slimes. After drainage the slime lagoons were stabilized with coarse rock material from the mine.

Totally about 1.5 Mt of shale products, mainly leaching residue, and about 0.25 Mm³ of slimes were stored in the disposal area. The water seepage from the solid was collected in surrounding ditches and returned from the collecting pond (capacity 120 000 m³) to the water treatment plant described above. The cleaned water was flowing by gravity to the buffer pond (280 000 m³). From there the cleaned water was pumped about 5 km to the recipient river (Flian). The water quality in the recipient downstream of the discharge was defined by the Water Court to contain concentrations of less than 75 mg/1 Mg and 3 pCi/1 Ra. These limits were met already in the buffer pond.
However, during the operation period 1965-69 pyrite weathering in the waste shale was found to cause a problem to be considered in the development and planning of expanded production (see below) as well as for the restoration programme (see chapter 7).

Radiological conditions

The final steps in the uranium recovery process - precipitation, drying, packaging and storing of the yellow cake - were radiologically classified ("blue areas"). Special clothing and breathing protection devices had to be used by personnel handling the dry product. Personal dosimeters were used intermittently and health physicists from AEs facilities at Studsvik were engaged in monitoring and supervising the radiological work conditions. Dosimeter readings and results from uranium analyses on urine samples consistently indicated very low values. A maximum individual dose lower than 3 mSv/year was calculated for personnel working full time in the classified areas of an expanded plant (5-11).

Exposure to the personnel from external radiation and radon inhalation in other parts of the process were also considered (5-11). However, the low uranium content and the extreme tightness of the shale resulted in relatively low intensity of the external radiation and radon emanation. For example, the external radiation dose rate from the uranium shale in the open pit and on the leaching residue in the waste disposal area was about 2.5 μSv/h at 1 m above the surface, giving a maximum individual dose rate of about 2 mSv/y. The radon concentration in the air of the mineral dressing plant was about 110 Bq/m³ and directly above the surface of the leaching vats max 150 Bq/m³ (compare the present Swedish limit 70 Bq/m³ in new dwellings).

5.4 The utilization of the Ranstad plant

Chapter 6 outlines vital decisions for the Ranstad uranium project. Their effects on the practical operation of the Ranstad plant and on the technical development work are summarized in the following (5-12).

After the commissioning of the Ranstad plant in August 1965 the first fiscal year 1965/66 was a test period with the plant running at capacities 2/7 and 3/7 of the nominal one. AE reported the test period results to the Government in August 1966 (5-13).
Due to the uranium market conditions and the uranium requirement prognosis at the time (see chapter 6) it was decided to continue the production at 3/7 of the nominal capacity for the 3-year period 1966/67 to 1968/69.

For the 3-year period 1969/70 to 1971/72 it was decided to change the Ranstad plant operation from uranium production to process development aiming at lowering the costs for a future large-scale production. The background for this decision was an estimated considerable increase in domestic uranium demand in the 1980s, making the existing plant insufficient (5-14, 5-15).

In order to reduce the operating costs when using Ranstad as a pilot plant, the four 2000 t leaching vats were replaced by four 250 t, circular neoprene lined steel tanks, placed in one of the large vats. The new leaching installation was taken into production in the summer 1970. The personnel at the plant was reduced from 145 to 115 employees. Leaching could be performed in continuous operation but the mining and mineral dressing, as well as the uranium recovery installations were run only intermittently.

In a Government bill (5-16) early in 1972 the guide lines were given for a new development and investigation program during the 3-year-period 1972/73-1974/75, however, with a much reduced government financing. This was compensated for by a financial support from the State Power Board and technical assistance from the mining companies LKAB and Boliden AB.

The material flow was further reduced by the installation of a small pilot leaching system consisting of 8 columns made in neoprene lined steel, each taking 6 t of shale. This installation made it possible to test percolation leaching with up to 5 m bed height and more advanced leaching cycles. The personnel was cut down once more, now to 70 employees. The new installation was taken into operation at the beginning of 1973, when the 250 tonne system was decommissioned.

In the directives for the 3-year programme it was included that AE should report the development results, present a basis for the decisions to be taken for the future uranium production activities, and to identify suitable partners if an expanded production at Ranstad would be proposed. AE delivered this report (5-20) in 1974, outlining a
plant at Ranstad for the production of 1275 t uranium per year. As a result of the development work the project included a number of important technical improvements, which in combination with the larger capacity indicated considerably lower uranium production costs. This report was the basis for the production projects launched in the 1970's and the intensified engagement by the mining companies and the nuclear fuel consumers (see section 5.6).

5.5 Technical function and economy

The experience from the production period 1965-1969 was generally in accordance with the project ed plant specifications (5-12, 5-17).

During the most representative period 1966/67 - 1968/69 with 3/7 capacity operation, the uranium production was on an average 54 t/y, corresponding to 127 t/y at full capacity.

The average material yield was 74%, with losses:
- fines from coarse crushing 5%
- slimes (heavy media sep) 4%
- sink product ("""") 9%
- dust from fine crushing 8%

The uranium content in the leaching material was on average 306 g/t of which 95 g/t was lost in the leaching residue, giving a leaching yield of 68.9%. The loss in the uranium recovery was 1.2%, and thus the total chemical yield was 67.7%.

Some difficulties experienced during the test and production period have been mentioned in the plant description. Most important were the function of the open storage, which did not give the expected favorable effects on the leaching conditions, and the pyrite weathering occurring in the disposal area. Other problems were the transport of dust from the electro-filter, where the truck transport had to be replaced by pumping to the disposal area, and the clogging of screens, eventually remediated by heating.

The operating cost (excluding investment and capital costs) at 3/7 capacity during 1965/66 - 1968/69 was on an average $13 per lb U₃O₈, a quite reasonable value in comparison with the expected corresponding operating cost of $10-11 per lb at 7/7 capacity (5-18). However, the US AEC purchase price for domestic uranium within the "strech-out" programme 1967-1970 was $8 per lb U₃O₈ and the price for foreign uranium was about $10.5 per lb (5-18).
5.6 Process development and investigations at Ranstad in 1965-1975

From the start of the Ranstad plant, and especially after 1969, the interest was gradually changing from the immediate uranium production in the existing plant to the possibilities of large scale utilization of the shale deposit at Ranstad at a later time. This resulted in very extensive process development efforts and investigations during the period 1965-1974, leading to the projects for re-opening of the production in 1975-1980 (see section 5.8).

For the process development the existing plant was an extremely valuable resource, giving possibility to test practically all parameters and unit operations with correctly prepared material, at realistic operation conditions and on a sufficiently large scale. The installation of new, smaller size leaching equipment, mentioned in section 5.5, gave increased flexibility for the experimental work and reduced the operating costs considerably.

Also the personnel resources were available, because many of the specialists from the design and construction period of the Ranstad plant were still engaged in the local staff and elsewhere in the AE organisation, especially at the Vinterviken laboratory, where a group was still working with chemical and thermal experiments on shale.

In the local Ranstad organisation the laboratory was of great importance, having capacity to perform practically all necessary analysis on solid materials and solutions from plant operation as well as from experimental activities.

In the efforts to reduce the uranium production costs the main interest was until 1975 focused on improvements of the existing "AE-process", described in section 5.3. However, the possibilities to produce by-products were also considered, especially to produce sulphuric acid and building material by burning the shale, before or after the leaching (5-13, 5-18).

Experiments were also performed on heap leaching and bacterial leaching instead of percolation leaching in vats. The results were quite promising on laboratory scale, but when tested on the 2000 tonne scale the process proved to be too slow and unreliable for practical utilisation (5-8).
The results of the process development are included in AE reports to the Government in 1968 (5-14, 5-15), 1971 (5-19) and 1974 (5-20).

In the 1968 report, AE presented a study of alternative sizes of the plant at Ranstad, with capacities up to 1800 t/year uranium and with an improved process. The following table, derived from the report, illustrates the importance of the process improvements as well as the scale of production.

<table>
<thead>
<tr>
<th>Option</th>
<th>Uranium production t/year U</th>
<th>Total cost $/lb oxide</th>
</tr>
</thead>
<tbody>
<tr>
<td>Actual plant</td>
<td>134</td>
<td>20.1</td>
</tr>
<tr>
<td>4 x actual plant</td>
<td>550</td>
<td>13.6</td>
</tr>
<tr>
<td>Improved process</td>
<td>600</td>
<td>9.4</td>
</tr>
<tr>
<td></td>
<td>1200</td>
<td>8.9</td>
</tr>
<tr>
<td></td>
<td>1800</td>
<td>9.1</td>
</tr>
</tbody>
</table>

The anticipated process improvements included increased yield in shale preparation and in chemical treatment. For the options 1200 and 1800 t/y U underground mining and production of sulphuric acid were assumed.

The process development and investigation programmes for the following periods, 1969/70 - 1971/72 and 1972/73 - 1974/74, were focused on the specification of a plant with a capacity of 1200-1300 t/y U. As described in section 5.4, the experimental work was based on new, smaller installations for leaching. The interim report delivered by AE in 1971 (5-19) presents a plant with the following main characteristics:

- Mine production: 6 Mt/y
- Material yield in the mineral dressing: 89.3 %
- Leaching material: 5.4 Mt/y
- Chemical yield: 76 %
- Uranium production in the form of sodium uranate: 1250 t/y
- Start of production, lead time: 5.5 years
As a result of the interim report 1971 AE and the State Power Board (SPB) agreed on sharing the financing and leadership of the programme 1972/73-1974/75, to clarify the possibilities of a large scale uranium production at Ranstad. One additional factor in the SPB’s interest in the Ranstad project might have been the possibility to use underground rooms after mining as the lower basin in a pumping hydro-electric plant for peak loads. LKAB and Boliden AB agreed to give support in special areas of the programme (5-8).

The results of the continued process development and investigations were presented in AE’s report to the government in 1974 (5-20). This report had a special importance as the base for the following uranium production projects. The report outlines a Ranstad plant with technical specifications similar to those presented in the interim report:

- **Mine production**: 6.0 Mt/y
  - in open pit (declining): 3-0 "
  - underground (increasing): 3-6 "
- **Material yield in the mineral dressing**: 90 %
- **Leaching material, production**: 5.4 Mt/y
- **Type of leaching material**
  - max particle size: 2 mm
  - CaO-content: 1%
  - Uranium content: 300 g/t
- **Leaching yield**: 79 %
- **Solvent extraction, yield**: 99.7 %
- **Sulphuric acid consumption** (of shale weight): 6.2 %
- **Uranium production**: 1275 t/y
- **Life time of the plant**: 25 years
- **Start of production, lead time**: 5.5 years

The total production cost was calculated as 166 SEK/kg U (appr 13 $/lb oxide at 5SEK/$). The NUEXCO market price for uranium delivered 1974 was 8.2 $/lb oxide with a rising trend, estimated to give 17.7 $/lb for 1983 delivery.

The very important reduction of losses in the mineral dressing and increased yield in the chemical processes compared with the performance of the original Ranstad plant of 1965 (see section 5.5) were based on results of the process development. The 90% material yield was attained by the introduction of hydrocyclones for heavy media separation of limestone and by the recovery of dust.
Fig. 5-13. Shale preparation flow sheet presented in the 1974 report.

from the electro-filter by pelletizing with a sulphuric acid solution as illustrated in the flow sheet for the shale preparation, figure 5-13.

The higher uranium yield in the leaching was achieved by a finer grained feed material (-2 mm instead of -3 mm) and a longer leaching time (using 7 leaching vats instead of 4). The addition of sulphuric acid solution to the final product from the dressing plant resulted in a lower CaO-content and a "curing" effect, favorable for the percolation in the leaching vats and thus
contributing to a good leaching yield. The dust pellets were proved to sustain the leaching without clogging effects and to give a high uranium yield.

For the recovery of uranium from the leaching solution the ion exchange-ELUBEX process was replaced by a direct solvent extraction process, figure 5-14. The new process was tested in a pilot plant, capable of treating the solutions from the 5 t/d leaching plant.

![Flow sheet for solvent extraction of uranium](image)

Extensive investigations and studies were necessary for the anticipated large scale operation. SGU was engaged for the investigation of the complicated groundwater conditions in the area (5-21, 5-22). The field work included drilling and seismic testing to clarify the stratigraphy and the water levels and qualities in the different aquifers. The handling and disposal of waste products were studied in co-operation with VBB. The open pit was planned for the disposal of leaching residue and the underground mine for the disposal of slimes from neutralization (5-23).

Various types of covering systems and revegetation of the leaching residues were tested in different scales. Of special importance was a large scale and long time experiment started in 1972. 15000 tonnes of leaching residue were laid out in the disposal area after homogeneous addition of fine-ground limestone. The heap was covered by two different types of sealing material (moraine with
bentonite and crushed limestone, respectively), protected by a layer of moraine and top soil. To monitor the flux and quality of the infiltrating water there were sampling pans installed underneath and inside the heap at different levels. The results from this experiment were of major importance for the final restoration of the disposal area (chapter 7).

5.7 Projects to re-open uranium production

The 1974 report illustrated the possibilities to produce uranium in large scale with the advanced AE-process and initiated the projects during the period 1975-1980 to re-open production at Ranstad. The process development in the existing facilities continued in parallel with the project activities. However, the efforts were eventually focused on the elimination of environmental impact and on the possibilities to recover also other useful products from the shale.

**Project Ranstad 75**

In January 1975 LKAB, AE and SPB agreed to cooperate, with LKAB as the leading partner, in "Project Ranstad 75" for a large-scale exploitation of the uranium resources in the Ranstad area. LKAB was granted a governmental loan for the period 1975/76-1976/77, to be repaid if a commercial production would be established.

An application for a siting permit was presented to the government in June 1975 (5-24) describing a plant for a uranium production of 1275 tonnes per year from 6 Mt of shale, mined in open pit and underground. For the anticipated lifetime of 25 years an area of 25 km² had to be mined, mainly underground.

The proposed process (figure 5-15) was virtually the same as described in the 1974 report. The leaching plant illustrates the size of the process installations: A 500 m long construction with a line of 7 neoprene rubber protected steel vats, measuring 60x45x6.5 m, each with a capacity of 15 500 tonnes of shale.

The environmental impact description outlined in the application was supported by further investigations and studies of the hydrogeological conditions (5-26) and the disposal of waste products (5-27).
The application was submitted for consideration to a large number of authorities and interested parties. The judgements from central authorities were mainly in favour of the project, while the local parties were mostly critical for environmental reasons. The local debate on all aspects of the project was further intensified by plans for a limestone quarry and an artillery and rifle range in the same area (5-28). Before the local communities had decided whether to use their right to veto the project or not, LKAB withdrew the siting application in November 1975 in order to introduce technical improvements, especially to reduce the amount of soluble compounds in the neutralization slimes before their disposal. The intention was to renew the siting application in late 1976. A step-wise neutralization method was developed and tested with good results in a new 5 t/d-scale set up in 1975-76. This method included evaporation of waste solutions to give a fertilizer product of potassium and magnesium sulphate. Processes for the production of sodium sulphate and molybdenum oxide were also tested. These improvements were considered possible to introduce in a large plant.

The production was estimated to be economically interesting at a uranium price of 31 $/lb uranium oxide, approximately corresponding to the NUXCO
1976 market price. The sale of the production, roughly filling the demand for the decided Swedish reactor program, was preliminarily negociated with the SKBF.

However, after the parliament election in the autumn 1976, the new government declared the large-scale uranium production to be inconsistent with its nuclear policy and demanded that the project should be focused on the recovery of other products from shale resources. Project Ranstad 75 was closed and a final report was presented to the government in March 1977 (5-29). Other descriptions of Project Ranstad 75 are given in ref 5-8, 5-30, 5-31.

Mineral Project Ranstad

In late 1976 LKAB presented "Mineral Project Ranstad", a combined production-research facility at Ranstad. The purpose was to clarify the possibilities to recover the valuable constituents of the alum shale ("full recovery") for the long term Swedish raw material supply.

Fig. 5-16. The AE-process for Mineral Project Ranstad
LKAB applied for a siting permit in May 1977 (5-32). A production plant for 200 t/y uranium from 1 Mt shale was proposed as economical and technical support to the planned research and development facilities. An up-dating of the existing Ranstad plant was intended to include the improvements of the AE-process, described above, see figure 5-16.

Only open-pit mining was planned, however, integrated with the disposal of the leaching residue, see figure 5-17. The residue is here returned to its original place. Eventually the ground water (5-25) will resume a level slightly higher than before mining, thus flooding the leaching residue ("wet disposal") and effectively preventing pyrite weathering by cutting off the necessary oxygen supply.

The R&D-facilities included the existing and new equipment of laboratory and pilot scale for strong acid leaching, solvent extraction and gasification/burning of shale or leaching residue. Several ideas for "full recovery" were tested, for example the process illustrated in figure 5-18.
In the autumn of 1976 the local communities vetoed the project Ranstad and LKAB documented the work performed in a pre-project report to the government (5-33).

Project Ranstad 1978-81

The production of uranium at Ranstad had once again been stopped, but the utilization of the alum shale resources was still considered important. The industry's interest was demonstrated by the formation of the R&D company ASA (AB Svensk Alunskifferutveckling), owned 50/50 by LKAB and Boliden AB, in the autumn of 1977. The government ordered a general investigation of the alum shale resources (5-34) and of the conditions in the Ranstad area (5-35).

For the period 1978/79 - 1980/81 ASA was granted a loan of 128 MSEK by the government for continuation of the R&D on the alum shale, primarily in the Ranstad area but also in the Kvarntorp area and elsewhere. In 1978 RSA (Ranstad Skifferaktiebolag), a separate daughter company to ASA, was established, owned by ASA (60%), LKAB (20%) and Studsvik Energiteknik AB, the former AB Atomenergi (20%). RSA took over the facilities and personnel from Studsvik Energiteknik AB.
The support from a production plant was still considered necessary for the long-term and costly development of a commercial "full recovery" process. Project Ranstad 1979/81 therefore included the planning of a production plant for 470 t/y of uranium from 2 Mt/y of shale, using the improved AE-process in the existing plant, enlarged to the double capacity.

The process development was continued in laboratory and pilot plant scale, now concentrated on a process ("DMS") with strong acid leaching of the shale before the thermal treatment.

A full documentation for a siting license application was produced to be filed at mid 1980. However, negotiations with SKBF for the sale of the uranium production indicated that the now dropping market prices made a viable production at Ranstad impossible. In July 1980 RSA therefore decided to stop the plans to re-open uranium production. The staff was reduced from 70 to about 20 persons to maintain the facilities and to finish the R&D programme by mid 1981. Project Ranstad 1978/81 was summarized in a report to the government in March 1982 (5-36).
5.8 Concluding comments

The government cost for the activities related to uranium production at Ranstad can be summarized as follows (current value of money):

<table>
<thead>
<tr>
<th>Period</th>
<th>MSEK</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>1960-65</td>
<td>145</td>
<td>Investment in the Ranstad Plant (AE)</td>
</tr>
<tr>
<td>1965/66-1974/75</td>
<td>90</td>
<td>Operation and R&amp;D at Ranstad (AE) (incl 10 MSEK from SPB)</td>
</tr>
<tr>
<td>1975/76-1977/78</td>
<td>65</td>
<td>Loan to LKAB for projects and R&amp;D</td>
</tr>
<tr>
<td>1798/79-1980/81</td>
<td>128</td>
<td>Loan to ASA for projects and R&amp;D</td>
</tr>
<tr>
<td></td>
<td>-26</td>
<td>20% repayment</td>
</tr>
<tr>
<td>Total</td>
<td>402</td>
<td></td>
</tr>
</tbody>
</table>

The uranium produced until 1977 amounted to 212.5 tonnes, delivered to SPB (15.8 t, 1970) and to SKBF (196.7 t, 1977).

In spite of the very low uranium content in the Swedish alum shale resources, the AE-process for a long period made a production at Ranstad interesting for the domestic uranium supply. However, the steep drop in market prices from the late 70's and onwards, with production from 10-100 times richer ores abroad, finally made the exploitation at Ranstad economically impossible, probably for a long time into the future.
REFERENCES

5-1 hörmander, O
Investigation of the expansion of the shale based uranium industry. (In Swedish).
AB Atomenergi, July 1957

5-2 Hörmander, O
Ranstad is planning for start of operation.
(In Swedish)
Reaktorn, May 1964

5-3 Peterson, A
Ranstad - a new uranium processing plant.
Processing of low-grade uranium ores. Proceedings of a panel, Vienna, 27 June - 1 July 1966
IAEA, Vienna 1967

5-4 Hörmander, O., Forsgren, P., Strandell, E.
Uranium recovery from shale - process description. (In Swedish).
Teknisk Tidskrift, 10 May, 1961.

5-5 Hörmander, O., Sundström, E., Fredriksson, F., Olsson, G.
The Ranstad works for the recovery of uranium from shale.
(In Swedish).
Teknisk Tidskrift, 10 May, 1961.

5-6 Bergström, W.
AB Atomenergi 1965.

5-8 Strandell, E.
Ranstad - 35 years development of processes for the recovery of uranium from shales of Middle Sweden. (In Swedish). To be published.

5-9 Olsson, G.
Ranstad - a new opencast mine.
Symposium on opencast mining, quarrying and alluvial mining.

5-10 Janelid, I., Olsson, G.
The Janol method - a new mining concept.

5-11 Project Ranstad
Description of radiological conditions.
PM in support of license application under the Atomic Energy Law. (In Swedish, not submitted).
5-12 Olsson, G.
Summary of operation results at the Ranstad works during the period 1 July 1966 to 1 July 1969. (In Swedish).
AB Atomenergi TPM-IR-486, 1971

5-13 The Ranstad works.
Report on background, construction, results of test operation, and different alternatives for continued activity. (In Swedish).
AB Atomenergi, August 1967

5-14 Uranium from Swedish shale.
Investigation by AB Atomenergi on large-scale recovery of uranium from shales in the Billingen area. (In Swedish).
AB Atomenergi, September 1968.

5-15 Uranium from Swedish shale.
Complementary report to the investigation issued in September 1968. (In Swedish).
AB Atomenergi, November 1968.

5-16 Government Bill 1972:57

5-17 Nöjd, L.-Å., Peterson, A., Svenke, E.
AB Atomenergi 1967.

5-18 Önnesjö, K.E.
The Ranstad investigation. (In Swedish).
Department of Finance 1967:4.

5-19 Development work on the recovery of uranium, status report June 1971. (In Swedish).
AB Atomenergi, 1971

5-20 The Ranstad project
AB Atomenergi, 1974

5-21 Ground water investigations in the Ranstad area and programme proposal. (In Swedish).
SGU, 30 January 1973

5-22 Hydrogeological views on planned mining of uranium shale in open-pit and underground at Ranstad, and of limestone at Rådene, county of Skaraborg. (In Swedish).
SGU, July 1974

5-23 The Ranstad works
Management of residual products, main alternative. (In Swedish).
VBB, November 1974
5-24 Siting application according to Section 136a of the Building Law for uranium production etc. at Ranstad, county of Skaraborg. (In Swedish). LKAB 13 June 1975

5-25 Project Ranstad
Hydrogeological views on planned uranium recovery in Billingen at Ranstad, county of Skaraborg. (In Swedish). SGU, March 1977

5-26 Ground water and uranium shale mining at Ranstad.
Summarizing views. (In Swedish).
SGU May 1975

5-27 Ranstad 75
Management of residual products. (In Swedish).
VBB, May 1975

5-28 The South of Billingen
The communities of Falköping, Skara, Skövde. (In Swedish).
May 1975

5-29 Project Ranstad 75
Mining of 6 Mt shale per year, pre-project. Summarizing part. (In Swedish)
LKAB, State Power Board, AB Atomenergi March 1977

5-30 Andersson, A., Olsson, G.
Uranium recovery from Swedish low grade bituminous shales Nuclear Engineering International, 1975

5-31 Olsson, G.
Environmental aspects of the Ranstad expansion. (In Swedish).
Swedish Mine Association, Communication No. 144, Vol.9. 25 November 1975

5-32 Mineral Project Ranstad
License application according to Section 136a of the Building Law for activity at the Ranstad Works in the communities of Skövde and Falköping, county of Skaraborg. (In Swedish).
LKAB, 16 May 1977

5-33 Mineral Project Ranstad
Production and research plant for the mining of 1 Mt of shale per year. Pre-project. (In Swedish).
LKAB, State Power Board, AB Atomenergi, 1978

5-34 Alum Shale. (In Swedish).
SIND PM 1978:2,3
State Industry Authority, 1978
5-35 Billingen - 4 examples
SOU 1977:47

5-36 Project Ranstad 1978/81
Production and research plant for the mining of 2 Mt of shale per year, Pre-project. (In Swedish).
AB Svensk Alunskifferutveckling, Ranstad Skifferaktiebolag, March 1982
6 URANIUM SUPPLY POLICY

The policy in uranium supply has been intimately tied to considerations in the fields of overall energy and atomic energy policies. To understand the changes in uranium supply policy, it has to be viewed within the broader perspective of developments in these fields. This has been our ambition in this chapter.

6.1 Independence and self support

The Atomic Energy Committee, formed in 1945, considered the utilization of atomic energy so vital to the future of the country and the efforts necessary to be of such magnitude, that the State should carry the main responsibility for the activity. At the same time the committee emphasized the importance of an intimate cooperation with private enterprise. Thus the Atomic Energy Company, AB Atomenergi (AE), was formed at the end of 1947 as a share-holding company with 4/7 of its share capital owned by the State and 3/7 by 24 different enterprises, primarily private or community owned power utilities and a number of industries in the areas of mining, iron and steel, and engineering.

As uranium had the character of a strategic raw material it was clear that a Swedish atomic energy programme had to be based on domestic resources. The important task for AE was therefore, besides basic physical and chemical R&D, to take a closer look at the Swedish uranium resources at Kvarntorp and at Billingen. The low uranium contents implied, however, large technical and economical problems. The pilot uranium extraction plant at Kvarntorp, built in the early 1950's, was therefore essential to clarify the prerequisites for recovery of uranium from shale. Also the further stages of uranium treatment to finished fuel elements were studied (6-1).

So, by the middle of the 1950's, a level of resources and competence had been established in Sweden. The choice of reactor type, with its consequences for demand of uranium and fuel cycle services, was still open, however.

At the first United Nations conference in Geneva on the Peaceful Uses of Atomic (Nuclear) Energy (PUNE), in 1955, much of earlier classified information was released by the great power countries.
A very optimistic sentiment as to a rapid introduction of nuclear energy on a commercial scale surrounded the conference. This was a great stimulus to the Swedish programme. The second PUNE Conference in Geneva, in 1958, was attended by more than 200 people from Sweden engaged in the field.

The 1956 atomic energy decision

In December 1955 a committee was appointed by the Minister of Commerce and charged with the task to propose a programme for the continued work in the atomic energy area. The committee reported already after 3 months and presented guide lines for a wide Swedish development programme, including reactors, fuel, and organisation. AB Atomenergi should continue to have responsibility for practical development work. To support a forceful State leadership of the programme, the committee proposed an advisory delegation on atomic energy matters and drafted an atomic energy law. The law would require a license to possess atomic fuel and to construct and operate reactors and atomic energy plants. These guide lines were to be decisive for the Swedish atomic energy programme over several years to come.

As the possibility to import enriched uranium in large quantities was considered extremely uncertain, it was felt that domestic natural uranium should form the basis for the Swedish programme. The reactor programme proposal was thus focused on heavy water moderated and natural uranium fuelled reactors, for district heating and, later, for electricity production. The raw materials programme aimed at the possibility of self sufficiency in fuel supply. Uranium production should expand to cover the needs for the next ten years at least. Fuel elements should be produced at a central plant and R&D started on the use of plutonium enriched fuel. Domestic plants, for the separation of plutonium through reprocessing of spent fuel, and for production of heavy water, were foreseen.

The uranium production capacity at Kvarntorp was 5 tonnes per year in 1955. The uranium required to supply the reactor programme was estimated to reach a couple of hundred tonnes per year at the end of the 1960's. A new plant for 75-100 tonnes per year should therefore be commissioned. A doubling of the capacity at Kvarntorp to 10 tonnes per year should be realized and measures taken to increase the uranium treatment capacity at the Stockholm factory.
The Minister of Commerce supported in all essentials the committee proposal in his Bill to the parliament. Referring to the severe disturbances in oil import from the Suez crisis he emphasized the Swedish dependence on a secure energy supply and the desirable decrease of fuel import.

The committee proposals had been generally agreed upon regarding the reactor and fuel programme in the remittance process but strongly criticized as to the leading role given to the State through AB Atomenergi. However, as both AB Atomenergi and a parliamentary law committee strongly emphasized the importance of a close cooperation between AB Atomenergi and other interested industries, which, to a certain extent, would be free to cooperate also with foreign industries, the government bill (6-2), based on the committee proposals, was approved in the spring of 1956 by the parliament without debate. The parliament decided also to introduce a special tax on energy to cover i.a. costs for the atomic energy programme, including the utilization of domestic energy resources.

The 1956 year guidelines for the continued development implied greatly increased funding to AB Atomenergi. Its share capital was increased to 14 MSEK with an unchanged ratio between the State and the other share holders, who now increased their number to about 70.

The 1958 Government bill on atomic energy included the proposal from AB Atomenergi to construct a uranium recovery plant for 120 tonnes per year to continue the aim of self-support in atomic fuel supply (6-3). It noted that the uranium plant and the first nuclear power station (the Ägesta reactor, for production of heat and power) were not only so far the largest tasks in the Swedish atomic energy programme but also represented the most important stages of a programme for an active Swedish energy policy.

The 1958 bill also introduced special means for Swedish participation in international atomic energy cooperation, mainly for work within the International Atomic Agency, IAEA, and the European Nuclear Energy Agency, ENEA, formed by the Organisation of European Economic Cooperation, OEEC (today OECD-NEA). A body for Nordic cooperation in the atomic energy field, NKA, had also been created at this time.
The military connection

As indicated in the introduction of this report, the military aspect of nuclear energy was an important component of the initial Swedish interest in this field, which should not be forgotten in a discussion of the nuclear energy development in the country. Sweden was towards the end of the 1950's, according to the American Academy of Arts and Science, one of twelve countries with scientific, technical, and economic prerequisites to produce nuclear weapons. A very intense debate raged in Sweden at this time: should Sweden develop nuclear weapons or abstain?

The chief commander presented in 1957 five alternative proposals for a new defence order, four of which included nuclear weapons. In the fall of 1958 he demanded funding to start the R&D necessary for their production. The governing social-democratic party was severely split on the nuclear weapons issue, and only ten days later the Swedish foreign minister declared in a United Nations committee that Sweden supported a general nuclear weapons test ban, and he denied any preparations in Sweden aiming at the acquisition of nuclear weapons. The parliament voted to allow a fairly wide research on the protection against nuclear weapons by FOA: "to perform research in the atomic field and to study different factors of importance for the effect of the atomic weapon and to develop methods and equipment required for the protection against atomic weapons". It also voted to postpone a decision as to the funding of R&D aiming at their production. However, both the two large reactor projects under development in the 1960's, the Ägesta and the Marviken heavy water reactors, were designed to allow on-load discharge of low burn-up fuel, containing weapons grade plutonium (6-4, 6-5, 6-6).

On 5 August, 1963, a treaty on a limited nuclear weapons test ban was signed by the United States, the Soviet Union, and Great Britain, and declared open for accession by all States wishing to do so. The Foreign Affairs Committee discussed this issue on 6 August. All political parties agreed that Sweden should accede to the treaty, and the parliament voted accordingly on 20 November, 1963.
During the second half of the 1960's it became increasingly clear that the political majority was in favour of renouncing any ambition to acquire nuclear weapons. The Non-Proliferation Treaty was approved by the United Nations General Assembly 12 June, 1968, and went into force 5 March, 1970. Sweden signed NPT on 19 August, 1968, and ratified the Treaty on 9 January, 1970 (6-7).

6.2 From self sufficiency to trade in the market

The 1956 atomic energy programme thus relied on the use of domestic resources for a competitive supply of reactors and fuel to a domestic market expected to expand fairly rapidly. For the fuel cycle all stages would be covered by domestic plants. To the policy of independence in energy was added also the ambition to create a basis for an independent industrial activity.

The choice of reactor type

Three large projects dominated the scene by the middle of the 1960's: the Ågesta heat/power reactor, the Marviken 200 MWe power reactor, and the Ranstad uranium plant. The choice of the heavy water/natural uranium type of reactors was confirmed by a detailed technical evaluation by AE after the US had offered reduced prices of enriched uranium in 1957.

The Ågesta reactor originally produced 10 MWe and 55 MW heat, which was later increased to 12 MWe and 80 MW heat. Its construction and operation gave the necessary experience but also clearly indicated, that there were no prerequisites for this type of reactors. From 1959 the development was thus focused on large electricity producing reactors.

For the Marviken heavy water project different basic designs and design features were studied by AE, the State Power Board and ASEA, including pressure tank reactor (PHWR), boiling water reactor (BHWR) with nuclear superheat, natural uranium fuel and low enriched fuel. The originally in 1960 decided PHWR-design (pressure tank) was changed in 1963 to a BHWR-design. A number of serious uncertainties severely impeded the project, both of general character as to the choice between LWR...
and HWR, and regarding details of the Marviken design, i.a. unfavourable reactivity control characteristics with necessary redesigns of the shut down and emergency cooling systems and of the fuel elements. In April 1970, AE finally proposed to close down the Marviken project. The Government and the parliament decided to this end.

The reactor development in the United States was early focused on light water reactors (LWR) with pressurized (PWR) or boiling (BWR) light water. The 60 MWe Shippingport was started in 1957, the Dresden 210 MWe reactor in 1960, and several LWRs of 500 MWe ordered in 1963. The discussion in Sweden on our reactor programme was strongly influenced by the success of the American LWR technology. The Swedish power industry now considered a step directly to a full scale nuclear power station as commercially realistic. Based on an offer by ASEA in March, 1965, a new Swedish utility, the Oskarshamn Power Group (OKG), ordered the first LWR in Sweden, a BWR of 400 MWe, in July 1965.

The US had at this time declared its willingness to enter into long term agreements to supply other countries with both natural and enriched uranium. Sweden signed such an agreement in June, 1966, for a period of 30 years. It included a US commitment to meet the need of fuel supply for the first six Swedish power reactors over that time. Sweden guaranteed use for only peaceful purposes of the supplied fuel, and the US was granted certain inspection rights, assuming these would later be transferred to the IAEA. This was the start of the exclusively LWR programme, that we have today.

The Ranstad uranium project

The decision to construct the Ranstad plant was based mainly on the importance attached to the possibility of domestic supply, and little weight had been given to the aspect of competitive production costs. However, improved prospects of uranium import, declining world market prices, and delays in the reactor programme decreased the urgency of a domestic uranium production.

In the spring of 1959, AE investigated the issue of uranium import. The costs of the Ranstad uranium were found to be 70% higher than uranium imported from i.a. the US. If Swedish authorities were prepared to accept foreign or international
control, it would thus be of advantage to import the uranium needed during the 1960's. A continued decline of world market prices in 1960 to 6-8 US$ per pound uranium oxide increased this difference still further. Operational costs alone for Ranstad were around 9 US$/lb oxide (120 SEK/kg U). AE proposed that this difference should be treated as a cost for national preparedness and covered by means from the State. The government and the parliament accepted the proposed principle.

After several periods of delay due to political decisions the Ranstad plant was completed in the summer of 1965 at a total cost of 145 MSEK. However, because of the evident interest from Swedish utilities in LWRs, demonstrated by their order for a 400 MWe plant, and the US readiness to supply enriched uranium for their operation, it no longer appeared as likely as before that the future Swedish reactor programme would be based on natural uranium. The future operation of the Ranstad plant became uncertain.

Test operation of the plant in 1965-66 was technically satisfactory, but indicated substantial economic losses. AE proposed a more comprehensive utilization of the shale through its incineration with the production of sulfuric acid, electricity, and building materials. A 3-year programme with R&D connected to a limited plant operation was proposed. The economic and market conditions for such an extension were investigated and reported. The report (6-8) concluded, that the operation of the plant had so far been technically satisfactory, but that a change in direction of production to include the products mentioned did not have the potential to make the production profitable. A continued operation was motivated only if part of the future Swedish uranium supply was for some reason to be based on large scale domestic production. Because of the rapidly expanding nuclear energy programmes abroad at that time, and the uncertainty around the international uranium market the report strongly emphasized the need of guide lines for a thoroughly prepared long term uranium supply policy.

The minister in charge accepted the conclusions of the report and proposed to the parliament, that the conditions for a future full scale commercial exploitation of the Billingen shale deposit
should be clarified. However, also other possibilities, such as uranium import through long term contracts, should be considered.

AE reported in October, 1968 (6-9) on the technical and economic conditions for such an exploitation, and concluded that an increased domestic uranium supply from shale would in all probability be desirable during the 1980s. It recommended a risk distribution, including:

- preparation of a basis for decision in 1974 on extension of the Ranstad plant to 600 tonnes per year with the possibility of an increase to 1200 tonnes per year
- supply for 1975-1980 by import
- increased efforts in organized uranium exploration

A development programme was also proposed to support planning of the extension.

A complementary report from the department of industry (6-10) gave a general account of the Swedish atomic fuel supply. It agreed, that a large scale domestic uranium recovery from shale could be actual in the 1980's. A more limited development programme was deemed sufficient to improve the process and to reduce production costs. A successive liquidation of uranium production during the 3-year period was anticipated.

The conclusions approved by the parliament included a view of Ranstad as a possible supply alternative for the future. The near term reactor programme could be supplied by uranium import. Further decisions on Ranstad should be taken at the end of the period and based on the developments in the uranium market.

At the beginning of the 1970's a steep increase in the demand of uranium was expected with the then anticipated expansion of nuclear power programmes. This affected also the decisions taken in Sweden.

A group named "Project Ranstad 75" was formed in 1975 by the mining company LKAB, AE, and the State Power Board (SPB). Its objective was large-scale exploitation of Swedish shale, using the mining and processing technologies developed by AE. The long term goal was to supply total uranium requirements for a 13-reactor programme approved by the parliament in 1975. Plans included the mining
of 6 million tonnes of shale per year for 25 years with an annual production of 1300 tonnes of uranium. The ambition was also to recover the shale contents of aluminium, vanadium, molybdenum, and the fertilizer elements nitrogen, potassium and phosphorus, and to utilize the energy contents of the shale by combustion or gasification (6-12,6-13).

An investigation, initiated in 1976 by the department of industry on the regional effects of four alternatives for the exploitation of the natural resources of the Billingen mountain, reported in July 1977. The four alternatives included the decommissioning of the Ranstad plant and three levels of plant operation, up to the mining of six million tonnes of shale per year. Only this scale of operation had any marked effects on employment and the need of social service in the area (6-14).

The shift of government in 1976 again changed the direction of the Ranstad programme to a 3-year R&D programme, after also the municipalities in the Ranstad area had used their veto right to stop further exploitation.

The new R&D programme included environmental improvements in the disposal of the leach residues with a minimum of weathering and radon release, and to start a project for the recultivation of the mining areas exploited.

The outcome of the national referendum in 1980, and the following decision by the parliament to conclude the use of nuclear power by year 2010, decreased still further the motives for a domestic uranium production. The change in world market conditions with cuts of nuclear power programmes and low prices of uranium also contributed to a decision finally to conclude the Ranstad project with a complete restoration of the area satisfying high environmental standards. This is described in more detail in chapter 8.

6.3 The formation of a fuel supply company

Going back to the early 1970's, the department of industry published in 1971 a comprehensive analysis of Sweden's nuclear fuel supply up to year 1990. A main reason was the decision required as to continued development of domestic uranium resources, including the Ranstad programme (6-14).
Nuclear power was in 1971 forecast to supply one third of Swedish electricity consumption in 1980, and about one half in 1985. A long term planning programme should be started.

One conclusion of the analysis was that the security alone of nuclear fuel supply was not a sufficient reason to expand the Ranstad plant at the beginning of the 1980's. However, for the second half of the 1980's substantial price increases and uranium scarcity could not be excluded, which could seriously disturb our electricity production. A reasonable goal in energy supply policy would thus include preparations, so that a complete uranium production plant could be ready around 1985. It thus suggested, that AE, in parallel with negotiations on commercial exploitation, elaborate a programme to this end. It was also suggested that exploration work for new uranium deposits continue.

In the areas of uranium enrichment and spent fuel reprocessing it was not excluded, that domestic plants could be motivated in the future. Nuclear power was also analyzed as to its sustainability in a situation of isolation from foreign supply and found to offer possibilities of extended operation.

In view of the necessity to coordinate continued efforts and investigations in the nuclear fuel area it was finally suggested, that a new organization be created for this purpose. Such a body should be a share holding company, jointly owned by the state and the utilities. It should have responsibility for various measures to secure the nuclear fuel supply and for the successive development of the nuclear fuel programme within the general guidelines given by the state and thus satisfy both the supply security interest of the society and the commercial interests of the utilities.

These proposals were included in a bill to the parliament in the spring of 1972. After parliament approval and an agreement between the State Power Board, Sydkraft AB (the Southern Power Co.), and Oskarshamnsverkets Kraftgrupp AB (today OKG AB), the Swedish Nuclear Fuel Supply Co., SKBF, was formed on 26 September 1972. The group of shareholders was later, in 1981, enlarged to include also the Forsmark Kraftgrupp AB, FKA (6-15).
The most important tasks of the company were initially at the front end of the fuel cycle: the procurement of natural uranium and enrichment services. With the changes in political conditions for nuclear power in the 1970's, the main responsibility became focused on the back end: the development, planning, construction, and operation of facilities for the management and disposal of spent nuclear fuel and radioactive waste from Swedish nuclear power stations. Its name was consequently later changed to the Swedish Nuclear Fuel and Waste Management Co., SKB.

SKB is also responsible for the R&D programme for the management and disposal of spent fuel and radioactive waste prescribed by the Nuclear Technology Act of 1984. SKB in addition manages issues related to uranium exploration, uranium storage, enrichment of uranium, and spent fuel reprocessing. The company assists in the procurement of uranium. The licensing authorities are the National Institute of Radiation Protection, SSI, and the Nuclear Power Inspectorate, SKI.

The present nuclear power programme, 12 reactors of together 10 GWe, requires around 1600 tonnes of uranium annually. For the period 1990-1999 the requirement is estimated at 14900 tonnes. At the end of 1990 the Swedish utilities had secured 11000 tonnes, by uranium in storage and by long term contracts. The main part of the supply is based on long term contracts, where Canada and Australia each will supply 40% of future contracted deliveries (6-16, 6-17).

REFERENCES


6-3 Government bill 1957, appendix 12.

6-5 Sköld, P.E., et al
Swedish atomic weapons? (In Swedish).
Tiden, Stockholm 1959.

6-6 Ahlmark, P.
The Swedish atomic weapons debate. (In Swedish).
Aldus/Bonniers, Stockholm 1965.

6-7 Broden, K., Carleson, G., and Hultgren, Å.
International developments in the nuclear fuel cycle with emphasis on transport and safeguards. (In Swedish).

6-8 The Ranstad investigation. (In Swedish).
Department of finance, Stencil 1967:14.

6-9 Uranium from Swedish shale. (In Swedish).
AB Atomenergi, 2 October 1968.

6-10 The Swedish uranium policy. (In Swedish).
Department of industry, PM 1969:2.

6-11 Carlsson, O., and Nöjd, L-Å
Uranium production from low-grade Swedish shale. IAEA-CN-36/277.

6-12 Carlsson, O.
Ranstad—one of the world’s largest uranium deposits. (In Swedish). JkA No.1/80.

6-13 Billingen—4 examples. (In Swedish).
Department of industry. SOU 1977:47.

6-14 Sweden's supply of nuclear fuel.


SKB 1993.
7 RANSTAD RESTORATION PROGRAMME

7.1 Background

After the plans to re-open the uranium production had been cancelled in 1980 (chapter 5) RSA/ASA continued the process development on alum shale utilization with reduced staff until 1984. The activities included work on the DMS-process and thermal experiments in a synthetic gas reactor (SGR), financed with the remainder of ASA’s loan for 1978/79–81/82 and additional grants from R&D-funds for fossil energy recovery. RSA also continued the necessary drainage pumping in the open pit and the treatment of waste water from the disposal area.

When the license for shale mining at Ranstad expired at the end of 1984, RSA/ASA decided to stop all activities based on alum shale. The staff was noticed and the localities were let out to other companies. One of these, Ranstad Industricentrum AB, acquired RSA in 1987.

When RSA was established and took over the Ranstad plant in 1978, the agreement included a clause that Studsvik Energiteknik AB maintained responsibility for restoration of areas affected by the earlier uranium production. Consequently Studsvik Energiteknik AB took charge of draining and waste water treatment at Ranstad as well as of preparations for final restoration from fiscal year 1984/85 with special government financing until the end of 1988.

From 1989 the Ranstad restoration was included in a programme according to "the Studsvik law" (7-1), for the decommissioning of facilities and management of waste from earlier government funded development of nuclear power in Sweden. These activities are financed by a fee charged to the utilities on the current production of nuclear power. The fund was administrated by The National Board for Spent Nuclear Fuel from 1989 until mid 1992, when it was transferred to the Swedish Nuclear Power Inspectorate. The former state-owned AB Atomenergi that had changed its name to Studsvik Energiteknik AB and finally to Studsvik AB, was taken over by the State Power Board in 1991. The utilities established a separate company, AB SVAFO, for implementation of all the activities following the "Studsvik law", including the restoration at Ranstad from mid 1992.
7.2 Earlier restoration experiments and measures

The open-pit area

In connection with the mine preparation and quarrying during the 1960's, restoration work was performed by tractor levelling of the 10-20 m high embankment along the about 2000 m starting border of the open-pit area. Also the greater part of the mined and refilled area was levelled with a crawler tractor, according to the mining license stipulations, supervised by the County administration. At the end of the 1970's a large-scale recultivation test was made, comprising an area of about 30 000 m², or nearly half of the refilled area ("the east branch" of the open pit). To create a suitable ground profile for farming and foresting about 200 000 m³ of blasted rock and moraine had to be moved by crawler tractors and 75 000 m³ was transported and spread out by trucks. Over the whole area a ground profile was created with rock at the bottom, covered by at a minimum of 2 m of moraine and 0.1 - 0.75 m of top soil, depending on the intended use of land. The area was divided into fields prepared for corn cultivation, grazing land, and forestry. The harvest results were followed up by the Swedish Farming University and proved to be in level with those in the surrounding region. This field test demonstrated the possibility to restore the land to normal use after mining alum shale in open-pit.

The waste disposal area

As mentioned in chapter 5 it was noted during the plant production period in the 1960's that weathering of the shale residues occurred in the disposal area, and experiments were started to find more suitable disposal methods for the future. The dominating reaction at weathering of the shale leaching residue is oxidation by oxygen in the atmosphere, in the presence of moisture, of pyrite (FeS₂) in the shale to form iron sulphate and sulphuric acid. Also other elements, such as aluminum, magnesium and manganese are dissolved in this process, the more so as the weathering proceeds and pH is lowered. In the laboratory experiments no anaerobic weathering could be observed, except that Fe²⁺ may be reduced to Fe⁰ by the pyrite weathering. It was also shown that leaching residue stored under water was not affected by any notable weathering. Compaction of the shale residue reduced the
weathering effect, even if the shale was in direct contact with the atmosphere.

The experiments resulted in two disposal options:

- the "dry method", with disposal of the residue above the ground water level, under a tight capping system, and

- the "wet method", with disposal under the ground water level and with surrounding tight walls to prevent water flow to the sides.

As mentioned in chapter 5 the "dry method" was tested in a pile of 15,000 tonnes, established in 1972 and still possible to monitor after completion of the final restoration in the waste disposal area.

The "wet method" of leaching residue disposal, integrated with the open-pit mining, was developed for the production re-opening projects during the 1970's (see chapter 5). A further development was a double wall system surrounding the submerged leaching residue, preventing any contaminated water release by horizontal flow, while the extremely tight alum shale bottom prevents vertical leakage.

During the 60's and 70's different parts of the waste disposal area were covered, in order to reduce the weathering, awaiting a final restoration. The most important works were made in 1978-79, when all remaining areas were covered with about 0.5 m of moraine from the open-pit. The cover was fertilized and recultivated with grass. The favourable effect of this gradual covering was clearly proved by steadily reduced pollutant contents in the water from the disposal area. However, the water still had to be cleaned before discharge to the recipient.

These earlier restoration measures were summarized in an RSA report to the County administration in 1981 (7-2).

7.3 Planning of the final restoration

During 1984/85 a government funded programme study was performed by Studsvik Energiteknik AB and RSA. The study was reported to the government in January 1986 (7-3), presenting the current situation at Ranstad and results from earlier
restoration measures. Two main options for the final restoration were outlined:

1. Water filling of the open pit and covering of the leaching residue in-situ, using the "dry method";

2. Moving the leaching residue back to the open pit for disposal under ground water ("wet method"), and total open-pit refilling.

The programme report specified the additional investigations needed for the choice of main restoration option and to produce a detailed restoration plan. Based on the report and considerations given by central and local authorities and organisations, the government accepted to finance the continued planning of the final restoration at Ranstad (7-4).

It may be noted that The Swedish National Environmental Protection Board during the mid 1980's investigated the environmental problems caused by a great number of closed down mining operations, especially sulphide ore mining in the middle of Sweden. Here again pyrite-marcasite weathering was the common origin of pollutants to the water streams (7-5). The Board promoted the treatment of mining residues at Bersbo, in the province of Östergötland, as a pilot project (7-6).

During the investigation phase 1986-88 relevant site-specific data were collected for the detailed restoration plan. This included mapping, studies of geological and hydrological conditions and analysis of internal and external water streams. The results from the 15000 t test pile were evaluated (7-7) as well as the Bersbo pilot-project.

The detailed plan for the final restoration of the Ranstad site was submitted to the County administration in October, 1988 (7-8). In short the plan proposed water filling of the open-pit and covering of the waste disposal area using the "dry method". In the industry area some of the buildings, including the silos, were to be demolished.

The aim of the restoration was to reduce and eventually eliminate any needs of maintenance at the site, by reducing the environmental impact to acceptable levels. A prediction of the water quality in the natural recipients after the restoration was therefore an important part of the restoration plan.
7.4 Realization of the restoration

After a comprehensive review permission was granted to start the restoration project in January 1990 (7-9) according to the restoration plan and additional instructions by the County administration. It should here be noted that the National Institute of Radiation Protection considered the proposed monitoring programme sufficient also for the radiological conditions.

The time schedule for the restoration project is shown in fig. 7-1.

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5-10 years

Figure 7-1. Time schedule for the total Ranstad restoration project.

From 1989 the project has been financed according to the "Studsvik law" (see section 7.1). In a cost calculation of the total programme presented in April 1990 (7-10), the Ranstad restoration cost was estimated at 140 MSEK.

To follow the project the County administration set up a reference group with representatives from the National Environmental Protection Board and the local communities.

The project organisation within Studsvik AB was supported by external consultants for the specification and supervision of the work to be performed.

The main civil engineering work included in the "primary restoration phase" was purchased as four major contracts, all from the same contractor.
The primary restoration phase was completed as scheduled by mid 1993. The work can be shortly described as follows.

**Restoration of the open-pit**

The very tight alum shale bottom has been used to create a lake with an area of 250,000 m², by letting the groundwater fill up the mining area to about the original level and discharge to the natural recipient, a nearby creek. Before the water filling, waste rock along the mining front was removed to the bottom of the pit and covered by moraine. The highest part of the remaining embankment along the start side of the pit was lowered and the future bank profile was shaped to withstand wave erosion. Totally over 0.5 Mm³ of rock and earth material were moved before the water filling.

An important issue was to tighten the shaft for the underground primary crusher, which penetrated the alum shale bottom into the permeable sandstone. The solution was to inject the shaft walls using a sophisticated "dynamic" grouting method, followed by filling of the shaft with moraine of specified quality.

The water-filling of the lake was completed by mid 1993. A section of the lake is shown in fig. 7-2.

![Diagram of the lake](image)

Figure 7-2. Schematic section of the lake in the former Ranstad open-pit

One end of the lake is filled out to give shallow water, suitable for bird life, while the other end has a depth of up to about 15 m, with the banks partly suitable for swimming, see photo fig. 7-3.
Restoration of the waste disposal area

The establishment of a tight cover on the leaching residues and additional mill tailings was the most complicated part of the restoration, considering the County administrations requirement of a hydraulic conductivity less than $5 \times 10^{-9}$ m/s. Also such factors as the availability of material, the long-term stability and the experience from using the material had to be considered in the design of the covering system. The preferred solution is shown in fig. 7-4.

Figure 7-4. Section of the cover system used for the waste disposal area at Ranstad
Moraine is a reliable, natural material, having been used in many Swedish hydro power dam constructions. Within the disposal area at Ranstad a sufficient quantity (about 50 000 m$^3$) of an especially favourable type of moraine was found, containing a fine fraction of clay shale particles. This material proved to give the required low permeability without the anticipated mixing with bentonite. For the larger moraine quantities necessary for the protection layer, there was a good supply both within the disposal area and in the open-pit.

The moraine for the tight layer was screened for the removal of pebbles larger than 50 mm and homogenized in a mixer. After careful grading and compacting of the leaching residue, the moraine for the tight layer was spread with a laser-controlled grader and compacted with a 6 t vibrating roller. The resulting tight layer was strictly controlled by laboratory and field testing of such factors as moisture content, fine particle fraction and compaction. The total covered area was about 250 000 m$^2$. After quality acceptance the limestone layer for draining and the moraine layer for protection was successively laid out, see fig. 7-5.

Figure 7-5. The limestone being spread on top of the tight layer, covering the tailings
For the future monitoring of the cover system, a large number of pipes for observation of the ground water level above the tight layer were installed. For testing of the oxygen diffusion, lysimeters have been placed underneath the tight layer.

7.5 Water system and environmental control

After the completion of the primary restoration phase (fig 7-1) at mid 1993, the transition phase will start, with gradually reduced discharge of pollutants. The still necessary cleaning and pumping of the water from the waste disposal area, see fig. 7-7, is anticipated to be replaced with discharge to the natural, smaller recipient in the area, without cleaning, see fig. 7-8.

A compartment model has been used to predict the future impact on the natural recipient, based on the anticipated infiltration rate of 0.2 l/s in the disposal area. All polluting elements except Fe have been predicted to be within class 0 (none or negligible influence) in the classifying system used by the Swedish National Environmental Protection Board.
7.6 Concluding remarks

The final restoration of the areas affected by uranium production at Ranstad have been performed with a high quality standard.

In a report by AB SVAFO in 1993 (7-11) the costs for the restoration from 1989 until completion are estimated at 143 MSEK (price level 1993).

A funding of 15% of the original investment cost for the Ranstad plant in 1965 at an interest rate of 7% would have been enough to finance the restoration started in 1989.

A comprehensive description of the restoration at Ranstad is given in ref 7-12.
REFERENCES

7-1 Law on the financing of the management of certain radioactive waste etc.
SFS 1988:1597

7-2 Realized and planned restoration measures at the Ranstad works.
RSA, November 1981

7-3 Restoration at Ranstad.
Report from the Programme stage.
Studsvik Energiteknik AB, Ranstad Skifferaktiebolag.
January 1986

7-4 Government Bill 1986:87:100, Appendix 16

7-5 Södermark, B.
Mining waste.
The Swedish National Environmental Protection Board,
April 1986

7-6 Mining waste at Bersbo
VIAK 1986

Transport of water and oxygen through the cover layer on the leach residue deposit at Ranstad.
University of Linköping, December 1988

7-8 Restoration at Ranstad
Plan for post-treatment
Studsvik AB, Ranstad Industricentrum AB, October 1988

7-9 Plan for post-treatment of the mining area at Ranstad.
Approval by the County administration of the post-treatment plan worked out by Studsvik AB
County administration of the county of Skaraborg,
January 1990

7-10 Cost calculation 90
for the management of certain radioactive waste and the decommissioning of nuclear facilities.
AE Projekt AB, April 1990

7-11 Cost calculation 93
Costs for the management of older radioactive waste at Studsvik, and for the decommissioning of certain nuclear facilities.
AB SVAFO, April 1993

7-12 Linder, P., Sundblad,B *)
Remediation of the closed-down uranium mine in Sweden.
ASME 1993 International Conference on Nuclear Waste Management and Environmental Remediation.
Prague, September 5-11, 1993
*) Studsvik EcoSafe
CONCLUSION AND PERSPECTIVE

Up to the beginning of the 1970s, all forecasts of nuclear power expansion were high. In the Western world, nuclear power was forecast at 580 GWe for 1985, and at 2650 GWe for the year 2000. The uranium resources with production costs <80 US$/kg were, according to OECD, estimated at 1.8 mill. tonnes and additional uranium in the next cost category, <130 US$/kg, estimated at 1.3 mill. tonnes. This was not sufficient to supply the nuclear power forecast with low cost uranium, and exploration for additional uranium was thus stimulated, yielding new mines and extended production (8-1).

The strongly reduced rate of nuclear power expansion in the 1970s and 80s, as compared to forecasts, led to considerably lower uranium consumption than expected. At the same time the military strategic uranium reserves of the superpowers became saturated. This caused a decrease in uranium production and in lower uranium exploration for new capacity. Yet the low cost uranium reserves are today of the same order as 20 years ago and estimated to be sufficient to supply the present nuclear capacity for around 90 years.

While the global uranium requirements in 1991 were estimated at 53492 tonnes by the Uranium Institute, the world production was only 41525 tonnes in the same year, for a global nuclear power capacity of 320.8 GWe (8-2). The gap was filled by material from producer and utility inventories, from reprocessing of spent nuclear fuel, and by material slowly becoming available from the decommissioning of military stockpiles. The disarmament treaties START I & II imply that 75% of the 1100 tonnes HEU and 230 tonnes plutonium in nuclear warheads will become available for the civilian market after the implementation of the treaties (8-3). This will also contribute to compensate over several years the reduced production in the Western world. The January 1993 uranium price level, 150 SEK/kg on the spot market, was at an historically low level.

More important than, but not independent of, the development of economy and technology, are the development and perception of social values. Predominant values in the 1950s and the 60s, when the decisions were taken, that were important for the development of nuclear power, were entrepreneurialship and a positive acceptance of technical ad-
In the 1970s the climate changed to a questioning of lifestyles vs environmental protection with a growth of antiauthoritarian attitudes, that founded a hostile view of the use of nuclear technology. In the 1980s values changed back to more pragmatic and materialistic views. In 1990 a majority of the Swedish people, around 60%, agreed that the development and use of nuclear power had been, completely or in the main, a benefit for the country (8-4). Also, the number of people wishing to keep nuclear power until or beyond year 2010 increased dramatically from 1986, 29%, to 1990, 58% (8-5).

Thus, it seems that the future phasing out of nuclear power in Sweden is by no means defined as to when or how it will be replaced. Social values change, as do market conditions. Future generations will make their own decisions. It can never be excluded, that the vast Swedish uranium deposits will be used for power production in the future.

The authors of this report have felt it a privilege to work out this review of developments in the field of uranium recovery in Sweden. This feeling is based i.a. on their own participation over many years in the development of the nuclear energy system, that we now have in Sweden.

REFERENCES


ACKNOWLEDGEMENTS

Our thanks go first of all to Dr. E. Svenke, in the past Technical Director of AB Atomenergi and the first Managing Director of the Swedish Nuclear Fuel Supply, for his contribution and for most stimulating discussions. We also have greatly appreciated discussions and advice on documentation from my former colleagues in AB Atomenergi A.-L. Arnfjord, A. Peterson, and E. Strandell. Mr. B. Lilljha, the present owner of the Ranstad area and remaining facilities, gave generous support for studies in the Ranstad archive.

This project was originally started under a contract from AE Projekt AB. After the State Power Board took over ownership of Studsvik AB, it was transferred to AB SVAFO, jointly owned by the nuclear utilities the Southern Power Co., SPB, and the Forsmark Power Group, and managed by SKB. Our contact with AB SVAFO was Dr. I. Lindholm of SKB, and we want to thank him for a generous guidance.