EXPLOITATION OF URANIUM RESOURCES IN TURKEY -A SHORT REVIEW IN RETROSPECT OF THE TECHNICAL AND ECONOMICAL ASPECTS

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ABSTRACT

A brief summary of the history of discovery and exploration of uranium deposits in Köprübasi area with sampling activities would be made. A short description of the laboratory, bench scale and pilot plant tests performed at MTA (Directorate of Mineral Research & Exploration of Turkey) laboratories and at site in Köprübasi pilot plant facilities will be given together with a assessment of the results obtained.

A critical evaluation of the availability of mineable reserves, assessment of the reserve calculations, economical cut-off-grade concept used will be presented.

The results of the preliminary and detailed feasibility analysis prepared by experts working together with teams from MTA, Etibank (now Eti Holding A.S.), UNDP and IAEA at different stages of progress in the development of the deposits. The conclusions are re-evaluated taking into account the current uranium market with criteria used for investment decision at the time.

1. INTRODUCTION

The uranium occurences in the Köprübasi area is located 400 km WSW of Ankara on the main road from Demirci to Manisa near the northern side of the Demirköprü dam.

The local topography is strongly indented with occasionally steep slopes and ravines. The climate is mild with hot dry summers and fairly wet winters.

The uranium bearing sediments are mainly gravels, siltstones, sands and clays overlaying paleosoic gneiss and schists of the North Menderes Massive. Some beds of calcareous breccias, limestones and phosphates are also present. The mineralized horizons are not associated with any particular bed in the sediments and vary considerably in thickness from 0.25 to over 10 m. Average thickness is 4 m. The depth of the mineralized horizon from the surface also varies considerably from outcrops to over 25 m with an average thickness of 8 m.

The unconsolidated nature of both overburden and mineralized horizons should permit low cost mining without extensive explosive usage. The secondary uranium minerals (autinite,

metaautinite and torbemite) loosely adhere or coat the surfaces of individual grains. Pebbles or cracks are easy to leach and should allow low cost treatment by heap leaching.

First uranium exploration was started in 1957 by Mineral Research and Exploration General Management (MTA) and Köprübasi uranium deposits were discovered in 1961 as a result of aerial and ground surveys.

By 1963 a total of 65 surface trenches and 115 holes on a 50x50 meter grid has been completed. Drilling totalled 4500 metres and 5 shafts with a total depth of 82 metres were sunk.

Further exploration works were carried out in 1974 under a program sponsored by UNDP. Progress was made by using modern logging equipment for radiometric and electrical resistivity and conductivity measurements. Over 20 holes were drilled and logged in the Kasar deposit to confirm earlier results. Ore reserve calculations were reviewed by MTA in 1978.

2. ORE RESERVES

The result of earliest ore reserve calculations were reported in 1963. Additional ore reserve were discovered by MTA as more drilling was completed at Kasar (the Kasar deposit is considered to have the largest reserve and easier amenability to leaching process) and other orebodies culminating in the final geological reserves. Table-1 gives the final ore reserve estimations.

Table-1. Ore Reserve Estimation for Köprübasi Uranium Deposit.

Deposit Name	Ore Reserve			Waste m^3	Stripping Ratio m^3/ton
	Tons	%U	Contained Uranium ton U		
Kasar	1 325 760	0.0349	463	7 954 560	6.0
Çetintas	811 000	0.0258	209	3 244 000	4.0
Topalli	577 800	0.0329	190	1 848 960	3.2
Tasharman	1 408 300	0.0453	638	12 956 360	9.2
Total	4 122 860	0.0364	1 500	26 003 880	6.3

Source : MTA reserve report.

The above four deposits were selected for their suitability for open pit mining. Other known occurences in the area were not taken into account as they were submarginal for open pit mining. The only exception is Ecinlitas deposit where an appreciable amount of ore could be developed through underground mining.

It is assumed that above reserves could supply mining and processing facilities with a capacity of 230,000 tons run of mine ore producing some 80 tons U per year for 10 years.

The MTA reserves estimations were based on the polygonal method using minimum grade of 0.015 % U with no consideration to its economics, viability or phsycal warranty for mining.

The next report prepared by one of the experts of International Atomic Energy Agency (IAEA) contained a preliminary feasibility study based on MTA reserve estimation. Etibank calculated mineable reserves based on MTA reserve estimation by designing separate open pit mines for each of the four deposits.

Table-2. Mineable Reserves of Köprübasi Uranium Deposit

Deposit Name	Ore Reserve			Waste m^{3} 4	Stripping Ratio m^3/ton
	Tons	%U	Contained Uranium ton U		
Kasar	1 140 000	0.036	416	1 000 000	0.9
Çetintas	330 000	0.027	91	660 000	2.0
Topalli	310 000	0.037	114	470 000	1.5
Tasharman	510 000	0.040	312	1 900 000	3.7
Total	2 290 000	0.040	933	4 030 000	1.75

Source : Nuri Birtek, MTA Tarafindan Tespit Edilen Uranyum Zuhurlarinin Isletilmesi ve Sari Pasta Üretilmesinin Teknik ve Ekonomik Yönleri.

3. ORE TYPES

The mineralization of orebody can be classified under three main type, according to mineralogy and lithology.

i- Kasar type; The uranium occurs as autinite, meta autinite and torbernite in the fine matrix.

ii- Tasharman type; It contains significant amount of phosphate as dahllite and apatite. The carbonate content higher than Kasar. Specific uranium mineral has not been detected.

iii- Carbonate type; This type is characterized by high calcite and dahllite with some apatite.

4. ORE PROCESSING STUDIES

The laboratory ,bench and pilot plant scale tests were performed during 1964-1965 and early 1970's. Assistance was provided IAEA through UNDP "Assistance to MTA Technological Laboratories " Project. Different flowsheets for different class of ores were developed.

Laboratory investigations have been conducted on both agitation and column leaching in Ankara while pilot scale leaching has been carried out in vats and on heaps at Köprübasi Pilot Facilities of MTA.

Agitation leaching

Agitation leaching was carried out by MTA to study a variety of parameters. Kasar and Tasharman type ores are amenable to leaching by agitation with acceptable to good acid consumption figures.

Percolation leaching

Six columns were set up and the ore leached by a series method in which the acid fraction of the effluent in one column was passed to the next column to achive high uranium concentration required for solvent extraction. Washing stages also followed in a sequential, counter-current like manner.

As a further step, some pilot vats were constructed and Kasar samples were tested by percolation leaching in 24 tons lots. It has been shown that good recovery levels could be reached when leaching time was increased from 63 days to 385 days.

Heap leaching

While the first heaps were established using approximately 100 t samples of Kasar ore. Subsequently heap sizes were increased to 1000 t.

In order to produce a high grade effluent in the range of 0.8 to 1 g/l required for solvent extraction, recycling of the effluent was necessary.

Although Kasar type ores generally gave satisfactory recoveries in the heaps, Tasharman type ore and mixture of Kasar-Tasharman ores did not give very good recoveries due to gypsum caking.

Solvent extraction

The solvent extraction pilot plant erected at site had six stage mixer- settler unit with a nominal feed capacity of 80 litres per hour.

No difficulties were experienced in the process when using Kasar leach liquors; but liquors produced from Tasharman type ores contained significant amount of phosphate.

Precipitation of yellow cake

The strip liquor produced from solvent extraction plant was used to precipitate yellow cake using magnesia as a precipitating agent.

Kasar ore can be treated by acid leaching with an acceptable acid utilization (20 kg/t) followed by solvent extraction to produce yellow cake.

5. TECHNICAL AND ECONOMICAL EVALUATION

Basic operating and capital cost parameteres used at different stages of project development is given in Table-3.

Table-3. Basic Cost and Operating Parameters for Köprübasi Uranium Deposits

Parameter	Koch (1)	Etibank (2)	Hansen (3)
Mining cost/ ton	$ 1.50	$ 2.50	$ 9.10
Haulage cost/ ton/km	$ 0.15	$ 0.06	$ 0.06
Processing cost/ ton	$ 8.25	$ 10.70	$ 10.70
Other costs/ ton	--	--	$ 1
Process recovery (%)	80	--	80
Max. allow.total cost/ KgU	--	$ 70.00	$ 77.00
Return on costs (%)	--	10	--
Stripping cost/ m^3	Included in mining cost		$ 1.25
Mining recovery (%)	100	85	85
Mining dilution	0	10	10

(1) From L.W.Koch Report to IAEA, 1983.

(2) From Etibank's Preliminary Feasibility Study, 1984.

(3) From M.V.Hansen's Report to IAEA, 1985.

Summary of the results of the work done by IAEA expert Mr. M.V.Hansen, concerning Köprübasi Uranium Deposit is given in Table-4. Specific arbitrary cut-off grades as listed below were chosen for economic studies.

Table-4. Summary of Economical Evaluation of Köprübasi Uranium Deposit From M.V.Hansen's Report

	In place reserve			Mineable reserve									
Cut-off U3O8 %	Tons ore (*1000)	U %	Tons U	Tons ore (*1000)	U %	Tons U	Waste m^3 (*1000)	Total Operating. Cost (Mio.$)	Tons U Rec.	Operating. Cost KgU ($)	Total Capital Cost ($) (*1000)	Capital Cost KgU ($)	Total cost Kg U ($)
0.020	2237	0.031	743	2176	0.029	630	9687	58.96	454	130	7600	17	147
0.025	1736	0.036	629	1625	0.033	539	8653	45.82	409	112	7600	19	131
0.030	1202	0.042	507	1125	0.038	431	7864	34.065	342	100	7375	22	122
0.035	899	0.047	422	841	0.042	357	6852	26.694	290	92	7375	25	117

Capital costs used in the table above were as follows;

Mining equipment	3.600.000 $
Processing plant	3.500.000 $
Heap leach pad	500,000 $
Total	7.600.000 $

The estimated costs of per kilogram of uranium are considerably higher than the highest allowable cost. Furthermore, the reserves are considerably reduced from the earlier estimates as a result of applying a cut-off grade criteria. Another factor which could reduce the estimates is the result of breaking of the orebody up into progressively smaller pods as the cut-off grade is increased until the mining of the ore deposit effectively was impossible. This effect became quite noticable at the 0.035 % U_3O_8 cut-off.

In 1986 F.E. Mc.Ginley, the IAEA expert, made comparison of mineable reserves estimated by Etibank and Mr. Hansen during his mission. Tasharman deposits were eliminated by Mc. Ginley since the Tasharman type ore were not amenable to heap leaching.

6. ECONOMIC CUT-OFF GRADE ESTIMATION OF KÖPRÜBAŞI DEPOSITS BY ETIBANK

On November, 1987 Etibank studied again Köprübasi uranium reserves to determine the economic cut-off grades (e.c.o.g) of deposits. To find out the e.c.o.g. Etibank has made use of the computer program based on empirical formulae taken from Mr. T.Alan O'Hara's paper "Quick Guide to Evaluation of Orebodies". To evaluate e.c.o.g. of Koprübasi ores, costs are converted to Turkish Lira by using U.S. Bureau of Mines CES (Cost Estimating System) program.

Mr. Hansen's results such as daily tonnage, stripping, recoveries and haulage distances were accepted as input data to the computer program. In order to calculate e.c.o.g., price of uranium assumed to be $ 53.63/Kg U ($ 16.65/lb U, 1987 price).

Mr. M.V.Hansen has estimated economic cut-off grades for Kasar, Tasharman, Çetintas and Topalli separately using following equation. The results are given in Table-5

$$\text{Cut-off grade} = \frac{\text{(Mining+Haulage+Process+Other) costs \$ per ton}}{\text{1000 * Price of Yellow Cake / kgU * Process recovery}} * 100$$

Table-5. Separately estimated economic cut-off grades by Mr.Hansen.

Deposit	Economic Cut-off Grade
Kasar	0.040
Topalli	0.041
Çetintas	0.041
Tasharman	0.042
Blend	0.040 / 0.042

The above mentioned computer program was employed for Hansen's & Mc. Ginley's data separately. The output results show that economic cut-off grade is unacceptably above the actual ore grades for both of the cost data.

Table-6. Computer results for process & mining costs per ton Hansen's data for the deposit

Cut-off grade (% U_3O_8)	Process cost ($/ton)	Mining cost ($/ton)
0.020	18.55	12.49
0.025	20.70	15.53
0.030	23.43	21.02
0.035	25.82	25.81

Table-7. Economic cut-off grade values with Hansen's data
(Kasar + Tasharman + Çetinbas + Topalli)

Cut-off Grade (% U_3O_8)	Economic cut-off grade (% U_3O_8)	Reserve (tons)	Waste (tons)	Average grade (% U)	Production (tons/day)
0.020	0.076	217,000	1,937,400	0.032	7,200
0.025	0.088	162,500	1,730,000	0.036	6,300
0.030	0.107	112,500	1,572,800	0.042	5,600
0.035	0.124	84,100	1,370,400	0.047	4,850

Table-8. Computer results for process and mining costs per ton using Mc.Ginley's data for the deposit.

Cut-off grade (% U_3O_8)	Process cost ($/ton)	Mining cost ($/ton)
0.020	23.43	13.01
0.025	25.94	15.65
0.030	29.51	20.02
0.035	51.63	24.73

Table-9. Economic cut-off grade values with Mc.Ginley's data (Kasar+ Çetintas + Topalli)

Cut-off Grade (% U_3O_8)	Economic cut-off grade (% U_3O_8)	Reserve (tons)	Waste (tons)	Average grade (% U)	Production (tons/day)
0.020	0.088	119,400	702,000	0.030	2,730
0.025	0.100	87,700	572,600	0.035	2,200
0.030	0.119	60,100	463,600	0.040	1,750
0.035	0.181	43,000	376,600	0.044	1,400

7. DISCUSSION

The grades of the Köprübasi uranium deposit can be considered to be comparatively low.There are only very small separate occurrences.The only sizeable occurrence Kasar has not produced expected minable reserves after due dilegence studies performed.

The four small deposits that were classified as mineable later had to be reduced to three due to processing difficulties as Tasharman ore was not considered to be amenable to heap leaching by the processing experts. The elimination of Tasharman has caused the over head costs and production costs to increase due to smaller production rates.

Haulage costs were higher than usual as ore has to be transported from relatively far distanced open pits ranging from 5 to 18 km. to a small central processing facility contributing to overhead costs as separate roads and other investments such as electricity, water etc. were to be included in the capital expenditures.

Low uranium prices due to increasing productions from new rich discoveries and supressed demand and also the recent commercialisation of uranium from military sources, creates a bleak future for extremely low grade Köprübasi deposit reducing the probability of operation to virtually to non at present circumstances.

All the efforts to reduce the operating costs and mining costs such as reduction of waste volume per ton and increasing processing recovery etc for instance by not including Tasharman ore in the blend have failed and the resulting e.c.o.g figures were found to be even higher.

8. CONCLUSION

All studies indicate that some of the Köprübasi uranium deposits are amenable to usually low cost heap leaching and a small central processing facility with an 80 tpa product capacity could be contemplated to operate ten years. Lower cost open pit mining at various selected deposits can be performed and the resulting blend from the run-of-mine ores can be processed.

However, above referred disadvantages have not supported an investment decision; as the the cut-off grades calculated from estimated cost of production and sale price figures were three times higher than the average grades of the minable ore for all the blending options tested.

Lower level of uranium prices from those used in the previous studies combined with escalated investment and operating costs did not justify a renewed feasibility study by our team.

REFERENCES:

1. J.Cameron's report to IAEA, Development and Evaluation of Uranium Deposits.Oct. 1962-Sep.1963
2. MTA, Manisa-Köprübasi Uranyum Cevherleri Üzerine Pilot Çapta Teknolojik Çalisma. Aralik 1978
3. Etibank- Proje Tesis Dairesi Baskanligi, Sari Pasta Üretimi Ön Fizibilite Etüdü.
4. F.R.Hartley's Report to IAEA Ore Processing of Köprübasi Uranium Ores 1979
5. MTA, Manisa-Köprübasi Bölgesi Uranyum Cevher Konsantrelerinin (Sari Pasta) Çözündürme ve Çözücü ile Özütlenmesi (SX) Çalismalari., Mayis 1981
6. MTA, Manisa-Köprübasi-Tasharman Tüflü ve Konglomera Uranyum Cevherinin Laboratuvar ve Pilot Teknolojik Degerlendirilmesi. Haziran 1982
7. MTA, Manisa-Köprübasi-Kasar Uranyum Cevheri Üzerine Pilot Çapta Deneyler Ek Rapor 1. Aralik 1982
8. Nuri Birtek, MTA Tarafindan Tespit Edilen Uranyum Zuhurlarinin Isletilmesi ve Sari Pasta Üretilmesinin Teknik ve Ekonomik Yönleri. 1983
9. Etibank & L.W. Koch, Köprübasi Uranium Project, Preliminary Feasibility Study. December, 1983
10. Etibank-Proje Tesis Dairesi Baskanligi, Uranyum-Sari Pasta Üretim Projesi Hakkinda Not. 15 Ekim 1984
11. M.V.Hansen's Report IAEA, Estimation of Uranium Ore Reserves at Köprübasi 1985
12. Etibank-Proje Tesis Dairesi Baskanligi, Uranyum-Sari Pasta Üretim Projesi Hakkinda Not. 4 Kasim 1985
13. F.E. Mc Ginley's Report to IAEA, Exploitation of Uranium Resources, Turkey, May, 1986
14. T.Alan O'Hara, Quick Guides to the Evaluation of Orebodies, CIM Bulletin