Beneficiation of tungsten ores in India: A review

N KRISHNA RAO

Ore Dressing Section, Bhabha Atomic Research Centre, Begumpet, Hyderabad 500 016, India

Abstract. Tungsten is a strategic metal for India, and almost the entire requirement is met by imports. Extensive search for tungsten in recent years has led to the discovery of several deposits. However, almost all these are of low grade compared to world resources. Techno-economic exploitation of these deposits depends to a great extent on the development of suitable beneficiation technology within the framework of an ore-to-product integrated approach. This paper presents a review of the current research and development status of beneficiation technology applicable to these deposits.

The deposits covered are: (i) Degana, comprising of four distinct types, namely, quartz vein, eluvial, phyllite and granite, (ii) Balda, (iii) Khobna–Kuhi–Agargoan, (iv) Burugubanda–Tapaskonda, (v) Scheelite-bearing gold ores of Kolar and Hutti, and (vi) Madurai. Investigations on these ores have been mainly carried out by BARC, IBM, BRGM (France), NML and RRL (Bhubaneswar). While studies of BARC have been described in detail, those of the other laboratories are briefly discussed. Emphasis has been laid on discussing the industrial flow-sheets recommended during these investigations.

The strategy needed for the techno-economic feasibility of beneficiation of low grade tungsten ores are (i) effective pre-concentration at as coarse a size as possible, (ii) emphasis on higher recovery rather than on high grade of the concentrate, (iii) a two-product approach, one of high grade feasible by physical beneficiation methods and the other of low grade, to be upgraded by chemical methods to directly usable products, thus maximizing recovery, and (iv) a maximum utilization concept, aiming to recover all possible byproducts. The flow-sheets developed for the beneficiation of individual deposits are discussed in the light of the above strategy.

Keywords. Tungsten ores of India; beneficiation flow-sheets; international practice; pre-concentration; upgradation; economic level of concentration; by-product recovery; ore to product strategy.

1. Introduction

The special properties of tungsten and its alloys such as extreme hardness, wear resistance, high melting point, high density and low vapour pressure have made it one of the most sought after and unique metals. It finds applications in a number of industries, including defence and high technology fields, for which no other substitutes have so far been found. It is one of the strategic metals identified for stockpiling in many countries. Its critical use in several defence applications, a meagre indigenous production, and total dependency on imports with consequent drain in the foreign exchange, have made tungsten one of the strategic metals in India. As against the present demand of about 2500 tonnes per year, which is expected to reach about 4500 tonnes by the turn of the century, the indigenous production of tungsten is a meagre 20–30 tonnes per year (Anon 1991).

2. Geochemistry and mineralogy

Tungsten may be considered as one of the rarer elements in the earth's crust, the average crustal abundance is estimated to be at 1 to 1.3 ppm. The highest content among

Table 1. Various tungsten deposits and their locations.

Type of deposit	Major locations
Skarn	Primorskii Krai (Vostok II), Central Asia (Ingichke, Koitash), Caucasus (Turnyaz) all in USSR., Sang-Dong (South Korea), Khuan-Podi (China), Emerald Feney (Canada), Pine Creek (USA) and King Island (Australia)
Greisen	Transbaikal Region (Spokoininskii) Kazakhstan (Akchatau, Kara-Oba) in USSR., Sadisdorf, Pechdelgrun (GDR), Montebrasse (France), Pyaton, Sinkiangshang (China), Wolfram Camp, Terrangton (Australia)
Plutonogenic hydrothermal	Chukotka (Iultin), Transbaikal Region (Bukuka), Kazakhstan (Upper Kairakty) in USSR., Panas-Queira (Portugal), Cornwall (UK), Belfort (France), Red-Rose (Canada), Herberton (Australia)
Volcanogenic hydrothermal	Transbaikal Region (Barun-Shiveya) Central Asia (Tasor, Ikar), Cacausus (Zopki-to) in USSR., Usin, Siang (China), Akenobe (Japan), Tungoten-Queen (Canada), Yellow-Boulder, Atolia (USA), Ascension (Bolivia), Hillgrow (Australia)
Placers	Magaden Area (Iultin), Transbaikal Region (Serl Mountain), Kazakhstan (Kara-Oba) in USSR., Atolia (USA), Bvabin, Heida (Burma) and also in China, Indonesia, Thailand, Congo and Bolivia

igneous rocks is in granites, particularly S and A types. Sedimentary rocks on an average contain 1 to 2 ppm tungsten.

Wolframite group of minerals and scheelite are the main source of tungsten. Ferberite (FeWO_4) and huebnerite (MnWO_4) form an isomorphous series, with wolframite being the intermediate member. Scheelite (CaWO_4) often contains appreciable amounts of molybdenum either as isomorphous substitution or in the form of powellite [$\text{Ca}(\text{MoW})\text{O}_4$]. Cuprotungstite (CuWO_4), cuproscheelite [$(\text{CaCu})\text{WO}_4$], and stolzite (PbWO_4) are some of the uncommon tungsten minerals, while tungstite (WO_3), ferritungstite ($\text{Fe}_2\text{O}_3 \cdot \text{WO}_3 \cdot 6\text{H}_2\text{O}$), anthoinite [$\text{Al}(\text{WO}_4)(\text{OH})\text{H}_2\text{O}$] and mpororite are some of the secondary minerals.

3. Tungsten deposits

Both endogenous and exogenous tungsten deposits occur, the former types, however, dominate. Endogenous types are classified into four types according to Russian geologists, namely (i) skarn, (ii) greisen, (iii) plutonogenic hydrothermal (or granite related) and (iv) volcanogenic hydrothermal. Exogenous deposits are mainly represented by placer concentrations. Granite-related deposits occur in four main inter gradational genetic types, viz. magmatic disseminations, pegmatites, porphyries and veins, and are typically associated with a host of other characteristic metals like Sn, Mo, Ta, Bi etc. Table 1 gives some of the major world occurrences of these principle types of tungsten deposits (Anon 1988; Padmanabhan *et al* 1990).

Figure 1. Generalized flow-sheet for the beneficiation of tungsten ores.

4. Processing of tungsten ores: International practice

Most of the tungsten ores exploited world over have grades of $>0.5\% \text{ WO}_3$. When lower grade ores are worked, the economics is linked to some by-product minerals. Tungsten ore deposits can consist of simple wolframite, simple scheelite or a wolframite-scheelite combination. The beneficiation flow-sheet followed largely depends on the nature of mineralization in the ore body, and the size of liberation of the tungsten minerals. However the beneficiation process generally consists of a pre-concentration step after crushing and grinding of the ROM ore, followed by processing the pre-concentrate, concentrate cleaning or up-gradation step, and a final purification stage to meet the market specifications. A generalized flow-sheet (Padmanabhan *et al* 1990) generally followed for processing tungsten ores, is schematically shown in figure 1.

A variety of pre-concentration processes are employed to reject the bulk of barren and low grade waste prior to the main processing. Hand-sorting (Baldia *et al* 1984) is preferred for pre-concentration in many countries. Mechanical sorters based on the reflectivity differences between the minerals in the case of wolframite, and ultraviolet ore sorters in the case of fluorescent scheelite are extensively made use of (Anon 1979; Ermolenko *et al* 1985; Minonov *et al* 1987; Zhaboev *et al* 1987). However, gravity

separation is the most common pre-concentration method in a very large number of tungsten ore processing plants (Borchers 1979; Ottley 1979). The high density of both scheelite and wolframite facilitates their easy separation by gravity techniques. Scalping jigs, shaking tables, spirals and Reichert cones are widely used for pre-concentration. Heavy media separation in cyclones, using ferrosilicon as the medium, is used in the plant at Panasqueira, Portugal. Magnetic separation techniques are also used for pre-concentration, both for scheelite (non-magnetic) and wolframite (strongly paramagnetic) bearing ores. For example, low intensity magnetic separation is employed in Uludag mine in Turkey for the removal of magnetite, and high intensity magnetic separation for the rejection of garnet and other paramagnetic gangue minerals (Karahan *et al* 1980). Although flotation or reverse flotation is commonly practiced in concentrate cleaning stages, its use in the pre-concentration stage is being increasingly advocated, especially in cases where gravity methods fail to give satisfactory results. Scheelite floats comparatively easily compared to wolframite and a number of scheelite producers use flotation to advantage (Mitchell *et al* 1951; Babok and Viduetskii 1967; Auge *et al* 1975; Vasquez *et al* 1976; Texeira *et al* 1988). A number of reagents have also been tried for the flotation of wolframite, and a comprehensive literature on this is available elsewhere (Rao 1991; TRDDC 1991). Specially designed separators as well as new technologies are being suggested in pre-concentration of fine-sized tungsten mineral particles, viz. centrifugal separators (Han and Say 1985), shear flocculation (Warren 1975a,b; Jarrett and Warren 1977; Shao and Shi 1986; Koh *et al* 1986; Rao G V 1987), spherical agglomeration (Dawei *et al* 1986; Kelsall and Pitt 1987), high gradient magnetic separation (Gak *et al* 1983; Sun Shanlun *et al* 1984; Svoboda 1988), etc.

Processing of the pre-concentrate is generally a multistage operation, involving size reduction for near complete liberation, classification, gravity separation, direct or reverse flotation, and magnetic and high tension separations (Padmanabhan *et al* 1990). Wolframite is paramagnetic and electrically conducting, hence high intensity magnetic separation and high tension separation would be able to upgrade it. Scheelite is both non-magnetic and non-conducting. Sulphide minerals are normally removed from the pre-concentrate by reverse flotation, wherein they are floated while the tungsten values remain in the flotation sink. Pyrrhotite has poor flotation response, but it can be removed by low intensity magnetic separation. If pyrite proves difficult for removal by flotation, the pre-concentrate containing pyrite can be roasted, which converts pyrite into a magnetic form, separable by magnetic separation. Tables, and vanners are also used for upgradation of tungsten mineral concentrates.

Tungsten beneficiation plants normally operate with a recovery of 60–85% (Padmanabhan *et al* 1990). Most of the losses of tungsten occur in slimes, which are not amenable for normal gravity separation methods. Generation of tungsten mineral slimes occurs due to two reasons. First, tungsten minerals being friable in nature, are highly susceptible to differential grinding, and consequently get ground preferentially during crushing and grinding. Secondly, because of the high density of tungsten minerals, they tend to go into over-size fraction during classification by cyclones or hydraulic type of classifiers used in the grinding circuit, and get recycled to the grinding mill, leading to their over-grinding. To reduce excessive generation of slimes multistage crushing and grinding are often employed. One of the fundamental practice followed in modern milling is to size the ore as it passes through the plant and recover as much of the metal values as possible from each size at each stage. In other words the strategy is

to have size reduction and recovery in stages to meet the philosophy of "the earlier recovered, the more recovered" (Weisun 1982) and to avoid difficulty for treatment of slimes.

The market specifications for the concentrates demand a minimum grade of 65% and 70% WO_3 respectively for wolframite and scheelite concentrates, with stringent controls of sulphur, phosphorous and silica contents (Stafford 1985). However, in cases where marketable grade concentrates can not be achieved at reasonable recovery, a scheme of 'forward integration' is advocated (Borchers 1979). This scheme is slowly replacing the established system of separate centre of tungsten activity (viz. mining and ore dressing; metallurgical extraction; metal, carbide and ferro-tungsten production; fabrication, etc.). In situations where further upgrading of the low grade concentrate is constrained by increasing losses, the economic level concentrate produced is processed by the application of modern chemical extraction technology to achieve maximum recovery. A discussion on chemical beneficiation practices is beyond the scope of this review; however, a brief reference to the work carried out in India on this aspect is made at the end under 'General discussion'.

5. The Indian scene

The wide gap between demand and indigenous supply of tungsten raw materials, and the strategic status accorded to tungsten world over, a great emphasis has been placed in India on improving the tungsten resource base. A flurry of activities on exploration for new tungsten ore deposits, feasibility studies on the beneficiation of indigenous ore resources as well as extraction of metal, are in progress in many national laboratories and institutes. As a result of this intensive activity an *in situ* geological resource of nearly 30,000 tonnes of tungsten metal at a cut-off grade of 0.1% have now been established (see table 2). The information contained in table 2 is called from various sources, mainly from the Proceedings of the National Workshop on Tungsten Resources Development held at Bhubaneswar in 1987.

The major ore deposits are located (Sehgal and Satyanarayana 1987) in Degana in Rajasthan, Khobna–Kuhi belt in Maharashtra and Burugubanda–Tapaskonda belt in Andhra Pradesh. A potential major deposit is in the Jaurasi–Koerali belt of Almora District in UP. Deposits of minor importance are those occurring in Balda, Deva Ka Bara and Pali in Rajasthan, Agargoan and Kolari–Bhaonri in Maharashtra, Bankura in West Bengal, Madurai in Tamilnadu, Attapadi in Kerala and Gadag in Karnataka. Besides, the gold ores of Kolar (Anon 1985) and Hutti (Raju *et al* 1987) in Karnataka contain significant concentrations of scheelite potentially recoverable as a by-product of gold.

However most of these deposits are in the low grade category, the tenor varying between 0.1 and 0.2% WO_3 , as against the commonly exploited grade of 0.5% or more world over. The challenge therefore lies in establishing the feasibility of economic exploitation of these low grade resources. In order to achieve this goal, an 'ore to product' integrated approach (Rama Rao 1990) is called for, which visualizes an integrated strategy for the development of exploration, exploitation, extraction, production and utilization of the indigenous resources of strategic metals. Beneficiation forms an important and integral component of the exploitation strategy. In the case of tungsten, though resources have been established in the recent years, their exploitation

Table 2. Major tungsten deposits in India.

Locality	Type of deposit	W. mineral	Tenor range WO ₃ %	Resources million tonnes	Contained WO ₃ tonnes
1. Degana (Rajasthan)	Quartz veins in granite	Wolframite	0.2–0.5	25	425
	Disseminations in granite	Wolframite	0.03–0.12	168	134000
	Stockwork in phyllite	Wolframite	0.01–0.05	2.70	675
	Eluvial placer	Wolframite	0.01–0.06	3.30	1320
2. Balda (Rajasthan)	Greisenised pegmatite & granite	Wolframite	0.1–0.5	0.15	370
3. Khobna-Kuhi (Maharashtra)	Greisen & vein	Scheelite Wolframite (minor)	0.15–0.4	2.73	8190
4. Burugubanda- Tapaskonda (A.P.)	Pegmatite veins in Khondalite	Wolframite	0.1–0.16	11.0	16500
5. KGF (Karnataka)	Tailing dumps	Scheelite	0.04–0.2	1.0	720
	Gold-bearing lodes	Scheelite	0.07–0.2	0.37	700
6. Hutti (Karnataka)	Gold-bearing lodes	Scheelite	0.08	1.08	850
7. Almora (U. P.)	Skarn	Scheelite	0.1–0.15	30	3750
8. Kambalipatti- Rayapatti (Tamilnadu)	Skarn	Scheelite	0.1–0.3		
9. Bankura	Quartz veins	Wolframite			
10. Deva-ka-Bara (Rajasthan)	Skarns	Scheelite	0.14–0.4	0.06	130
11. Agargaon (Maharashtra)	Quartz-veins	Wolframite Scheelite	0.04–0.27	2.23	1300

have been held up due to non development of an techno-economically viable strategy for beneficiation. Considerable amount of research and developmental efforts in this direction have been continuing in several national laboratories. A review of these studies follows. Almost all of the potential tungsten ore deposits have been investigated at the Ore Dressing Section laboratory of Bhabha Atomic Research Centre, and these are covered in greater detail, while available information of the efforts at other laboratories are briefly discussed. Reports of the Strategic Minerals (Group XII) Sub-Committee of Geological Survey of India and of the BRGM (France) – MECL (India) Tin–Tungsten Collaborative Programme have also been extensively made use of in gathering the information on the beneficiation of Indian tungsten ores. Wherever possible reference to appropriate reports are made.

The beneficiation characteristics of the following tungsten ores of the country are covered in this review: (i) Degana, (ii) Balda, (iii) Khobna, (iv) Burugubanda–Tapas-konda, (v) Kolar and Hutti and (vi) Madurai.

6. Wolframite ore from Degana, Rajasthan

Degana is the only tungsten mine in the country where mining and production of wolframite concentrate, albeit in minor quantities, has been continuing from the year 1916 (Bansal *et al* 1987). The Degana deposit consists of three small hills near Rewat village 5 km west of Degana Railway Station in the Nagaur District of Rajasthan. Here the older phyllites of the Delhi Super Group are intruded by the Degana Granite. Wolframite-bearing quartz veins have intruded the granites as well as the older phyllites. While the quartz veins are the main carriers of wolframite, both the granite and part of the phyllite contain disseminated wolframite brought in by the impregnating siliceous solutions. On the slopes of the hills also occur wolframite bearing eluvial gravel bed, derived by the weathering of the granite. The gravel bed is consolidated by calcareous impregnations.

Five types of resources of wolframite in the Degana area are identified. These are (i) quartz vein type, (ii) the granite type, (iii) the phyllite type, (iv) the eluvial type and (v) the tailing and waste dumps of the earlier workings. Two categories of granite-type are recognized, a Trench lode granite which forms the broad zone of the granite encompassing the quartz veins, and the granite as a whole. The average grades and estimated resources in the different type of deposits (Bansal *et al* 1987; Patni *et al* 1987; Sehgal and Satyanarayana 1987; Anon 1988) are included in table 2, and their mineralogical composition (of representative samples investigated in BARC) in table 3.

6.1 Quartz vein type ore

The Ore Dressing Section, BARC has investigated in detail (Padmanabhan *et al* 1984, 1985) the beneficiation characteristics of a representative ore sample of the quartz vein

Table 3. Mineralogical composition of tungsten ores of Degana (Ore samples investigated in BARC) (All in %).

	Eluvial		Granite	Quartz vein	Phyllite
	Sample 1 (Siliceous)	Sample 2 (Calcareous)			
Quartz and felspar	77	51	62.5	76	85
Calcite and calcareous cement	15	38			
Topaz	4	6	13.5	9	7
Mica	3	3.5	23.4	11	3
Other transparent minerals	0.5	0.5	0.2	1	0.5
Opaque minerals (Incl. Wolframite)	0.5	1.0	0.4	3	2.5
Assay WO ₃	0.04	0.023	0.04	0.26	0.013

Figure 2. BARC flow-sheet (schematic) for the beneficiation of quartz vein type tungsten ore from Degana.

type made available by the then RSTDC, of average assay 0.27% WO_3 in the form of wolframite (with huebnerite: ferberite ratio being about 1). Quartz (76%), muscovite-zinnwaldite mica (11%) and topaz (9%) were the major minerals in the ore. Ore minerals identified, besides wolframite are, pyrrhotite, pyrite, chalcopyrite, ilmenite, magnetite, hematite, cassiterite, bismuthinite and native bismuth. Minor fluorite and trace monazite and zircon are also present. A trimodal grain size distribution of wolframite is observed: (i) coarse crystals of 2 to 8 mm size, (ii) medium sized grains or aggregates of platy crystals of size 50 to 250 μm , and (iii) very fine skeletal crystals of 10 to 25 μm size, generally occurring as inclusions within mica. The first two categories account for nearly 90% of tungsten distribution.

While developing a flow-sheet to beneficiate this ore the following aspects were kept in mind: (i) a substantial part of the tungsten values occur in sizes coarser than 2 mm, (ii) fine/crushing grinding of the ore to near complete liberation of wolframite at one stretch would lead to excessive wolframite slime generation, which is undesirable, and (iii) it is advisable to reject bulk of barren material at a relatively coarse size to reduce cost of grinding.

The flow-sheet developed to process the ore (Padmanabhan *et al* 1984) is schematically shown in figure 2. The ROM ore is stage crushed to all passing through 6 mm screen, and classified into two streams on a 28# screen. The coarse stream is processed by two stage jigging, and the cleaner jig concentrate is further concentrated by tabling after grinding to all passing through 28#. Sulphide minerals from the table concentrate are removed by flotation, followed by high intensity magnetic separation of the flotation sink, which gives a magnetic wolframite concentrate. The original screen undersize is processed by tabling followed by flotation and magnetic separation as

Table 4. Summary of results of semi pilot plant scale test on quartz-vein type ore (Flow-sheet given in figure 2).

	Product	Weight %	WO ₃ Assay %	WO ₃ Dist. %
Feed		100.0	0.264	100.0
+ 28 #	Coarse Feed	80.7	0.25	76.6
- 28 #	Fine Feed	19.3	0.32	23.4
+ 28 #	Cleaner Jig Conct.	1.6	9.40	57.0
+ 28 #	Cleaner Jig Tails	15.4	0.13	7.6
+ 28 #	Rougher Jig Tails	63.7	0.05	12.0
+ 28 #	Table Conct.	0.23	55.60	48.5
+ 28 #	Table Tails	1.37	1.64	8.5
+ 28 #	Sulphide Float	0.03	2.50	0.3
+ 28 #	Nonmag. Fraction	0.01	5.10	0.2
+ 28 #	Magnetic Wolframite Conct.	0.19	66.60	48.0
- 28 #	Table Conct.	0.50	9.20	17.4
- 28 #	Table Tails	18.8	0.084	6.0
- 28 #	Sulphide Float	0.20	0.60	0.5
- 28 #	LIMS Magnetics	0.05	1.50	0.3
- 28 #	HIMS Nonmagnetics	0.18	1.10	0.7
- 28 #	HIMS Magnetics (Wolframite)	0.07	60.00	15.9
	Combined Wolframite Conct.	0.26	64.80	63.9

above. In a semi pilot plant scale test this flow-sheet gave a final combined wolframite concentrate of 64.8% WO₃ at a recovery of 64%. Material balance obtained in the laboratory test is given in table 4.

In industrial scale the rougher jig tails can be subjected to a stage of grinding and processed by spiralling followed by tabling of the spiral pre-concentrate, in order to improve the overall recovery. Overall a recovery of + 65% in a + 65% WO₃ grade concentrate is confidently expected by following this flow-sheet.

After detailed laboratory studies in the laboratory of Atomic Minerals Division, Dwivedy (1988) suggested three different flow-sheets for the beneficiation of the quartz vein type ore. Of the three, flow-sheet 1 is nearly identical to the BARC flow-sheet, which gave a final wolframite concentrate assaying 66% WO₃ at a recovery of 62%. The other two flow-sheets envisage grinding the ore to all passing through 35#, followed by pre-concentration either by gravity methods or by wet high intensity magnetic separation. Final concentrating steps are reverse flotation and magnetic separation as in the BARC flow-sheet. However these flow-sheets are only suggested flow-sheets, and have been tested up to the pre-concentration stage only. One of these suggests pre-concentration by WHIMS after size reduction to - 35#, and in the laboratory test a recovery of 72.6% in the WHIMS pre-concentrate at a grade of 16.9% WO₃ could be achieved.

Indian Bureau of Mines (IBM) have also conducted tests on quartz vein type ore sample (Datta 1987) following more or less the flow-sheet developed by BARC. Initially, a concentrate assaying 53% WO₃ could be obtained at an overall recovery of 35.5% only. Later, however, some modifications were tried and the results obtained were said to be better than BARC results. In these tests, while the basic flow-sheet of BARC was retained, the particle sizes for treatment at different stages were changed.

Figure 3. Flow-sheet (schematic) followed by NML for the recovery of wolframite from waste dump from Degana, along with metallurgical material balance at important pre-concentration stages.

A pilot plant test following a flow-sheet consisting of spiralling, tabling, reverse flotation and magnetic separation has reported an overall recovery of 68% at a grade of 68.5% (Sehgal and Satyanarayana 1987), an excellent result indeed!

6.2 Waste dumps

During mining and processing of quartz-vein lode type ore over the years, the rejects after manual hand picking and dry processing were accumulated in the form of waste dumps. These dumps are also considered to be important resource for tungsten as it analyses on an average 0.1% WO_3 . Both NML (1992) and IBM (Rao G M 1993) have studied in detail the recovery of wolframite from the waste dumps. Granulometric analysis of a representative waste dump sample by NML has shown that there is no preferential concentration of tungsten values in any of the size fractions and hence

rejection by screening of any size fraction is out of question. The major findings of the NML study are:

- (i) Grinding to an optimum size is essential for liberation, and overgrinding needs to be avoided to reduce loss of wolframite values in slimes.
- (ii) WHIMS has been found to be very effective method for pre-concentration and produces a clean tail with the rejection of about 90% of mass of feed.
- (iii) Classification before WHIMS and processing the coarse and fines streams separately improves efficiency of separation in WHIMS.
- (iv) A two-stage flotation of sulphides, one in acidic circuit and the other in alkaline circuit is necessary to remove the sulphides effectively.
- (v) It would be advisable to aim at an economic level of concentration of 5–20% WO_3 followed by chemical beneficiation, for optimum economic recovery.

The flow-sheet worked out by NML for the processing of waste dump ore is schematically shown in figure 3. The final recovery will depend on the grade of the pre-concentrate; at 5% WO_3 a recovery of > 60% could be achieved, and at 30% WO_3 the recovery falls to about 46%. Rao (1993) reports results of tests carried out by IBM on two waste dump samples, analysing 0.14 and 0.15% WO_3 . From the first sample, by a process involving tabling, flotation, magnetic separation and separation in Mozley mineral separator, a final concentrate of 66.50% WO_3 at 59.8% recovery was obtained, while from the second sample the final concentrate assayed 66.3% WO_3 by a process involving tabling, flotation and retabling. The results are excellent indeed compared to what was achieved in the NML study, however the latter sample assayed only 0.08% WO_3 and for liberation a much finer grind was necessary.

6.3 Eluvial gravel

The eluvial tungsten ore from Degana is perhaps the most extensively tested among the different types of ore from Degana. Investigations have been carried out by National Metallurgical Laboratory (NML), BARC, Regional Research Laboratory, Bhubaneswar (RRL-B), IBM and by the Golder Moffit Associates (GMA) of UK. The overall average grade of the eluvial gravel bed is estimated to be 0.02 to 0.04% WO_3 (Patni *et al* 1987), even though some of the samples from the gravel bed tested assayed as high as 0.11% WO_3 (Datta 1987).

During the early fifties NML investigated the feasibility of beneficiation of a sample of the eluvial gravel analyzing 0.11% WO_3 (Banerjee and Narayanan 1952). By a combination of screening, jigging, tabling and magnetic separation a product analyzing 56.6% WO_3 with 57.7% distribution was obtained. Alternatively tabling followed by magnetic separation after size reduction to all passing through 35# produced a concentrate assaying 66.3% WO_3 with 48.8% distribution. NML also studied another sample analyzing 0.052% WO_3 during the early seventies (Kunwar *et al* 1972). While tabling and magnetic separation at 48# size yielded a concentrate assaying 13.6% WO_3 with 60% recovery, a complicated flow-sheet involving a combination of hydraulic classification, tabling, high tension separation, reduction roast followed by magnetic separation with 48# feed, produced a concentrate assaying 36.0% WO_3 with 50% recovery.

BARC carried out the tests during the early seventies. Two ore samples, with differing mineralogical composition, one with high siliceous gangue and the other with

Figure 4. Flow-sheet for the recovery of wolframite from the eluvial ore (calcareous matrix) from Degana, with metallurgical material balance.

appreciable calcareous gangue were investigated (Ghosh *et al* 1968; Narasimham *et al* 1972). These analyzed 0.04% and 0.023% WO_3 respectively. Mineralogical composition of representative samples of the two types are included in table 3.

An important feature of the flow-sheet developed is the rejection of bulk of the gangue minerals during the pre-concentration stage by the use of a dry pneumatic device designed and fabricated in-house. The ore after being reduced in stages to all passing through 10 mesh screen, is classified into two fractions, + 30# and - 30#. Both the fractions pre-concentrated in the pneumatic device. The coarser pre-concentrate is processed on jigs in two stages and the cleaner jig concentrate was upgraded by magnetic separation, after grinding to all passing through about 65#. The finer pre-concentrate was cleaned on wet shaking tables, and further upgraded by magnetic separation, followed by high tension separation. This flow-sheet, schematically shown in figure 4, is followed in the case of the calcareous type of ore sample. From a feed grade of 0.023% WO_3 a final concentrate assaying 61% WO_3 could be obtained at nearly 80% recovery. The high recovery was possible due to the fact that in the eluvial ore wolframite occurs in a near liberated state, and that during size reduction to all passing through 10# in this 'calcareous'-cemented sample further sliming of the brittle wolframite does not take place.

Figure 5. Flow-sheet for the recovery of wolframite from the eluvial ore (siliceous matrix) from Degana, with metallurgical material balance.

The 'siliceous' ore sample was, on the other hand, more complex. Firstly the $-10+30\#$ and $-30\#$ fractions in the ROM differed in mineralogy and other characteristics from similar size fractions obtained after crushing the ROM coarser sizes, and hence had to be processed separately. Secondly the hard nature of the silica cemented ore necessitated considerable crushing to reduce the ore to all passing through 10#, and this resulted in generation of considerable slimy wolframite affecting its recovery. The flow-sheet adopted to process this type of the ore is schematically shown in figure 5. Overall a recovery of only 62% could be achieved at 53% WO₃ grade. The study has also shown that cassiterite and monazite are the potential byproducts of processing the eluvial ore. Both these minerals are concentrated during pre-concentration by gravity methods and are separated from wolframite only during final magnetic and high tension separation stages.

The investigations clearly showed that (i) processing should be done at the coarsest size possible compatible with the grain size of wolframite, so as to avoid sliming of brittle wolframite, (ii) some sort of pre-concentration, preferably in a dry process, at a relatively coarse size, is necessary not only to make the subsequent concentration

process more efficient, but also to make the beneficiation process economically viable, and (iii) the grade of the deposit varies considerably and in order to develop an industrial flow-sheet a proper blending of the ROM ore is essential.

The bench scale work carried out by M/s Golder Moffit Associates of U.K. in the early eighties, at the request of Mineral Development Board, confirmed the need for pre-concentration at a relatively coarse size, and desirability of desliming before treatment. However their test work produced a pre-concentrate assaying 5.4% WO_3 and 0.4% Sn at recoveries of 54.6% and 43.3% respectively (Lahiri and Patni 1988).

Subsequently Regional Research Laboratory, Bhubaneswar took up further work on the lines suggested by Golder Associates. These studies (Narasimhan K S 1988) have shown the feasibility of application of dry magnetic separation for the pre-concentration of wolframite, after size reduction of the ore to – 2 mm size. The finding was substantiated through bulk processing of 8 tonnes of the gravel sample using disc type high intensity dry magnetic separation at 1.2 Tesla magnetic field intensity. From a sample of feed grade 0.02% WO_3 , a pre-concentrate assaying 0.4% WO_3 at an estimated recovery of 60% was achieved. According to Narasimhan (1988) the separation at magnetic intensity of 1.6 Tesla magnetic intensity can improve the recovery. A test carried out by M/s CARPCO by two-stage magnetic separation using induced rolls and lift type magnetic separators, and reported by Narasimhan (1988), gave nearly 86% recovery at a grade of 1% WO_3 , though the feed grade of the sample tested was high, at 0.18% WO_3 . The pre-concentrate obtained from the eluvials is expected to be amenable to further enrichment in the same way as the wolframite-bearing quartz vein type of ore.

6.4 *Phyllite type ore*

Two samples of the so called phyllite ore, made available by Rajasthan State Tungsten Development Corporation, were studied by BARC (Shukla *et al* 1985). Essentially the samples were micaceous quartzites affected by pneumatolytic alteration, and assayed 0.013% and 0.022% WO_3 . Mineralogically both the samples were similar, analyzing, quartz 85%, mica 3%, topaz 7%, and opaque minerals including iron oxides 3%. Liberation analysis indicated that over 60% of tungsten values were liberated at a grind of 80% passing through 65#. Ore samples ground to about 50% passing through 200# were hydraulically classified into three fractions, each of which were processed on wet shaking tables in two stages, separately. Sulphides from the cleaner table concentrates were floated, and the flotation sink subjected to magnetic separation in a laboratory WHIMS. The final concentrates assayed 11 to 13% WO_3 at overall recoveries of 45 to 48%, in the two samples. The results are summarized in table 5.

6.5 *Granite type ore*

Exploration work undertaken by Rajasthan State Tungsten Development Corporation followed by geostatistical analysis during mid eighties have indicated that the granite has been mineralized more or less uniformly, and that the average tenor is about 0.08% WO_3 in a total of 169 million tonnes of ore (Patni *et al* 1987). Notwithstanding its low tenor, exploitation of the granite presents a challenge in view of the sheer vastness of the deposit. Studies have been carried out both at Indian Bureau of Mines

Table 5. Results of beneficiation of phyllite type ore from Degana.

	Expt. 1	Expt. 2	Expt. 3	Expt. 5
Feed Assay (WO ₃ %)	0.013	0.013	0.013	0.013
<i>Tabling Stage</i>				
Concentrate Wt. %	0.82	3.58	5.13	1.37
Concentrate Grade (WO ₃ %)	0.85	0.24	0.19	0.57
Overall Recovery %	54	66	76	60
<i>Sulphide Flotation Stage</i>				
Concentrate Wt. %			4.61	1.21
Concentrate Grade (WO ₃ %)			0.20	0.63
Overall Recovery %			71	58.8
<i>Cleaner Tabling Stage</i>				
Concentrate Wt. %				0.215
Concentrate Grade (WO ₃ %)				3.26
Overall Recovery %				54
<i>Magnetic Separation Stage</i>				
Concentrate Wt. %		0.381	0.341	0.057
Concentrate Grade (WO ₃ %)		2.15	2.15	11
Overall Recovery %		63	56	48

and at BARC on the feasibility of beneficiation of this type of the ore. While the ore samples tested by IBM assayed 0.08 to 0.09% WO₃, the bulk samples sent by RSTDC to BARC for testing analyzed 0.035 to 0.05% WO₃ only.

The Degana granite ore consists of pink, buff, grey and dark coloured varieties of coarse to medium-grained granite and fine-grained aplite. Mesocratic types grade into highly siliceous leucocratic varieties. All the varieties show extensive greisenization. Mineralogically orthoclase, microperthite, sodic plagioclase and quartz are the essential minerals. Muscovite mica, biotite and topaz are the main accessory constituents. Most of the muscovite appear to be the zinnwaldite variety. Due to greisenization some of the feldspar and quartz have been replaced by topaz and zinnwaldite. Minor fluorite, tourmaline and calcite are also observed. Most of muscovite shows violet tinge in thicker flakes, and are often studded with pleochroic haloes around monazite and zircon inclusions.

Wolframite occurs in the form of medium to very fine sized subhedral to anhedral grains (varying in size from 150 to below 10 μ m), introduced along minute fractures, grain boundaries and rain interstices of felsic minerals and cleavage planes of mica and topaz. Other opaque minerals observed are magnetite, ilmenite, pyrite, pyrrhotite and trace chalcopyrite. Typical mineralogical composition of a sample of the ore investigated by BARC (Shukla *et al* 1988) is given in table 3.

6.6 IBM test results

Three samples of the granitic type of ore from different parts of the Degana granite body, with a feed grade varying between 0.07 to 0.09% WO₃ have been studied by IBM (Rao and Satyanarayana 1987). The main process adopted for the tests is identical, size reduction to all passing through 65#, classification and tabling, flotation of combined table concentrate to remove sulphides, followed by magnetic separation of the flotation

Table 6. Summary of IBM test results on granite type ore.

Product	Weight %	WO ₃ Assay %	WO ₃ Dist. %
<i>Test No. 1: Initial Grind — 65 #</i> (Feed Assay 0.087%)			
Combined Table Concentrate	1.10	4.02	50.44
Magnetic Concentrate	0.06	53.01	37.50
<i>Test No. 2: Initial Grind — 65 #</i> (Feed Assay 0.071%)			
Combined Table Concentrate	0.80	3.83	42.66
Magnetic Concentrate	0.053	40.14	31.12
Mozley Concentrate	0.026	59.04	23.24
Mozley Tailings	0.027	19.26	7.88
<i>Test No. 3: Initial Grind — 35 #</i> (Feed Assay 0.097%)			
Combined Table Concentrate	1.50	3.95	61.87
Magnetic Concentrate	0.119	43.99	54.95
<i>Test No. 4: Initial Grind — 35 #</i> (Feed Assay 0.084%)			
Spiral Pre-concentrate	44.18	0.164	80.46
Combined Table Concentrate	0.64	8.35	63.3
Magnetic Concentrate	0.081	57.35	56.46
Mozley Concentrate	0.054	65.60	43.05
Mozley Tails	0.027	40.85	13.41
<i>Test No. 5: Initial Grind — 65 #</i> (Feed Assay 0.084%)			
Final Magnetic Concentrate	0.081	58.38	58.67
Mozley Concentrate	0.063	64.64	48.67
Mozley Tails	0.019	36.12	7.80

Figure 6. IBM flow-sheet for the beneficiation of the Degana granite-type ore, with metallurgical material balance.

Table 7. Summary of BARC test results on granite type Degana ore (Feed Grade 0.04% WO₃).

Test No.	Initial grind	Major process steps	Gravity conct		Final conct	
			Grade WO ₃ %	Dist. WO ₃ %	Grade WO ₃ %	Dist. WO ₃ %
1	30% passing through 200 #	Hydraulic classification, tabling in two stages, cleaner table conc. upgraded in WHIMS	0.81	36.5	4.1	35.2
2	The coarse and medium fractions of hydraulic classification reground and processed as above		6.63	46.4	21.0	42.0
3	Similar to Test No. 2		6.57	45.2	18.6	44.1
4	90% passing through 150 #	Bartles Vanner in three stages	2.12	39.0	—	—
5	90% passing through 150 #	Pre-conct. in WHIMS, flotation of mica from mags., tabling of flotation sink, followed by final upgradation in WHIMS	9.60	35.8	16.7	35.8
6	90% passing through 150 #	Processed as above, flotation sink processed in vanner in two stages simulating BMS and CBC (Flow-sheet in figure 7)	13.0	45.0	21.2	44.0
7	90% passing through 150 #	Flotation of mica, followed by tabling of flotation sink, final up-gradation by WHIMS (Flow-sheet in figure 8)	10.0	44.0	22.1	43.7

sink. The feebly magnetic fraction is further upgraded by separation on Mozley Laboratory Mineral Separator. The final results obtained in the different experiments are tabulated in table 6. The sample no. 3 is also tested by the route of pre-concentration by spiralling followed by tabling of the spiral pre-concentrate. The flow-sheet followed is schematically given in figure 6. Samples reduced in size to all passing through 65# gave marginally superior result compared to the sample reduced to 35#. The former resulted in a final concentrate assaying 64.4% WO₃ grade at a recovery of 48.7% while the latter resulted in a grade of 65.6% at a recovery of 43.0%.

6.7 BARC test results

Detailed studies were carried out on two samples of granite ore made available by RSTDC. Both the samples were identical in their characteristics, and replicate analysis gave the head assay as $0.04 \pm 0.003\%$ WO₃. The sample also assayed 0.23% Li₂O, with lithium occurring in the form of zinnwaldite. Liberation study indicated that for

Figure 7. Flowchart of BARC Test 1 for the recovery of wolframite from the Degana granite-type ore, with metallurgical material balance.

optimum liberation of wolframite, the ore needs to be ground to all passing through 150# (100 μ m), which will liberate about 80–85% of the tungsten values in the ore (Shukla *et al* 1988).

Batch beneficiation tests using shaking tables with slime deck, followed by magnetic separation, gave 35–45% recovery of tungsten at a grade of about 20% WO₃ (Shukla *et al* 1988). Pre-concentration before subjecting to tabling was attempted on Bartles machines (Bartles Mozley Separator and Bartles Cross Belt Concentrator), and by wet high intensity magnetic separator. The latter gave superior results. Summary of results obtained in different schemes of tests are given in table 7. Results of two tests following alternate schemes of upgradation of the WHIMS pre-concentrate are schematically depicted in figures 7 and 8.

Based on a number of tests, an industrial flow-sheet is suggested (Shukla *et al* 1988). The flow-sheet schematically shown in figure 9 involves pre-concentration by WHIMS, followed by flotation of mica and gravity beneficiation of flotation sink. Since the ore needs fine grinding for optimum liberation, gravity beneficiation using Bartles Mozley Separator and Cross Belt Concentrator are suggested. Final upgradation is achieved by xanthate flotation of sulphides followed by magnetic separation. A final

Figure 8. Flowchart of BARC Test 2 for the recovery of wolframite from the Degana granite-type ore, with metallurgical material balance.

concentrate assaying 30–35% WO_3 at an overall recovery of 40–45% starting from a feed of tenor 0.04 to 0.05% WO_3 is expected. The highlight of the flow-sheet is a lithiferous mica product assaying 0.8 to 0.9% Li_2O , which can be upgraded to > 1% Li_2O . Another potential byproduct is a high grade topaz concentrate, Expected material balance is also indicated in the figure 9.

6.8 Discussion

With the exploitation of the meagre resources of quartz-vein type ore by underground mining becoming uneconomical, the future of Degana prospect depends on harnessing the vast granite-type ore. Notwithstanding the very low tenor of the ore samples tested by BARC, two distinct flow-sheets have been developed and tested—one by IBM, and the other by BARC. In addition NML is also carrying out exhaustive studies on the beneficiation of this ore under a NML–DRDO collaborative Project (Chakravorty, Personal Communication). While the IBM flow-sheet involves pre-concentration by gravity methods—spiralling followed by tabling, the BARC flow-sheet resorts to WHIMS followed by reverse flotation of mica for the pre-concentration. At first sight the IBM flow-sheet appears more attractive, but the techno-economic feasibility of either of the flow-sheets will depend on the particle size of liberation, and the grind needed for optimum liberation of wolframite. With relatively coarser grind of liberation (+ 100 to + 150 mesh), the IBM flow-sheet will be more advantageous; but with any finer grind necessary for liberation (– 150 mesh), this process is unlikely to yield high

Figure 9. Flow-sheet (schematic) suggested by BARC for the beneficiation of granite-type tungsten ore from Degana, with anticipated tungsten material balance.

recoveries owing to the use of spirals, which are not effective in fine sizes. In such cases the BARC flow-sheet will be more effective, as WHIMS is an efficient process even upto about 10 μ m particle size.

7. Balda tungsten deposit

The tungsten mineralization in the Balda area of Sirohi District of Rajasthan is associated with a leucogranite locally known as Balda Granite, which belongs to the Erinpura Granite Suite. The tungsten mineralization in the form of wolframite occurs in shear zones and greisenised pegmatites localized at the contact of metasediments and the intrusive granite. The quartz veins and greisenised pegmatites characteristically contain, besides quartz, alkali-felspar and sodic plagioclase, tourmaline, zinnwaldite,

Table 8. Summary of IBM test results on Balda tungsten ore (Greisen Type, Feed Grade 0.2% WO₃).

	Feed grade	Concentrate grade WO ₃ %	WO ₃ Recovery %	Major process steps
1.	0.197	16.0	40.12	Tabling, Mag. Sepn.
2.	0.21	23.56	51.54	Jigging, Tabling, HTS
3.	0.21	15.77	68.09	Jigging, Tabling (– 35 #)
4.	0.21	15.28	69.36	Jigging, Tabling
5.	0.21	26.08	57.92	Jigging, Tabling
6.	0.21	14.82	60.18	Jigging, Tabling
7.	0.44	45.3	70.7	Tabling
8.	0.17	37.58	39.60	Tabling (– 100 #)
9.	0.20	18.47	51.32	Tabling (– 35 #)
10.	0.20	10.78	49.23	Tabling (– 65 #)
11.	0.16	23.45	66.11	Tabling
12.	0.17	10.0	30.00	Tabling, Mozley LMS
13.	0.16	40.20	74.45	
14.	0.23	57.33	50.00	Tabling, HTS
15.	0.19	5.01	13.65	Tabling, Mozley LMS

Table 9. Summary of IBM test results on Balda tungsten ore (Granite Type, Feed Grade < 0.1% WO₃).

	Feed grade	Concentrate grade WO ₃ %	WO ₃ Recovery %	Major process steps
1.	0.081	19.00	50.00	Tabling, Mag. Sepn. (– 65 #)
2.	0.10	4.12	49.75	Gravity methods
3.	0.045	0.87	32.51	Tabling (– 100 #)
4.	0.035	0.84	30.2	Tabling
5.	0.08	0.79	42.38	Tabling (– 65 #)
6.	0.045	0.08	24.68	Tabling (– 100 #)
7.	0.08	6.24	34.0	Tabling, Mozley LMS
8.	0.045	0.16	53.43	Flotation
9.	0.007	2.13	36.78	Tabling, Mozley LMS
10.	0.015	4.0	39.26	Tabling, Mozley LMS
11.	0.013	0.59	27.5	Tabling (– 35 #)
12.	0.04	9.98	64.4	Tabling, Mozley LMS
13.	0.08	6.24	34.00	Flotation
14.	0.0086	2.17	20.95	Tabling, Mozley LMS
15.	0.03	2.17	6.5	Tabling, Mozley LMS

fluorite, topaz, apatite, rutile, arsenopyrite, pyrite, chalcopyrite and wolframite (Bhattacharjee *et al* 1987).

During mid- and late eighties, Indian Bureau of Mines had carried out indicative batch beneficiation tests on a number of samples from Balda (Rao and Subrahmanyam 1988). The process followed in most of the tests is the same, concentration by gravity beneficiation methods (jigging and/or tabling) after size reduction, followed by up-gradation by magnetic separation or electrostatic separation, and in some tests by separation in Mozley mineral separator. A few samples have also been processed by flotation, but the collector reagent used are not indicated. However none of the

Figure 10. Schematic flow-sheet for the beneficiation of Balda tungsten ore, followed in BARC test, with metallurgical material balance.

flotation tests have yielded any worthwhile concentration. Two types of samples have been tested, a lean grade granitic ore (WO_3 % < 0.01 to 0.10), and a medium grade vein sample (WO_3 % 0.16 to 0.23, and in one case 0.44%). The results obtained are summarized in tables 8 and 9. The medium grade samples have yielded very encouraging results, indicating their amenability for beneficiation.

The bulk sample of the Balda ore studied by BARC (Shukla and Rao – under publication) essentially consisted of quartz veins and greisenised pegmatite veins and had the following mineralogical composition: Quartz and feldspar 70%, Mica and Tourmaline 28%, Fluorite 1.5%, Wolframite and other opaque minerals 0.5%. The sample analyzed 0.30 to 0.35% WO_3 . The flow-sheet followed for the beneficiation of the ore is schematically illustrated in figure 10. The sample is crushed in a roll crusher to all passing through 6 mm screen. The crushed product is pre-concentrated in a laboratory jig. The pre-concentrate is screened at 48# and the screen coarse is further concentrated in a cleaner jig, and the screen fines in a shaking table. The tailings of the rougher jig was crushed to all passing through 10#, classified at 48#, and the coarse and fine fractions are processed on jigs and shaking tables respectively, in two stages, rougher and cleaner. During tabling a middling fraction was also collected. The final result obtained are also given in figure 10. The combined concentrate analyzed 60.5% WO_3 at 61.8% recovery, the combined middlings assayed 5.5% WO_3 at 24.6% distribution. It should be possible to improve the final concentrate recovery by recirculation of middling streams at appropriate points.

8. Khobna–Kuhi–Agargaon tungsten ore

Occurrence of a small tungsten deposit at Agargaon, falling in the Proterozoic Sakoli basin, was established during 1966–69 by Geological Survey of India, and later during 1978–79 tungsten occurrence in a number of localities in the basin was observed. This belt of tungsten occurrence is called the Kuhi–Khobna–Saongi belt (Bhoskar *et al* 1987). Geologically, tungsten mineralization is located in the north-western margin of the Sakoli synclinorium. The mineralization, predominantly scheelite with minor wolframite, is confined to greisen zones with swarms of tourmaline veins in a host of quartz–chlorite–mica schist.

In the Khobna area, the tungsten mineralization occurs in a zone called ‘Main Greisen Zone’, containing two types of greisen veins, viz. tourmaline-rich and quartz-rich greisens. Besides major amounts of quartz, mica and tourmaline, the greisen contains varying quantities of potash feldspar, chlorite, apatite, fluorite, calcite, sphene, topaz and opaque minerals. While scheelite is the predominant tungsten mineral in the oxidized zone of the greisens, in the primary zone wolframite is also frequently observed, though still scheelite is dominant. The opaque minerals identified include ilmenite, rutile, magnetite, hematite, pyrite, pyrrhotite, chalcopyrite, molybdenite,

Table 10. Mineralogical and chemical composition of Khobna tungsten ore (Ore samples investigated by BARC).

Mineral	Primary ore		Oxidized ore	
	I	II	I	II
Quartz and feldspars	59.9	64.5	64.5	
Micaceous minerals	27.7	29.1	27.3	
Tourmaline	9.0	2.9	3.2	
Garnet	0.7		0.2	
Apatite	0.1	1.1	0.2	
Calcite	0.1			
Fluorite			1.7	
Scheelite	0.4	0.3	0.25	0.25
Wolframite	0.2	0.01	0.05	0.01
Magnetite	0.35	0.17	2.10	
Other oxide minerals (Ilmenite, Rutile, Hematite, Goethite etc.)	0.37	0.85		
Other opaques (Sulphides)	1.13	1.07	0.40	
Pyrite	0.40	0.44		
Pyrrhotite	0.20	0.15		
Chalcopyrite, etc.	0.40	0.4		
Molybdenite	0.03	0.09		
Others	0.10	Tr		
<i>Chemical Analysis:</i>				
WO ₃	0.36	0.274	0.26	0.216
Cu	0.16		0.078	
Ni	0.005		0.005	
Co	0.004		0.003	
Mo	0.01		< 0.01	
Sn	< 0.1

Figure 11. Liberation of tungsten values in different size fractions of Khobna tungsten ore (primary type).

bismuth, bismuthinite, sphalerite, galena etc. in minor to trace amounts. Magnetite has given place to hematite and goethite in the oxidized zone. Also the sulphide mineral content in the oxidized zone is very minor.

Batch as well as semi pilot plant scale studies have been carried out on samples of ore collected from both oxidized and primary zones, by BRGM (France), IBM and BARC (ODS), at the request of MECL under BRGM-MECL Tin Tungsten Collaborative Programme. While the test results of BARC are described in detail in the following pages, the results obtained by BRGM and IBM are briefly discussed.

8.1 BARC test results

8.1a Ore characteristics: The mineralogical and chemical characteristics of two samples each from the oxidized and primary zone supplied by MECL are given in

Figure 12. Flow-sheet-I followed in BARC tests for beneficiation of Khobna tungsten ore.

table 10 (Narasimhan 1989; Shukla *et al* 1991). In both the samples at about $-65\#$ nearly 90% of the scheelite values are liberated, which can be taken as optimum liberation size. Liberation characteristic of the primary ore is illustrated in figure 11. Chemically the oxidized samples analyzed 0.26 and 0.22% WO_3 , while the primary zone samples analyzed 0.36 and 0.27% WO_3 .

8.1b Batch beneficiation tests: During the initial stages of testing (Shukla *et al* 1989), it was found that the high mica content of the samples was presenting problems in comminution. Grinding the sample to liberation size was resulting in undue over grinding of highly brittle scheelite, affecting its recovery in the subsequent gravity operations. Reduction in size to all passing through 16# (1 mm) by open circuit grinding in a peripheral discharge rod mill was found adequate to liberate most of the scheelite, at the same time avoiding over grinding of scheelite. Based on detailed batch tests, two flow-sheets were developed for large scale tests. These are schematically illustrated in figures 12 and 13 (Shukla *et al* 1991).

Flow-sheet I: Involves (i) size reduction in stages to all passing through 6 mm screen, (ii) screening at 20 mesh, (iii) pre-concentration of the coarse fraction by jigging followed by tabling of the jig concentrate after regrinding, (iv) pre-concentration of the

Figure 13. Flow-sheet-II followed in BARC tests for beneficiation of Khobna tungsten ore.

finer stream (– 20#) by spirals followed by tabling, and (v) open circuit grinding of the jig tails to all passing through 16# screen, followed by processing as in (iv) above.

Flow-sheet II: Involves (i) size reduction to all passing through 16# screen, (ii) classification at a cut size of about 35# and (iii) pre-concentration of both the coarse and fine streams by spiralling followed by tabling.

Upgradation: The pre-concentrates are upgraded by low intensity magnetic separation to remove magnetite and other iron oxide minerals, followed by xanthate flotation to remove sulphides, and finally by high intensity magnetic separation to concentrate scheelite in the nonmagnetic fraction. Any wolframite values present will be concentrated in the magnetic fraction of the last stage, along with tourmaline, garnet and some iron oxide minerals. This fraction can be further upgraded by high tension separation.

8.2 Large scale beneficiation tests

8.2a Tests with primary ore sample (sample I) (Test no. 1): This test was carried out following the flow-sheet I given above. At the end of the pre-concentration stages, the recoveries and assays of the pre-concentrates and the final concentrates obtained after upgradation by magnetic separation and flotation are given in table 11.

Table 11. Results of semi pilot plant scale tests on primary and oxidized ore samples from Khobna following flow-sheet I (figure 12).

Stream	Primary ore sample (Feed assay 0.27% WO ₃)			Oxidized ore sample (Feed assay 0.22% WO ₃)		
	Wt. %	WO ₃ assay %	WO ₃ dist. %	Wt. %	WO ₃ assay %	WO ₃ dist. %
<i>Pre-concentration stage</i>						
– 8 + 20 #, Jigging followed by tabling	0.28	22.5	21.4	0.09	36.0	16.8
– 8 + 20 #, Spiralling followed by tabling of reground jig tails	0.10	30.1	9.6	0.09	29.0	12.8
– 20 #, Spiralling followed by tabling	1.10	10.6	43.3	0.28	31.5	39.6
Total	1.48	14.2	74.3	0.46	31.9	68.2
<i>After upgradation</i>						
– 8 + 20 #, Jigging followed by tabling	0.09	60.3	19.0	0.05	65.0	15.2
– 8 + 20 #, Spiralling and tabling of reground jig tails,	0.04	70.0	9.1	0.04	54.0	11.5
– 20 #, Spiralling followed by tabling	0.16	64.3	39.4	0.13	62.2	38.2
Total	0.29	63.8	67.5	0.22	61.2	64.9

Another 8.6% of WO₃ values were recovered by scavenging the spiral middlings, though at a lower grade of about 2.1% WO₃. The results show that the primary jig tails carry appreciable WO₃ values in unliberated state, and that it is necessary to process the same after regrinding. Thus the flow-sheet does not in any way reduce the overall grinding requirements.

Test no. 2: In this test the flow-sheet II is followed. After size reduction in a roll crusher to all passing through 5 mm screen, the ore was ground in an open circuit rod mill to all passing through 1 mm. The rod mill discharge was classified in a stationary CTS screen with a separation size of 460 μ m, and both the coarse and fines streams were separately processed in spirals followed by tables. The grades and recoveries obtained at the end of the pre-concentrate stage and at the final stage are given in table 12.

It is clear that in order to obtain a better grade final scheelite concentrate, it is necessary to have higher grade pre-concentrates than what was obtained in this experiment, which means that during tabling stage weight collection should be optimized.

Test no. 3: During the test no. 2 it was observed that the feed to the rod mill (i.e. the roll crushed product) contained as much as 30% finer than 1 mm size, which resulted in over grinding of the already liberated scheelite in the rod mill. In this experiment this was avoided by pre-screening the roll crushed product through 1 mm screen, and subjecting only the screen oversize to grinding in the open circuit rod mill. Further processing was the same as in flow-sheet II. The results obtained at the pre-concentrate and the final stage after upgradation are also included in table 12. It was seen that even

Table 12. Results of semi pilot plant scale tests on primary ore samples from Khobna following flow-sheet II (figure 13).

Stream	At pre-concentration stage			After final upgradation		
	Wt. %	WO ₃ Assay %	WO ₃ Dist. %	Wt. %	WO ₃ Assay %	WO ₃ Dist. %
<i>Flow-sheet II:</i>						
+ 460 μ m, Spiralling followed by tabling	0.35	8.32	12.9	0.10	26.3	11.3
– 460 μ m, Spiralling	0.88	15.00	59.5	0.23	53.3	55.9
Total	1.23	13.10	72.4	0.33	45.1	67.2
<i>Modified flow-sheet II:</i>						
+ 460 μ m, Spiralling followed by tabling	0.10	26.3	11.3	0.18	24.3	17.8
– 460 μ m, Spiralling followed by tabling	0.23	53.3	55.9	0.09	48.0	16.8
Total	0.33	45.1	67.2	0.31	58.9	72.8

Table 13. Results of semi pilot plant scale tests on oxidized ore sample from Khobna following flow-sheet II (figure 13).

Stream	At pre-concentration stage			After final upgradation		
	Wt. %	WO ₃ Assay %	WO ₃ Dist. %	Wt. %	WO ₃ Assay %	WO ₃ Dist. %
<i>Flow-sheet II:</i>						
+ 460 μ m, Spiralling followed by tabling	0.13	16.9	10.8	0.06	36.8	10.6
– 460 μ m, Spiralling	0.40	32.1	60.7	0.20	60.5	58.5
Total	0.53	28.3	71.5	0.26	55.0	69.1
<i>Modified flow-sheet II:</i>						
+ 460 μ m, Spiralling followed by tabling	0.15	34.2	19.0	0.08	60.0	17.8
– 460 μ m, Spiralling followed by tabling	0.42	34.2	54.9	0.20	68.8	51.2
Total	0.57	34.2	73.9	0.28	66.2	69.0

though higher WO₃ distribution reported in the rougher spiral tails, better stage recoveries and grades of concentrates were realized.

Flow-sheet no. II as followed in test no. 3 has given the best result in terms of recovery. It should be possible to improve the grade by optimizing the weight per cent collection in the pre-concentration stage, or alternatively by subjecting the final concentrate to a further stage of cleaning on tables.

8.2b Tests with oxidized ore sample (sample 2) (Test no. 1): This test was carried out following Flow-sheet I. Because of the near total absence of sulphides and magnetite in

Table 14. Results of tests on upgradation of the concentrates.

Product	Sample 1 (primary ores)		Sample 2 (oxidized ore)	
	WO ₃ assay %	WO ₃ dist. %	WO ₃ assay %	WO ₃ dist. %
Table conct.	73.6	86.3	75.0	80.0
Table tails	13.6	13.7	29.4	20.0
Feed	45.8	100.0	57.2	100.0

Table 15. Mineralogical composition and partial chemical analyses of upgraded scheelite concentrates from Khobna (BARC Tests).

	Primary ore sample	Oxidized ore sample
<i>Mineralogical composition</i>		
Scheelite	92.0	95.0
Quartz	1.0	0.2
Mica	0.5	0.1
Tourmaline	0.2	0.2
Fluorite, Apatite and Topaz	3.3	3.0
Pyrite	2.0	1.0
Chalcopyrite etc.	1.0	0.1
Iron oxides	—	0.5
Total	100.0	100.0
<i>Chemical Analysis:</i>		
WO ₃	73.6	75.0
SiO ₂	1.73	0.59
S	1.34	0.46
Fe ₂ O ₃	1.89	1.07

this sample from the oxidized zone of mineralization, the stages of flotation of sulphides and low intensity magnetic separation were skipped. Grades and recoveries of the pre-concentrates and the final concentrates are given in table 11.

It may be noted that because of the near absence of wolframite in this sample, high stage recoveries were obtained in the final scheelite concentrates during magnetic separation.

Test no. 2: In this test the flow-sheet II was followed. Here too low intensity magnetic separation and flotation stages were omitted. The results obtained are given in table 13.

Test no. 3: This test followed the flow-sheet of test 3 with sample I, except for the sulphide flotation stage. The results after pre-concentration and final stages are given in table 13.

8.2c Final upgradation: The final scheelite concentrates obtained in the different experiments both with primary and oxidized samples, assay 45 to 66% WO₃ at recoveries between 65 to 73%. Microscopic examination of these concentrates revealed

Table 16. Cu, Mo and Ag contents of sulphide floats obtained from primary ore sample from Khobna (BARC Tests).

Test	Stream	Wt %	Cu %	Mo %	Ag ppm
No. 1	+ 20 # Float from jig concentrate	0.02	4.93	5.29	200
	– 20 # Float from cleaner table conc.	0.20	4.91	1.06	150
No. 2	+ 460 μ m, Float from cleaner table conc.	0.04	4.85	nd	
	– 460 μ m, Float from cleaner table conc.	0.11	4.66	1.10	120
	– 460 μ m, Float from cleaner table tails	0.08	4.63	1.11	120
No. 3	+ 460 μ m, Float from cleaner table conc.	0.02	4.61	4.85	140
	– 460 μ m, Float from cleaner table conc.	0.11	4.75	0.85	120

that the main contaminants are quartz, fluorite, topaz and minor sulphides, mainly pyrite. The marketable grade of scheelite concentrate is + 70% WO_3 and in order to achieve the same it is necessary to upgrade the concentrates. An attempt was made to do so by gravity methods, by subjecting the final concentrates to a stage of tabling (recleaner stage). This test was carried out by mixing the concentrates obtained in the different experiments. Table 14 gives the results.

Mineralogical and partial chemical analyses of the above table concentrates are given in table 15.

8.2d Improvements in recovery: There is scope to improve the recovery further by scavenging the spiral middlings, and by recirculating it along with table cleaner tails at appropriate stages. By this 4–5% increase in the recovery can be anticipated. Similarly the magnetic fraction of the high intensity magnetic separation stage constitutes an important low grade concentrate, which can be upgraded by gravity separation. Most of the tungsten values contained in them is due to wolframite. It may be noted here that in the primary sample, wolframite is often an important constituent. In one of the bore hole composite sample made available by MECL earlier, wolframite contributed nearly one third of the tungsten distribution in the sample.

8.2e By-products: The primary ore sample contains minor values of copper, molybdenum and silver, which have the potential to be recovered as valuable by-products. These values are concentrated in the float fraction of the sulphide flotation stage. The Cu, Mo and Ag contents of important sulphide floats obtained in the three tests with the primary sample are given in table 16. Under optimum conditions the total weight percent collection in all the flotation concentrate streams will be 0.25 to 0.3%. The average grade of these combined concentrates will be Cu 4 to 5%, Mo 1.4 to 1.6% and Ag 120 to 150 ppm. It should be possible to obtain a high grade molybdenite concentrate by differential flotation of the combined flotation concentrate.

Taking into account the results obtained during semi pilot plant scale tests and recirculation of the streams as suggested in the flow-sheet, it should be readily feasible

Table 17. IBM test results on Khobna ore (for flow-sheet see figure 15).

Product	Weight %	WO ₃ Assay %	WO ₃ Dist. %
Oxidized ore sample (feed assay 0.22% WO ₃):			
Mozley Conct. I	0.15	73.1	49.6
Mozley Tails I	0.12	3.5	1.9
Mozley Conct. II	0.38	6.0	10.3
Feed		0.22	
Primary ore sample (feed assay 0.27 % WO ₃):			
Mozley Conct. I	0.21	72.1	56.4
Mozley Tails I	0.04	4.6	0.6
Mozley Conct. II	0.23	7.0	6.0
Feed		0.27

Figure 16. Industrial flow-sheet (schematic) suggested by IBM for the beneficiation of Khobna tungsten ore.

The sulphide concentrates obtained in the case of the oxidized ore yielded grades of 6.0% Cu and 0.2% Mo, while those obtained with primary ore analyzed 11.0% Cu and 0.8% Mo.

Figure 17. Flow-sheet followed in the BRGM semi pilot plant scale test for the beneficiation of Khobna tungsten ore.

Based on the semi pilot scale tests IBM have suggested an industrial flow-sheet for the beneficiation of the Khobna ore (BRGM-MECL Rept. 1991), which is schematically shown in figure 16. The projected efficiency of this flow-sheet is 55% and 62% recovery respectively for the oxidized and the primary ore, at 70% WO_3 grade.

8.4 BRGM tests

BRGM (France) also carried out semi pilot plant scale tests on one bulk sample each from the oxidized and the primary zone. The former analyzed 0.290% WO₃, 0.077% Cu and 60 ppm Mo, and the latter 0.214% WO₃, 0.130% Cu and 262 ppm Mo. Besides scheelite (major) and wolframite (minor), small amounts of cuprotungstite is also observed to occur. The flow-sheet followed for semi pilot plant scale tests

Table 18. BRGM test results on Khobna Ore.

Product	Weight %	WO ₃ Assay %	WO ₃ Dist. %
Oxidized ore sample (feed assay 0.29% WO ₃):			
Final concentrate from spiral circuit	0.144	70.37	41.34
Final concentrate from scavenger circuit	0.015	45.60	3.59
Final concentrate from primary fines circuit	0.038	62.04	9.56
Combined concentrates	0.197	66.83	54.48
Combined high grade middlings	0.107	26.59	11.98
Total conct. + middlings	0.304	52.69	66.46
Primary ore sample (feed assay 0.214 % WO ₃):			
Final concentrate from spiral circuit	0.154	73.27	57.93
Final concentrate from scavenger circuit	0.024	68.00	6.54
Final concentrate from primary fines circuit	0.055	51.94	11.47
Combined concentrates	0.233	69.12	75.94
Combined high grade middlings	0.015	22.94	1.82
Total conct. + middlings	0.248	66.24	77.76

(BRGM-MECL Rept 1991) is schematically shown in figure 17. As can be seen, after size reduction to -1 mm, the ore is classified at $80\ \mu\text{m}$, the coarse stream is processed on spirals, and the spiral tailings after regrinding is scavenged on shaking tables. The primary slimes are processed on Mozley Multi Gravity Separator. The gravity pre-concentrates are upgraded successively by sulphide flotation, wet low intensity magnetic separation and wet high intensity magnetic separation. The test results are summarized in table 18.

Based on these results an industrial flow-sheet has been suggested by BRGM (BRGM-MECL Rept. 1991), which is schematically illustrated in figure 18. The process adopted is projected to have an efficiency to yield 65% and 70% WO₃ concentrate grades at 60% and 75% recoveries respectively from the oxidized and primary ore feeds.

8.5 Discussion

Tungsten in the Khobna ore occurs in the form of scheelite, with only minor values present in wolframite. An important observation made in the investigations by all the three laboratories is that the manner of initial size reduction has a critical role to play in obtaining optimum recovery of tungsten values. This is because, scheelite being highly brittle, gets easily over ground during size reduction, especially if close circuit ball milling is employed. In order to avoid excessive sliming of scheelite, both the BARC and BRGM flow-sheets suggest primary size reduction in rod mill, to all passing

Table 19. Mineralogical composition of Burugubanda tungsten ore (BARC Test Sample).

Mineral	Wt. per cent
Quartz and kaolinized feldspar	90.0
Graphite	6.0
Garnet	1.5
Sillimanite, topaz, zircon and monazite	0.5
Opagues (other than graphite)	2.0
Wolframite	0.13
Rutile and ilmenite	0.37
Hematite and other iron oxides	1.30
Sulphide minerals	0.20
Total	100.0

As in the case of Khobna, BRGM, IBM and BARC have carried out bench and semi pilot plant scale tests on representative sample of the ore from Burugubanda, collected and made available by NMDC. The results of BARC tests are described in detail here, while those of IBM and BRGM are discussed in brief.

9.1 BARC test results

Tests were carried out on a 5 tonne representative sample of the ore. Megascopically the ore contained two major lithological units (Sunilkumar and Rao 1991): (i) quartz-kaolinized feldspar-graphite schist and (ii) quartz veins and lenses with minor altered feldspar. The graphite-gneiss/schist is an aphanetic lepidoblastic medium-grained rock, a schistosity being imparted by the sheaf like aggregates of graphite flakes. Feldspar-quartz veinlets and quartz lenses of varying size occur along the schistosity of planes of the gneissic rock. Graphite and quartz are the major constituents, with minor amounts of altered feldspar, garnet, ore minerals, and traces of sillimanite, topaz, zircon and monazite. The ore minerals identified, besides wolframite, are rutile, ilmenite, magnesiochromite, hematite, lepidocrocite, goethite, and sulphides—pyrite, marcasite, pyrrhotite, chalcopyrite and sphalerite.

Wolframite has widely varying habit and grain size, in the form of coarse pinching and swelling veins of one to four cm in width, narrow veinlets of one to two mm width and as medium to fine sized disseminations in the quartz veins as well as graphite gneiss. Two types of wolframite grains are observed, one in the form of equidimensional grains and grain aggregates, and the other in the form of fine sheaf-like crystalline aggregates. Rutile is the nigrine variety (identified by BRGM), and is a characteristic and striking constituent of this ore. Quantitative mineralogical analysis of a representative sample of the ore is given in table 19.

Replicate chemical analysis of four separately drawn feed samples gave a mean WO_3 assay of 0.092% with a sample standard deviation of 0.008%. The fixed carbon content was 5.0%.

Liberation analysis showed that wolframite is coarsely liberated. A sample crushed to all passing through 8# contained about 61% liberated wolframite. In sizes finer than 210 μm (65#) wolframite was almost completely liberated except for few very fine

inclusions within quartz, accounting for about 2% of the overall distribution, which was considered irrecoverable. Work Index determined by Bond's method was 11.59 kwh per short ton or 12.77 kwh per metric ton.

9.1a Beneficiation tests: During batch beneficiation tests (Padmanabhan *et al* 1991; Rao *et al* 1992) the following strategies were adopted:

1. Separation of quartz-rich pegmatitic pieces from the rest (graphite-rich) by hand-sorting, and processing the two streams separately.
2. Classification of the crushed ore into different fractions and processing them separately.
3. Grinding the crushed ore in a rod mill, flotation of graphite from the ground ore and processing the flotation tailings for the recovery of wolframite.
4. Suitable sizing of the crushed ore into coarse and fines, processing the coarse stream by jigging and the fines stream by tabling after flotation of graphite.

Strategy 1: The details of the experiment carried out is schematically shown in figure 19. The hand sorting was carried out on + 6 mm fraction of the as received material, after reducing in size the coarse boulders manually. The hand-sorted quartz-rich fraction was crushed to all passing through 12.5 mm screen, then screened successively on 6 mm and 28# screens. The + 6 mm and the + 28# streams are processed on jigs, while the - 28# fraction was treated on shaking tables. The graphite-rich initial fines (- 6 mm) was screened through 28# screen, and the coarser and the finer fractions are processed by jigging and tabling routes respectively, the latter after flotation of graphite. The graphite-rich hand-sorted coarse was reduced in size to - 28# by grinding in a rod mill, and processed by shaking tables after graphite flotation. The material balance at the various pre-concentration stages are included in figure 19.

While these concentrates are not marketable beneficiation products, they nevertheless give a precise idea about the grade and recoveries that are feasible to be achieved during pre-concentration. It should also be possible to improve the recovery by recirculation of various middling products. The main contaminants in the jig concentrates were quartz, rutile, sulphides, iron oxides and garnet. The table concentrates had also similar mineralogical composition, except that the proportions of garnet and rutile were much higher. While quartz and garnet can be separated by high tension separation, iron oxides, pyrite and pyrrhotite can be removed by magnetic separation.

It was also noticed that while the jigging operations exhibited very high selectivity in respect of separation of wolframite, in the tabling operations the selectivity was rather poor, mainly because of the presence of high amounts of minerals like rutile and garnet.

Strategy 2 (Classification of the crushed ore followed by separate processing of the streams): In the previous experiment it was seen that during initial crushing, the wolframite values mainly remain in the coarser size fractions and that these are amenable to beneficiation by jigging. In this experimental scheme the ore feed crushed to all pass through 6 mm screen was classified into three size fractions, namely, - 6 + 1 mm, - 1 + 0.420 mm and - 0.420 mm fractions. The first two size fractions were processed by jigging, while the last fraction was subjected to graphite flotation first, and the flotation tails tabled to recover wolframite. The test strategy along with material balance at various pre-concentration stages is schematically shown in figure 20. Here also good selectivity was obtained during the jigging stage.

Figure 19. Flowchart of BARC semi pilot plant scale test on beneficiation of Burugubanda tungsten-graphite ore - Strategy 1, along with metallurgical material balance.

Strategy 3 (Grinding of the ore followed by graphite flotation and wolframite recovery from flotation tailings): This flow-sheet was primarily aimed at optimal recovery of graphite, with wolframite as a byproduct. Here the ore crushed to all passing through 6 mm was ground in a closed circuit pilot plant rod mill. Graphite from the ground ore was floated using kerosene as collector, pine oil as frother and sodium silicate as

Figure 20. Flowchart of BARC semi pilot plant scale test on beneficiation of Burugubanda tungsten-graphite ore - Strategy 2, along with metallurgical material balance.

depressant for the silicates. In the rougher float stage nearly 99% of graphite could be floated at a grade of 21.5% fixed carbon (F.C.). The float was cleaned three times, which improved the grade to 41% F.C. The rougher tails and the three cleaner tails are mixed and processed on shaking tables. The table concentrate assayed 9.2% WO_3 at a recovery of 28% only. The fine grinding of the sample (about 50% - 200#) perhaps resulted in over grinding of wolframite, leading to its lower recovery.

In an alternate scheme of the experiment, generation of excessive slimes was avoided by open circuit grinding, and an attempt was made to improve the efficiency of gravity pre-concentration by classifying the ground ore into coarse and fine fractions and processing the two fractions separately, after flotation of graphite, as was carried out in the previous experiment. The flotation tailings of the coarser stream was pre-concentrated on spirals followed by tables, while the fines stream was processed by two-stage tabling circuit. The classification was carried out on CTS stationary wet screens, at an effective d_{50} cut size of $135 \mu\text{m}$. The flow-sheet followed is schematically shown in figure 21 which also includes material balance at various stages.

Even though the results are better compared to the previous test without classification, they are still not satisfactory.

Strategy 4 (Separation of coarse wolframite by jigs and fine wolframite by spirals and tables after graphite flotation): In this scheme of experiment, schematically shown in

Figure 22. Flowchart of BARC semi pilot plant scale test on beneficiation of Burugubanda tungsten-graphite ore – Strategy 4, along with metallurgical material balance.

Table 20. Results of BARC Test No. 2 on Burugubanda ore following strategy 4 (for flow-sheet see figure 22).

Process streams	Weight %	WO ₃ assay %	WO ₃ dist. %
Feed	100.0	0.134	100.0
+ 16 mesh	60.6	0.15	67.7
- 16 mesh	39.4	0.11	32.3
Cleaner jig concnt. I (Hutch)	0.03	30.21	7.6
Cleaner jig concnt. II (Bed)	0.11	36.20	30.5
Jig concnt. III (scavenging jig tails)	0.12	2.94	2.6
Combined jig tails & - 16 mesh	99.7	0.075	59.3

Scope exists for improving the recovery by about 5% by recirculation of various middling streams in a continuous circuit. Recovery and grade obtained from the fines stream in the spiral-table circuit is poor, a feature observed in all the experiments under different schemes of processing. One of the reasons for the poor performance is the graphite coating onto the wolframite surface, affecting their hydrodynamic properties during thin film concentration prevailing in spirals and tables. Nevertheless with optimization and recirculation of middling streams it should be possible to improve both grade and recovery to about 12–15% and 30–35% (stage) respectively. The results also underscore the necessity to recover the tungsten values at as coarse size as possible and to avoid unnecessary overgrinding.

9.1b Upgradation of the gravity pre-concentrates: The main contaminants in the jig and table concentrates were garnet, rutile (nigrine variety), iron oxides, sulphides (pyrite, pyrrhotite and marcasite), zircon, monazite and quartz. Batch laboratory tests showed that while bulk of garnet and quartz could be separated by high tension separation, low intensity magnetic separation removed pyrrhotite and part of iron oxide minerals (altered magnetite). High intensity magnetic separation removed pyrite, remaining quartz and zircon in the nonmagnetic stream. Removal of rutile and part of the iron oxide minerals proved problematic and a multistage magnetic separation with repeated recirculation after careful size classification is necessary to reduce these contaminants. If the sulphide mineral content is high, they can be removed by flotation, before high tension and magnetic separation stages.

9.1c Byproducts: Graphite is the most important byproduct in the case of Burugubanda ore, which assays 5 to 7% fixed carbon. In some parts of the ore body it has the potential to be the main product, and tungsten byproduct. In the tests involving flotation of graphite, rougher floats assayed 25 to 30% F.C. carrying > 90% distribution. Two stages of cleaning by flotation improved the grade to 40 to 45% F.C. at about 85% overall recovery. Further cleaner stages of flotation with regrinding could upgrade the product to 70 to 75% F.C. To produce high grade concentrate of graphite, chemical leaching with HF was found necessary, which could improve the product to 90–92% F.C. Rutile (nigrine variety) is another potential byproduct of beneficiation of Burugubanda ore.

9.1d Industrial flow-sheet for beneficiation: Based on the results of several large scale tests, an industrial flow-sheet has been suggested (Padmanabhan *et al* 1991), which essentially follows the strategy 4 discussed above. The flow-sheet along with projected mass balance is given in figure 23. A recommended upgradation flow-sheet is illustrated in figure 24.

9.2 BRGM test results

BRGM carried out semi pilot plant scale tests on a 5 tonne representative sample of the ore from Burugubanda, analyzing 0.1% WO_3 and 7.6% fixed carbon. The flow-sheet adopted by BRGM (BRGM-MECL Rept. 1990; Save and Conil 1990) is schematically shown in figure 25. The procedure involved (i) crushing the ore to – 1 mm, (ii) screening at 315 μm , (iii) graphite flotation and spiral pre-concentration of the

Figure 23. Industrial flow-sheet (schematic) suggested by BARC for the beneficiation of Burugubanda tungsten-graphite ore along with the projected metallurgical material balance.

Figure 24. Flow-sheet recommended by BARC for the upgradation of wolframite pre-concentrate from Burugubanda.

+ 315 μm fraction, (iv) graphite flotation and classification of the - 315 μm fraction at 63 μm , followed by table pre-concentration of the + 63 μm fraction, and (v) pre-concentration of the - 63 μm fraction on the Mozley Multi Gravity Separator (MGS).

The material balance of the test at the end of the pre-concentration stage, is given in table 21.

Thus 52% of the tungsten values were recovered in various pre-concentrates at an average grade of 2.85% WO_3 . An additional 11% of the values were contained in various middlings at an average grade of 0.34% WO_3 .

Two routes were tried for upgrading the pre-concentrates: (i) gravimetric separation with Mozley Multi Gravity Separator after grinding the pre-concentrates to 80 μm , and (ii) electrostatic separation. Magnetic separation did not yield any appreciable upgradation mainly due to the presence of nigrine. The upgradation achieved by the two routes are given in table 22. Thus if a high grade concentrate is desired

Figure 25. Flowchart of BRGM semi pilot plant scale test on the beneficiation of Burugubanda tungsten-graphite ore.

Table 21. Results of BRGM semi pilot plant tests on Burugubanda ore (for flow-sheet see figure 25).

Pre-concentrate stream	Weight %	WO ₃ grade %	WO ₃ rec. %
+ 315 μ m spiral preconcentrate	0.63	2.33	17.28
- 315 + 63 μ m table preconct.	0.67	2.85	22.54
- 63 μ m MGS preconct.	0.26	4.10	12.35
Total pre-concentrate	1.56	2.85	52.17
Spiral middlings	2.20	0.37	9.47
Table middlings	0.48	0.25	1.41
Total gravity middlings	2.68	0.35	10.88
Spiral tailings	31.43	0.028	10.26
Table tailings	27.50	0.019	6.06
MGS tailings	14.83	0.050	8.03
Final tailings	73.36	0.029	24.39
+ 315 μ m graphite float	9.62	0.044	4.97
- 315 μ m graphite float	12.38	0.046	7.63
Combined graphite conct.	22.00	0.045	12.60
Feed	100	0.085	100

the recovery is only of the order of 15%; on the other hand at a low grade product of about 6% WO₃, a high recovery of 40 to 45% is possible. The industrial flow-sheet recommended by BRGM (BRGM-MECL Rept. 1990) is schematically shown in figure 26.

Table 22. Results of concentrate upgradation (BRGM test).

Product	Weight %	WO ₃ grade	WO ₃ rec. %
A. MGS route:			
Final concentrate	0.02	54.60	15.07
Low grade middlings	1.54	2.18	37.10
Total pre-concentrate	1.56	2.85	52.17
B. HTS route (in three stages):			
Final concentrate	0.63	5.76	42.66
Low grade middlings	0.93	0.88	9.51
Total pre-concentrate	1.56	2.85	52.17

Figure 26. Industrial flow-sheet (schematic) suggested by BRGM for the beneficiation of Burugubanda tungsten-graphite ore.

9.3 IBM test results

Indian Bureau of Mines also carried out tests on bulk scale on the beneficiation of tungsten-graphite ore from Burugubanda. The sample investigated by them (Ramanamurthy *et al* 1990), however, was of better tenor compared to the samples tested by both BARC and BRGM. The sample analyzed 0.21% WO₃ and 7% fixed carbon. After several tests, IBM arrived at an optimal flow-sheet for the beneficiation of the Burugubanda ore, keeping in view the recovery of both wolframite and graphite. Figure 27 schematically illustrates the flow-sheet suggested by them as well as the metallurgical results of the test.

While a mixture of kerosene and MIBC (0.026 kg/t each) was used during rougher flotation of graphite, MIBC (0.007 kg/t) was used during cleaner flotation stages. However the grade of graphite concentrate could not be improved even after seven regrinding and flotation stages due to fine interlocking with silicate minerals. Poor recovery of tungsten values is attributed to loss in slimes which could not be recovered. Upgradation of the concentrate also could not be achieved due to the presence of iron oxides, having similar magnetic susceptibility as wolframite.

reduction to — 6 mm only, and resorts to jigging to recover values occurring in the coarser sizes.

10. Scheelite bearing gold tailings from Kolar

The occurrence of minor tungsten in the form of scheelite in the gold ores of Kolar is well known, and the dumps of gold tailings accumulated over the years is recognized as an important resource base of this strategic material. The Balaghat and Walker's dumps are two of the important dumps, containing an estimated total resources of 720 tonnes of WO_3 (Anon 1985). The former averages 0.05% WO_3 and the latter 0.18%. Other low grade tailing dumps containing 0.01 to 0.03% WO_3 , accounting for a resource of 3300 tonnes are also reported. Extensive studies have been carried out on the recovery of scheelite from the Walker's dumps by IBM as well as BARC.

10.1 IBM studies

Indian Bureau of Mines carried out bench and pilot plant scale studies on the recovery of scheelite from the Walker's and Balaghat dumps. Based on the flow-sheet developed by IBM (figure 28) (IBM Rept. 1977), a pilot plant each to treat 50 tonnes of Walker's tailings and 200 tonnes of Balaghat tailings were set up in 1980 (Anon 1988). However the actual recoveries obtained at these plants were much below the design figures, which were attributed to the changed granulometry of the feed to the plants. Subsequently recovery operations from the Balaghat tailings were discontinued. Attempts were also made to recover values in the 'slimes' by the use of Bartles machines BMS and CBC, however these were not particularly successful (Anon 1988).

The process adopted in the Walker's pilot plant, after optimization, consists of a complex spiral-tabling gravity circuit for pre-concentration (Anon 1988). The tailings are first processed on primary spirals, and the concentrates tabled. The primary spiral tails are scavenged on secondary spirals, and the secondary spiral concentrates are tabled separately. The tails of primary and secondary tables are dewatered on a hydrocyclone, and the cyclone underflow scavenged again on primary and secondary scavenger spirals, and the respective concentrates are tabled. The middlings of the primary, secondary and scavenger tables are retabled for additional recovery. The table concentrates are mixed and processed by reverse flotation followed by magnetic separation of flotation sink to obtain scheelite concentrate as the nonmagnetic stream. The plant tailings are deslimed on hydrocyclone, and the cyclone overflow containing slimy scheelite particles are processed on Bartles machines (BMS and CBC) for additional recovery. This stream, however, produces only lean grade concentrates.

10.2 BARC studies

During mid eighties BARC investigated the reasons for the low recovery of scheelite at the BGML pilot plant and the means of improving the recovery (Shukla *et al* 1983, 1985). The samples studied by BARC assayed 0.1 to 0.14% WO_3 . Size analysis showed that 50–65% of WO_3 values are distributed in 45–50% weight of — 200 # material, and it is this high distribution in fines that poses the problem in efficient recovery of scheelite

Figure 28. IBM flow-sheet (simplified, schematic) for the recovery of scheelite from KGF gold tailing dumps.

in the flow-sheet followed at the pilot plant. The salient features of the BARC investigations are described below.

10.2a Pre-concentration by spirals: Tests were carried out on spirals with test feed prepared in different ways—direct, after desliming, and after classification. While on direct spiralling a recovery of 63.5% was obtained in the spiral concentrate, the recovery improved to 75.8% on spiralling deslimed feed, and to 83% on feed classified into + 100 and – 100 # fractions, at comparable enrichment ratios (Shukla *et al* 1983). The improvement of efficiency in spiralling is reflected in better recovery obtained in both coarse (+ 100 #) and very fine (– 400 #) size fractions.

10.2b Pre-concentration by tabling: Direct tabling on a laboratory Deister table (slime deck) gave pre-concentrates assaying 4–5% WO_3 at 65–73% recovery. The wide range of size distribution and the presence of ultrafines affected the efficiency of tabling. When deslimed feed was tabled after classification into + 100 # and – 100 # fractions, considerable improvements in both grade and recovery were realized. Results obtained in different experiments are summarized in table 23.

Table 23. Results of tabling of Walker's gold tailings after classification.

Test No.	Product	Wt. %	WO ₃ assay %	WO ₃ dist. %
1	+ 100 # table conct.	0.19	9.40	17.1
	– 100 # table conct.	0.48	11.70	54.3
	Combined table conct.	0.67	11.05	71.4
	Feed		0.10	
2	+ 100 # table conct.	0.42	5.20	18.8
	– 100 # table conct.	0.32	16.60	45.3
	Combined table conct.	0.74	10.13	64.1
	Feed		0.11	
3	+ 100 # table conct.	0.61	5.00	20.8
	– 100 # table conct.	1.30	6.70	59.6
	(Tabled after desliming)			
	Combined table conct.	1.91	6.16	80.4
4	+ 100 # table conct.	0.22	14.80	25.9
	– 100 # table conct.	0.20	20.60	34.0
	– 100 # scav. table conct.	0.21	6.60	10.9
	Combined table conct.	0.63	13.91	70.8
	Feed		0.12	
	(In this test classification was done after desliming)			

Table 24. Results of upgradation of table pre-concentrates from Walker's gold tails.

Expt. no.	Description of the experiment	Product	Wt. %	% WO ₃ assay	% WO ₃ dist.
1	Flotation of arsenopyrite	Float	42.3	0.12	0.3
	Magnetic separation of sink	Nonmagnetic	30.8	75.0	96.3
		Magnetic	26.9	3.4	3.4
	Feed (clnr. table pre-conct.)		100.0	24.0	100.0
2	Flotation of arsenopyrite	Float	39.6	0.14	0.2
	Magnetic separation of sink	Nonmagnetic	30.2	76.0	95.6
		Magnetic	30.2	3.3	4.2
	Feed (clnr. table pre-conct.)		100.0	24.6	100.0
3	Flotation of arsenopyrite	Float	49.6	1.9	5.0
	Magnetic separation of sink	Nonmagnetic	22.3	75.5	88.0
		Magnetic	28.1	4.9	7.0
	Feed (clnr. table pre-conct.)		100.0	19.2	100.0

10.2c Upgradation of the table concentrates: The main gangue minerals contained in the gravity concentrates were arsenopyrite (20–25%) and amphiboles (50–55%). Arsenopyrite was floated using amyl xanthate as collector and pine oil as frother. The flotation sink was subjected to high intensity magnetic separation. The nonmagnetic scheelite concentrate assayed + 70% WO₃ at over 90% stage recovery, starting from table concentrates assaying 20–25% WO₃ (cleaner table concentrates) (Shukla *et al* 1983). The results are given in table 24.

Figure 29. Flow-sheet of the semi pilot plant scale test 1 carried out at BARC on the recovery of scheelite from Walker's gold tailing dump (up to the pre-concentration stage), with metallurgical material balance.

10.2d Semi-pilot plant scale tests: Two alternative gravity beneficiation flow-sheets were formulated based on the batch test results, and a large scale test each was carried out following each flow-sheet. These tests were carried out up to the gravity pre-concentration stage only. The two flow-sheets are schematically shown in figures 29 and 30 along with the material balance obtained at each stage. The flow-sheet involving spiralling gave a final gravity pre-concentrate of 14.7% WO_3 at 63.7% overall recovery, while that involving tabling of classified feed gave a final pre-concentrate assaying 13.8% WO_3 at 77.6% recovery.

10.2e Alternate flow-sheets: As an alternative to all gravity flow-sheet, processes involving direct flotation of scheelite were also attempted by Shukla and co-workers (Shukla *et al* 1974, 1985). Direct flotation of scheelite using oleic acid, sodium carbonate and pine oil in different concentrations gave rougher floats analyzing 0.5 to 0.8% WO_3 at 66 to 72% recovery. Using a modified reagent combination on deslimed feed 74–76% recoveries could be obtained in floats assaying 3.8 to 5.2% WO_3 grade. Results are summarized in table 25. As an alternative to direct flotation, which did not yield high grade concentrates, tabling followed by flotation of scheelite was also attempted. The results are included in table 25. The table concentrates assayed 3 to 5% WO_3 at recoveries of 63 to 75% (76 to 87% stage recovery from deslimed feed), and flotation using a combination of oleic acid, sodium carbonate and pine oil gave a float concentrate assaying 64.5 to 69.2% WO_3 at 82 to 86% stage recovery (52 to 56% overall recovery). Direct flotation certainly offers an attractive alternative to all gravity route.

Figure 30. Flow-sheet of the semi pilot plant scale test 2 carried out at BARC on the recovery of scheelite from Walker's gold tailing dump (up to the pre-concentration stage), with metallurgical material balance.

10.3 Discussion

With near exhaustion of the high tenor Walker's gold tailing dump and the prospect of closing down of gold recovery operations at KGF in the near future, any feasibility of continuing recovery of scheelite from gold tails remains bleak. The existing flow-sheets, mainly based on gravity operations, cannot sustain economic recovery of scheelite from the leaner tenor ($< 0.1\%$ WO₃) dumps. A silver lining in this context is the study by Rao G V (1992), which has shown the feasibility of application of selective flocculation of scheelite fines as a means of pre-concentration.

11. Scheelite bearing gold ore from Hutti, Karnataka

The Hutti schist belt in the Raichur district of Karnataka, representing a part of the greenstone belt of the Dharwar craton, is well known for its gold mineralization. Association of scheelite mineralization in and around the auriferous lodes was noticed by Geological Survey of India during 1967–68. Since then a total reserve of more than a million tonnes of ore containing on an average 0.08% WO₃ have been proved or

Table 25. Results of flotation of scheelite from Walker's gold tailings.

Expt. no.	Reagent combination	Wt. % of float	WO ₃ assay %	WO ₃ dist. %	Feed grade
A. Direct flotation:					
1.	Oleic acid	8.2	1.42	55.0	0.22
2.	Oleic acid	23.4	0.67	71.5	0.18
	Sodium carbonate				
	Pine oil				
3.	-do-	14.8	0.80	70.0	0.20
4.	Reagent 801	4.2	3.80	73.5	0.21
	Sodium carbonate				
	Kerosene, pine oil (after desliming)				
5.	-do-	2.9	5.20	75.5	0.20
Expt. no.	Product	Wt. %	WO ₃ assay %	WO ₃ dist. %	Feed grade
B. Tabling followed by flotation (oleic acid + sod. carbonate + pine oil):					
1.	Table concentrate	3.0	4.0	67.7	0.18
	Float concentrate	0.15	64.5	55.6	
2.	Table concentrate	3.2	3.5	63.6	0.18
	(after desliming)-				
	Float concentrate	0.13	69.2	52.5	

indicated (Anon 1985). Scheelite is found to occur as small lenses, stringers, veinlets, streaks, blebs and disseminations along the contacts of the auriferous grey quartz veins, in the form of grains and aggregates of 20 to 50 microns.

After extensive bench and pilot plant scale tests, IBM recommended a flow-sheet, schematically shown in figure 31, giving a scheelite concentrate of 62 to 65% WO₃ at a recovery of 32% (Anon 1985). Work carried out in association of Golder Moffit Associates of U. K. showed that by use of Bartles fine gravity machines (BMS and CBC), additional scheelite could be recovered from the table tails. Tests carried out at Cornwall, in association with Bartles Company enabled formulation of a process flow-sheet involving processing the coarse stream in a table circuit (32% recovery) and treating the fines on Bartles machines. The fines recovery flow-sheet, given in figure 32, includes pre-concentration on Bartles Mozley separator, concentration on Bartles cross belt concentrator, and upgradation to a product of 74% WO₃ by alkaline pressure leaching and synthetic scheelite precipitation. This stage gives an additional 20% recovery, making a total of 52% recovery.

Work carried out by HGML in association with Lurgie GmbH, Germany, showed that it is possible to produce WO₃ concentrates of 65% grade at a recovery of 70% by a process consisting of flotation and leaching. The beneficiation process developed (Raju *et al* 1987) involves flotation of sulphides using xanthate, and scheelite by using a special reagent developed specially for Hutti ore, and cleaning the rougher scheelite float in four cleaner flotation stages. The cleaner concentrate is upgraded by acid leaching. The possibility of applying the same process to recover scheelite from the Hutti gold tailings was also investigated, which however did not yield encouraging results.

Figure 31. Gravity beneficiation flow-sheet (schematic) for the recovery of scheelite from Hutti gold ore, suggested by IBM.

It is now understood that the question of putting up a scheelite beneficiation plant at Hutti has been kept in abeyance due to the depressed condition of world tungsten concentrate market.

12. Scheelite ore from Madurai district, Tamilnadu

A sample of the ore made available by Geological Survey of India to BARC for beneficiation tests assayed 0.2% WO_3 . The mineralogical composition of the sample is given in table 26. Liberation analysis indicated that most of scheelite was liberated at a relatively coarse size of $-35\#$. Rao *et al* (1988) studied the beneficiation characteristics of this ore sample.

Results of bench scale beneficiation tests are summarized in table 27. Direct tabling of the ground ore in two stages (rougher and cleaner) gave a concentrate assaying 5.6% WO_3 at 86% recovery, which on magnetic separation gave a scheelite concentrate of 27% WO_3 . This showed the prima facie amenability of the ore to gravity beneficiation.

In the second test the ore crushed to all passing through 28# was jigged in a Denver mineral jig and the jig pre-concentrate (grade 0.5% WO_3 , 57% recovery) tabled. The

Figure 32. Flow-sheet (schematic) of the Fines gravity circuit for the recovery of scheelite from Hutti gold ore slimes, based on HGML-GMA (UK) tests.

Table 26. Mineralogical composition of Madurai scheelite ore.

Mineral	Weight %
Quartz and Felspar	58.8
Pyroxene	23.8
Garnet	13.5
Sphene	0.9
Apatite	0.5
Fluorite	2.0
Scheelite	0.3
Opagues (magnetite, cassiterite, ilmenite, pyrite etc.)	0.2
Total	100.0

table concentrate on magnetic separation yielded a high grade scheelite concentrate assaying 74.5% WO_3 with nearly 98% stage recovery. The jig tails on scavenging by table, followed by magnetic separation gave another low grade concentrate of 8% WO_3 .

Table 27. Experimental results of beneficiation of scheelite ore from Madurai, Tamilnadu.

Process	Scheelite concentrate		
	Wt. %	WO ₃ %	WO ₃ Dist. %
Direct tabling, mag. separation of table concentrate			
Feed assay 0.203 % WO ₃			
N. mag. of T. C.	0.59	26.6	77.7
Jigging, tabling of jig hutch and tails, mag. sepn. of jig hutch and table conc.			
Feed 0.223 % WO ₃			
N. mag. of jig hutch	0.166	74.5	55.7
N. mag. of table conc.	0.551	8.0	19.2
Classification at 100 #, tabling.			
Magnetic separation of table conc.			
(For flow-sheet see figure 33)			
Feed assay 0.165 % WO ₃			
N. mag. of table conc. 1 (+ 100 #)	0.105	67.3	43.2
N. mag. of table conc. 2 (+ 100 #)	0.038	37.8	8.4
N. mag. of table conc. 3 (− 100 #)	0.184	27.0	30.2

Figure 33. Flowchart of the beneficiation test carried out on the Madurai skarn-type scheelite ore (for results see table 27).

In the third test, classification followed by tabling route was followed. The scheme of the test is shown in figure 33. The results are included in table 27. At the pre-concentration stage nearly 86% recovery could be obtained, and on cleaning followed

Figure 34. Flow-sheet I (schematic) suggested for the beneficiation of Madurai scheelite ore.

by magnetic separation resulted in a combined scheelite concentrate of 41% WO_3 at a high overall recovery of nearly 82%. By optimizing the process parameters at the tabling stage (i.e. by collecting a higher grade concentrate, or by a cleaner stage) it should be possible to obtain a final concentrate of marketable grade at a minimum of 75% overall recovery.

Based on these bench scale test results two flow-sheets were suggested (Rao *et al* 1988) for large scale pilot plant scale testing, which are schematically shown in figures 34 and 35.

13. General discussion and conclusions

In order to achieve self reliance in strategic metals Rama Rao (1990) has called for an integrated 'ore to product' approach as a framework. Tungsten is one of the metals where the ore end of the situation in the country is not comfortable, while excellent know-how exists for extraction and production techniques. To make the ore to product concept successful in the case of tungsten, the urgent need is the development of techno-economically feasible process flow-sheets for the beneficiation of the reasonably good reserve potential of tungsten deposits in the country.

The foregoing description of the investigations carried out by different researchers and agencies, particularly the national laboratories like BARC, IBM, NML, RRL

Figure 35. Flow-sheet II (schematic) suggested for the beneficiation of Madurai scheelite ore.

(Bhubaneswar), etc. have amply proved the technical feasibility of beneficiation of the various potential low grade tungsten deposits of the country. But the economic viability of exploitation does not depend only on technical feasibility. Factors like tenor of the ore, minimum economic level of throughput, complexity of the beneficiation process flow-sheet, overall recovery achievable, grade and purity of the concentrates, etc play major roles in the economic viability of exploitation.

Experience in the development of flow-sheets for the beneficiation of low grade ores of strategic metals like tungsten and tin has shown the necessity of adopting the following basic strategies in order to make the process techno-economically feasible and competitive (Rao N K 1992). These are:

1. Size reduction in stages to meet the philosophy of "the earlier recovered the more recovered" and to avoid generation of "difficult to treat" slimes.
2. An effective pre-concentration step in a high throughput and efficient operation aiming at high recovery, rejecting bulk of the barren material in the initial stage at as coarse a size as possible, and to confine the mineral values to a small volume, which can be subjected to drastic processing steps cost effectively.
3. Improving the economics of operation by the concept of maximum utilization by the recovery of all potential byproducts.
4. Keeping in view a long term perspective for technology development and upgradation.
5. Aiming at an economic level of concentration with maximum recovery (rather than high grade at the cost of recovery), by a judicious combination of physical and chemical methods of beneficiation.

Table 28. Potential of Degana tungsten ores.

Ore type	Single high grade product			Two grade products			
	Average tenor, WO ₃ , kg/T	% Rec.	Recovered WO ₃ , kg/T	High grade product % Rec.	Low grade product % Rec. (with WO ₃ grade in brackets)	Total recovered WO ₃ kg/T	Additional recovery by two product approach WO ₃ , kg/T
Quartz vein	2	60	1.2	50	25(20%)	1.5	0.3
Eluvial	0.3	50	0.15	40	30(20%)	0.21	0.06
Phyllite	0.2	—	—	—	50(10%)	0.10	0.10
Granite	0.4	—	—	—	50(20%)	0.20	0.20
	1.0	30	0.3	25	40(10%)	0.65	0.35

The concept contained in item 5 above needs elaboration. There is no doubt that most of our tungsten deposits are of inferior tenor, or of low tonnage, or both, and hence a strategy different from the countries endowed with richer and larger deposits is called for. While developing beneficiation flow-sheets it is often sought to achieve what are called marketable grade concentrates. But India, which imports bulk of its requirements, is not in the run for marketing tungsten concentrates, on the other hand, what we need are 'utilizable grade concentrates' or 'concentrates of economic enrichment'. To make exploitation of low tenor ores feasible, the emphasis has to be on the production of such concentrates of economic enrichment, with emphasis on recovery, rather than marketable grade concentrates with emphasis on grade. The concept of 'forward integration' put forward by Borchers (1979) is relevant in this connection. In this strategy leaner grade concentrates of economic enrichment, which can vary from 5 to 40% WO₃ are directly utilized, or are further processed by chemical beneficiation or extraction methods to produce directly usable products like APT, tungsten oxide powder or synthetic scheelite.

This concept in most cases calls for production of two types of concentrates, a high grade product of what is called marketable grade concentrate that is feasible by a simple process of upgradation, and a low or medium grade product containing all the rest recoverable values. The advantage of this approach of maximizing recovery is illustrated by taking the example of Degana ore deposits. Table 28 illustrates the feasible recoveries under the two approaches.

Economically exploitable reserves of the quartz-vein type tungsten ore in Degana, which is amenable for beneficiation to produce marketable grade concentrates at high recoveries, are nearly exhausted. Continuing underground mining has become economically unsustainable in the present depressed world tungsten scenario. Continuation of Degana in the tungsten scene will depend on large scale exploitation of richer portions of the fertile granite as a whole (or the so called trench lode granite) minable by open cast methods. It is heartening to note that Hindustan Zinc Ltd., who have now taken over the Degana tungsten property, are in the process of preparation of feasibility report on this line of activity (Wadhavan 1992).

Though marketable grade concentrates of scheelite can be produced at reasonably good recoveries from the Khobna deposit, maximizing recovery of the strategic metal

Table 29. Tungsten deposits—Present status, prospects and problem areas.

Deposit	Present status	Future prospects	Areas where additional R & D studies are needed
1. Degana	Exploitation of the meagre resources of quartz-vein type by underground mining has become uneconomical. Other types, except granite, are of too low tenor for economic exploitation.	Future of the deposit depends on exploitation of granite-type ore. Economics of W recovery will depend on extent of minable ore of average grade $> 0.1\%$ WO_3 and on maximizing byproduct recovery and utilization.	<ol style="list-style-type: none"> 1. Best means of pre-concentration with maximum recovery. 2. Chemical upgradation of low and medium grade concentrates. 3. Recovery of Li from low grade high volume concentrates of lithiferous mica.
2. Khobna	Mainly scheelite ore, amenable for beneficiation with reasonably high recovery.	Present proved reserve position may not sustain an exploitation throughput of economic-scale level.	<ol style="list-style-type: none"> 1. Utilization/upgradation of off-grade products. 2. Maximizing the recovery of potential byproducts like Cu, Mo and Ag which may improve the economics of exploitation.
3. Burugubanda-Tapaskonda	Low tenor ore. Beneficiation is somewhat problematic; only low to medium grade concentrates are feasible at reasonable recoveries.	Economics of exploitation linked to graphite recovery.	<ol style="list-style-type: none"> 1. Chemical upgradation of low to medium grade products. 2. Effective means of separation of garnet from wolframite in fine sizes. 3. Improving grade of graphite product by physical methods.

tungsten calls for the production of a low to medium grade product too, as already explained. As far as the Burugubanda–Tapaskonda deposit is concerned, there is no alternative to production of a low to medium grade wolframite concentrate, if reasonably good recoveries are desired. In order to optimally utilize all these low to medium grade products, it is necessary to develop indigenous technology for their chemical upgradation into directly usable products. A good beginning has been made in this direction in some of the recent publications on the subject (Paramaguru *et al* 1987; NML Rept. 1990; Srinivasan *et al* 1991; Subramanian *et al* 1992). Results obtained in the NML study (Premchand 1993), following the pressure leaching with soda followed by solvent extraction, are highly encouraging. A technology which is applicable for both wolframite and scheelite concentrates and their mixtures, however, will be most appropriate and desirable in the context of optimizing the utilization and conservation of scarce tungsten resources of the country.

In the present knowledge of indigenous resources position of tungsten, the most promising deposits are only three, namely, Degana in Rajasthan, Khobna in Maharashtra and Burugubanda–Tapaskonda in Andhra Pradesh. Table 29 sums up the present status, future prospects and problem areas where more R & D inputs are needed to make the exploitation of these deposits economical and more attractive.

Acknowledgements

The author wishes to thank Dr C K Gupta, Director, Materials Group, BARC for his sustained interest and encouragement. He wishes to place on record his deep sense of appreciation for the help rendered by all his colleagues in the Ore Dressing Section, BARC, particularly Mr S K Shukla, Mr T Sreenivas, Mr N P H Padmanabhan, Mr D Narasimhan and Mr T S Sunilkumar.

References

- Anon 1979 *Mining Mag.* 28
- Anon 1985 *Tungsten mineral commodity survey, Report of sub-committee on strategic minerals (Group XII)* (Nagpur: Geological Survey of India) pp 70–78
- Anon 1988 *Tungsten mineral commodity survey, Report of sub-committee on strategic minerals (Group XII)* (Nagpur: Geological Survey of India) pp. 1–15
- Anon 1988 Minutes of the second meeting of the Project Monitoring Committee on Tungsten Development sponsored by DRDO/DMRL – Report of the specialist committee.
- Anon 1988 *Beneficiation of tungsten ores, Report of the sub-committee on strategic minerals (Group XII)*, (Nagpur: Geological Survey of India) pp. 54–57
- Anon 1991 *Metal News* 13 25
- Auge P, Bahr A and Koser H 1975 *Proc. 11th International mineral processing congress* (Cagliari: Univ. of Cagliari) pp 691–712
- Babok V K and Viduetskii M B 1967 USSR Patent No. 202, 020.
- Baldin S A, Viktorov M N and Tereshkov V N 1984 *Tsvetn. Met.* 1984 88
- Banerjee S K and Narayanan P I A 1952 NML Internal Report No. 11/52
- Bansal R K, Lahiri A K and Bahree R C 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development* (Bhubaneswar) pp. 77–86
- Bhattacharjee J, Chattopadhyay B, Mukhopadhyay and Sweta Chattopadhyay 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development* (Bhubaneswar) pp. 55–60
- Bhoskar K G, Mohan M, Jain S S, Mohanty A K and Saha A K 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*. RRL (Bhubaneswar), DMRL (Hyderabad) pp. 21–32

- Borchers P 1979 *Tungsten, Proc. 1st international tungsten symp.*, (Stockholm, London: Mining Journal Books) pp. 64–77
- BRGM-MECL Tin Tungsten Collaborative Programme – Burugubanda – Tapaskonda Tungsten Project, 1990 Pre-feasibility Report on economic recoverability of tungsten from the Burugubanda deposit, D-2 Stage (Unpublished)
- BRGM-MECL Tin Tungsten Collaborative Programme-Khobna Tungsten Project, 1991 Pre-feasibility Report on Khobna Tungsten Deposit, D-2 Stage, Vol. 1, Chapter: Beneficiation, pp. 1–41 (Unpublished)
- Chatterjee A K, Negi M S, Rao K S and Singh L 1987 in *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 47–54
- Datta P S 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 61–70
- Dawei W, Keron W and Jicun Q 1986 *IJMP* 7 361–71
- Dwivedy K K 1988 *Beneficiation of tungsten ores; Report of the sub-committee on strategic minerals (Group XII)*, (Nagpur: Geological Survey of India) pp. 58–63
- Ermolenko V I, Yur'ev V I, Nesterov V G, Berdichevskii B I and Krasivina L E 1985 *Tsvetn. Met.* **1985** 85–88
- Gak T L, Bayshulakov A A and Gucalyuk T G 1983 *Izv. Vyssh. Ucheb. Zaved. Cvet. Metal* **1983** 16
- Ghosh C, Pai D R, Narasimhan J B and Majumdar K K 1968 BARC Internal Report BARC/Met/12–1968
- Han K N and Say W C 1985 *Chem. Abstr.* **103** 40343r
- Indian Bureau of Mines 1977 Pilot plant studies at site on recovery of scheelite from Walker' tailing dump, Kolar Gold Fields IBM/R. I. No. 357
- Jarrett R G and Warren L J 1977 *Proc. Australasian Inst. Min. Metall.* **262** 57–65
- Karahan S, Demirci A and Atamedir R 1980 *World Mining* **33** 46–51
- Kelsall G H and Pitt J L 1987 *Chem. Engg. Sci.* **42** 679
- Koh P, Andrews J and Uhlhen P 1986 *IJMP* **17** 45
- Kunwar R K, Chakrabarti D M and Mathur G P 1972 NML Internal Report No. 668/72
- Lahiri A K and Patni S K 1988 *Beneficiation of tungsten ores, Report of the sub-committee on strategic minerals (Group XII)*, (Nagpur: Geological Survey of India) pp. 121–125
- Minnonov I I, Ermolenko V I, Yur'ev V I, Zvrev V V and Litvintsev E G 1987 *Tsvetn. Met.* **1987** 99–101
- Mitchell W Jr, Sollengerger C L and Kirkland T B 1951 *Trans. AIME* **190** pp. 60–64
- Narasimhan D 1989 *Petrographic Report on Khobna-II ore from Khobna, Maharashtra: Annexure to Laboratory studies on the beneficiation of tungsten ore from Khobna, Maharashtra, by Shukla et al Report submitted by ODS, BARC to MECL (Unpublished)*
- Narasimhan J B, Degalesan S N and Majumdar K K 1972 BARC Internal Report, BARC/I-200/1972
- Narasimhan K S 1988 *Beneficiation of tungsten ores; Report of the sub-committee on strategic minerals (Group XII)*, (Nagpur: Geological Survey of India) pp. 110–120
- National Metallurgical Laboratory, Mineral Beneficiation Group, 1990 Chemical beneficiation studies: Interim Reports 4 and 5 on NML Thrust Area Project on Beneficiation and purification of tungsten ores of India, sponsored by DRDO/DMRL, April 1990 and Oct. 1990
- National Metallurgical Laboratory 1992 Report on physical beneficiation of tungsten ore and pre-concentrate samples from Degana Rajasthan (Unpublished Report) pp. 1–94
- Ottley J D 1979 *Minerals Sci. Eng.* **11** 99–121
- Padmanabhan N P H, Rambabu Ch, Das K K, Rao G V U and Pradip 1985 *Progress in metallurgical research: fundamental and applied aspects* (eds) S P Mehrotra and T R Ramachandran (Kanpur: IIT) pp. 39–44
- Padmanabhan N P H, Rambabu Ch, Pradip Das K K, Venkitakrishnan R R, Narasimhan D and Rao G V U 1984 *Beneficiation of wolframite ore from Degana, Rajasthan*, Report submitted by ODS, BARC to RSTDC, Rajasthan (Unpublished)
- Padmanabhan N P H, Rao M M, Sivaramakrishnan K, Sunil Kumar T S and Rao N K 1991 *Studies on beneficiation of wolframite-graphite ore from Burugubanda, Andhra Pradesh, Part I. Beneficiation studies: Report submitted by ODS, BARC to MECL (Unpublished)*
- Padmanabhan N P H, Sreenivas T and Rao N K 1990 *High Temp. Mater. & Process.* **9** 217
- Paramaguru R K, Jena K N and Sahoo P K 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 117–122
- Patni S K, Lahiri A K and Bahree R C 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development* RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 41–46
- Premchand 1993 *Processing of low grade tungsten ore concentrates by hydrometallurgical route – technology and economic issues with particular reference to India*. Preprint Copy, pp. 1–35

- Raju K K, Krishnan R, Gururaj Naik J S and Mahanti S C 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 109–116
- Ramanamurthy V V, Dhargave R G, Doifode K T, Barthwal Y M, Kapale K Y, Sutaone A T, Ruikar M R and Rao G M 1990 *Bulk testing of tungsten-graphite sample from Burugubanda Area, Andhra Pradesh* Report of Investigation No. IBM/R.I. Submitted by IBM to MECL (Unpublished)
- Rama Rao P 1990 *Metal News* 12 2
- Rao G M 1993 *Miner. & Metals Rev.* 13
- Rao G M and Satyanarayana K 1987 *Tungsten resources development, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 87–90
- Rao G M and Subrahmanyam N N 1988 *Beneficiation of tungsten ores: Report of the sub-committee on strategic minerals (Group XII)* (Nagpur: Geological Survey of India) pp. 34–53
- Rao G V 1992 *Trans. Indian Inst. Metals* 45 275–281
- Rao M M, Padmanabhan N P H and Rao N K 1992 *Proc. national seminar on gravity concentration*, RRL, Bhubaneswar, Preprint Vol. pp. 217–226
- Rao M M, Shukla S K, Narasimhan D, Iyer N V and Rao N K 1988 *Recovery of scheelite from tungsten ore from Karungalakudi, Madurai District, Tamilnadu*: Report submitted by ODS, BARC to GSI, Economic Geology Division, Tamilnadu (unpublished)
- Rao N K 1991 *Proc. Indo-US workshop on advances in chemical metallurgy*, Bombay (eds) C K Gupta, M K Totlani and N Birks (New Delhi: DST and Washington: NSF) pp. 115–122
- Rao N K 1992 *Exploration of low grade ores of strategic metals in India—Role of the ore dressing strategy: Platinum Jubilee Lecture delivered at the 79th Session of Indian Science Congress, Baroda*
- Save M and Conil P 1990 *BRGM-Mecl Tin-Tungsten collaborative programme, Burugubanda Tungsten Project Phase 2, Semi pilot plant beneficiation of Burugubanda (India) graphite tungsten ore* Report submitted by BRGM to MECL (Unpublished)
- Sehgal M N and Suryanarayana M 1987 *Tungsten resources development, 1987, Proc. national workshop on tungsten resources development*, RRL (Bhubaneswar) and DMRL (Hyderabad) pp. 1–12
- Shao Varui and Shi Daxin 1986 *Chem. Abstr.* 105 156514t
- Shukla S K, Kumar K V, Majumdar K K and Degaleesan S N 1974 *Studies on the beneficiation of scheelite from Kolar gold tailings* Report submitted by ODS, BARC to BGML (Unpublished)
- Shukla S K, Natarajan R, Das K K, Venkitakrishnan R R, Subrahmanyam N P and Rao G V U 1983 *Laboratory investigations on improving the recovery of scheelite from gold tails* Report submitted by ODS, BARC to BGML (Unpublished)
- Shukla S K, Natarajan R, Das K K, Venkitakrishnan R R, Subrahmanyam N P and Rao N K 1985 *Proc. national seminar on mineral processing and IX Annual convention of IIME., Jamshedpur*, Preprint Vol. pp. 259–271
- Shukla S K, Padmanabhan N P H, Rao M M, Subrahmanyam N P, Venkitakrishnan R R and Rao N K 1985 *Beneficiation of phyllite-type low grade wolframite ore from Degana, Rajasthan* Report submitted by ODS, BARC to RSTDC, Rajasthan (Unpublished)
- Shukla S K, Rao M M, Subrahmanyam N P, Iyer N V and Rao N K 1988 BARC External Report BARC/1408
- Shukla S K, Rao M M, Venkitakrishnan R R and Rao N K 1989 *Laboratory studies on the beneficiation of tungsten ore from Khobna, Maharashtra* Report submitted by ODS, BARC to MECL (Unpublished)
- Shukla S K, Rao M M, Venkitakrishnan R R, Murthy P S, Narasimhan D and Rao N K 1991 *Beneficiation studies on Khobna tungsten ore samples – Semi pilot plant scale tests*; Report submitted by ODS, BARC to MECL (Unpublished)
- Shukla S K and Rao N K *Beneficiation of tungsten ores* (Geological Survey of India (Under publication))
- Srinivasan G N, Murthy B V R, Sahu B N, Jouhari A K, Dey D N and Jena P K 1991 *Trans. Indian Inst. Metals* 44 167
- Stafford P T 1985 *Minerals, facts and problems* (Washington: U.S. Dept. Interior) p. 881
- Subramanian C, Suri A K, and Gupta C K 1992 *Trans. Indian Inst. Metals* 45 207–213
- Sun Shanlun, Chen Quiren and Yang Yue 1984 *Proc. miner. proc. extr. metal conference* (eds) M J Jones and P Gill (Kunming, China: Inst. Min. Metal and Chinese Society of Metals) p. 521
- Sunil Kumar T S and Rao N K 1991 *Studies on beneficiation of wolframite-graphite ore from Burugubanda, Andhra Pradesh, Part II. Petrographic studies*; Report submitted by ODS, BARC to MECL (Unpublished)
- Svoboda J 1988 *Magnetic methods for the treatment of minerals: developments in minerals processing* (ed.) D W Fuerstenau (Amsterdam: Elsevier) Vol. 8, p. 538

- Tata Research Development and Design Centre, Pune, 1991 Final progress report on development of selective flocculation technology for recovery of tungsten values from slimes (Vol. I), Submitted to Dept. Mines, Government of India, New Delhi
- Texeira O C G, Oliveira J F and Villas Boas R C 1988 *Developments in mineral processing: Froth flotation* (eds) S H Castro and J Alvarez (Amsterdam: Elsevier) pp. 175–184
- Vasquez L A, Ramachandran S and Grauerholz N L 1976 *Flotation – A. M. Gaudin Memorial Volume* (ed.) M C Fuerstenau, pp. 580–596
- Wadhavan A C 1992 *Trans. Indian Inst. Metals* **45** 347
- Warren L J 1975a *Trans. IMM* **84** C99–104
- Warren L J 1975b *J. Colloid Interface Sci.* **50** 307
- Weisun W 1982 *Tungsten: 1982, Proc. 2nd int. tungsten symp.* (San Francisco, London: Mining Journal Books) pp. 64–70
- Zhaboev M N, Semochin G A, Blinor Yu I, Dzhamoev F M, Novikov V V and Tereshchenko S V 1987 *Tsvetn. Met.* **1987** 92